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MAY, 1914



Bulletin of the American Institute of Mining Engineers

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4-7 ACTUAL SIZE

7,000 TONS OF ORE
OF WHICH THE ABOVE IS AN AVERAGE SAMPLE,
HAVE BEEN FED TO AN
8 FT. HARDING PEBBLE MILL
AND SUCCESSFULLY GROUND

OF COURSE THIS IS COMPARATIVELY SOFT. NO
PEBBLE MILL COULD SUCCESSFULLY HANDLE $1\frac{1}{4}$ IN.
PIECES OF EVEN ORDINARILY HARD ORE; HOWEVER,
A HARDING MILL CAN, AND SEVERAL ARE TAKING
THE TOTAL PRODUCT OF STAMPS FITTED WITH
SCREENS AS LARGE AS 1 IN. X 2 MESH.

HARDING CONICAL MILL CO.,

LONDON:
SALISBURY HOUSE

50 CHURCH ST.,
NEW YORK

Cable Address: Halharding, New York

AMERICAN INSTITUTE OF MINING ENGINEERS
29 West 39th Street, New York, N. Y.

PROPOSAL FOR MEMBERSHIP

Mr. _____
(Name in Full)

Occupation _____

Address _____

is hereby proposed by the undersigned, as a _____

of the American Institute of Mining Engineers.

} Signatures of three
Members or
Associates.

Place of birth _____ *Year of birth* _____

*Education, general and technical, when, where and how acquired,
with degrees, if any.*

<i>Dates</i>	
_____	_____
_____	_____
_____	_____
_____	_____
_____	_____
_____	_____
_____	_____

[illegible]

_____ Signature

Dated _____ *191* _____

EXTRACTS FROM THE CONSTITUTION.

ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

No. 89

MAY

1914

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTKEY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STOUTON, Editor, F. O. PIERCE, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

SALT LAKE CITY MEETING

The One Hundred and Eighth Meeting of the Institute, for the presentation and discussion of technical papers, will be held at Salt Lake City, Utah, August 10 to 14, 1914.

ALL PAPERS TO BE PRESENTED AT THIS MEETING MUST BE IN THE HANDS OF THE SECRETARY ON OR BEFORE MAY 31, 1914.

1913 BULLETINS WANTED

The Institute is out of *Bulletins* for the year 1913 for the following months:

APRIL, JUNE, JULY, AUGUST.

Demands are made on this office for copies of these *Bulletins* and the Institute would be glad to buy them at 50 cents each, in good condition. Those willing to sell the *Bulletins* are asked to communicate with this office.

BOARD OF DIRECTORS

Meeting of Mar. 27, 1914.—President Thayer announced the appointment of H. S. Munroe as a member and E. Gybbon Spilsbury as Chairman of the Library Committee of the Institute.

The President announced the appointment of the Committee on Papers and Publications, the Technical Committees, the Committee on Increase of Membership, and the Committee on Junior Members and Affiliated Student Societies.

The President announced the appointment of Dr. George F. Kunz as a member of the Reception Committee of the International Engineering Congress, 1915.

It was voted that the Institute accept the invitation of the *Mining and Scientific Press* to occupy space at the Panama-Pacific Exposition next to them and avail itself of the service of their representative to receive and register members of the Institute who visit this space during the exposition.

The sum of \$125 was appropriated to the St. Louis Local Section.

The President was authorized to appoint delegates to attend the meeting of the Advisory Committee in connection with the Bureau of Standards in Washington, on Tuesday, Apr. 14, 1914.

119 Members, two Associates, and six Junior Members were elected to membership in their respective classes. The status of one Junior was changed to that of Member.

The resignations of 19 Members were accepted.

The deaths of 7 Members were reported.

The date of the San Francisco meeting of the Institute was placed for Sept. 27-30, 1915.

It was voted that no arrangements should be made by the Institute for a special train from the East to the Salt Lake City meeting.

An Executive Committee of the Committee on Papers and Publications was appointed, as follows: Joseph W. Richards, George C. Stone, James F. Kemp, Thomas T. Read, and Bradley Stoughton.

The sum of \$175 was appropriated to the Colorado Local Section for the year 1914.

TRANSACTIONS FOR MEXICO

Many copies of Volume XLIV, which were sent to members of the Institute residing in Mexico, were returned to this office with a statement that they could not be delivered. Unless word is sent to us to the contrary, we shall not attempt to deliver copies of the *Transactions* to these same members until conditions in Mexico warrant a reasonable hope of their reaching their destination, but all such volumes of *Transactions* will be held at the office of the Institute and delivered to members upon receipt of their order, at any future time.

HADFIELD RESEARCH PRIZE

Through the generosity of Sir Robert Hadfield, Honorary Member, the Directors of the American Institute of Mining Engineers are enabled to announce the Hadfield Research Prize of \$1,000 for the best contribution to the publications of the Institute upon the general subject of The Different Forms and Combinations of Carbon with Iron, including those in Iron Alloys. The prize will be awarded at the Annual Meeting of the Institute in February, 1916, to the best paper upon this subject which is presented to the Institute before Nov. 1, 1915, provided such paper is deemed worthy by the Institute's Iron and Steel Committee.

The following suggestions are also added for the guidance of those who may take part in this research. While it is not desired to define closely the exact lines or scope of the proposed research, as it is advisable to make these as broad as possible, the object in mind may be said to be generally as follows:

To elaborate and find out the best methods of determining the forms of carbon in steel or iron, including those in iron alloys. A portion of the work would probably be a continuation of the researches which have in the past been carried out by Jullien, Abel, Muller, Ledebur, T. Sterry Hunt, Akerman, Arnold, Stead, E. D. Campbell, Hogg, Parry, Upton, and others.

In a generic way, metallurgists now speak of carbides, sub-carbides, double carbides, special carbides, and other combinations. It is very desirable that these should be accurately defined and understood.

It is also desirable to know whether there are other or new forms; if so, can these be separated and their characteristics obtained.

It may be interesting to point out that the carbon compounds now definitely known are stated to number over 80,000. It is very probable, therefore, that there is room for much valuable and useful research to be carried out with the object of increasing our knowledge of the various combinations of carbon with iron, as probably some of these are still unknown.

In addition to reasearch work upon particular forms of carbide which have not yet been determined, it is also desirable and necessary to determine the state in which the carbon exists. For example, there exists what is termed a "missing form" of carbon, about which little is known or understood. More light is required about this form, as for many years very little has been added to our knowledge on this subject. It would be desirable, for example, to know whether the carbon not accounted for as carbide is "missed" in consequence of its being in so fine a state of division, or whether it is present in some special form or condition.

It may be mentioned as a general statement that when steel is in the austenitic condition it is softer than when transformed to the martensitic formation. In the former, the carbon is considered to be in complete solution; yet steel showing martensitic structure is said to contain its carbon in complete solution also. If it could be shown that the martensitic formation results from the commencement of the falling out of solution of the carbon, this would be of great assistance to all those who are desirous to have increased knowledge in this direction. It is therefore desirable to know exactly in what state the carbon exists in the austenitic and martensitic formations.

It is also necessary, if possible, to ascertain the molecular constitution

of the carbides. Such a point has been raised by the able American scientist Prof. E. D. Campbell, and much important research work has been carried out by him with regard to certain particular combinations or forms of carbide. In other words, is the ordinary carbide Fe_3C , Fe_3C_2 , or is it some other combination? If so, what is its nature and molecular constitution?

The above seems to be an outline of the general direction which should guide those considering and taking part in this research. It is hoped that the results obtained will throw much light on the cause of the hardness of steel, also the nature and form of carbon combinations with iron and its alloys.

No special conditions are imposed as to the length of competing manuscripts (although diffuseness and repetition are always to be avoided), nor as to the number of investigators who may work together. No preference will be shown to Americans over investigators of any other nationality, nor to members of the Institute, nor to manuscripts in English over those in another language, but all will be judged upon the same terms. All manuscripts must be typewritten.

In order to facilitate the researches, every reasonable assistance will be given by the Institute Library and editorial staff, in making available the results of previous researches upon this and allied subjects. A bibliography of the subject will shortly be published in the *Bulletin* of the Institute, and a copy of this will be furnished gratis to those who expect to compete for the prize. As far as possible, copies of rare publications will be loaned from the Institute Library, and these loans will be made strictly in order of application and for a limited time only. In addition to these free services, there are special services which the Library is accustomed to render to members and non-members for a moderate fee, which may prove helpful to competitors. These include making abstracts, special bibliographies, translations, photoprint copies, etc.

Further information may be obtained from the undersigned,

BRADLEY STOUGHTON, *Secretary*,
AMERICAN INSTITUTE OF MINING ENGINEERS,
29 West 39th St., New York, N. Y., U. S. A.

VOTE OF THANKS TO SIR ROBERT HADFIELD

At a meeting of the Board of Directors of the Institute, held on Friday, Apr. 24, 1914 it was moved by Dr. Albert R. Ledoux, seconded by Prof. Joseph W. Richards, and unanimously voted, to extend a cordial vote of thanks on behalf of the Institute to Sir Robert Hadfield for his very generous gift establishing the Hadfield Research Prize, and for his thus initiating a movement which we have reason to believe will be followed by some of the Technical Committees of the Institute; and it was further resolved that this vote of thanks be published in the *Bulletin* in connection with an announcement of the prize.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period April 1 to 15.

William B. McKinley, New York, N. Y.
C. W. Goodale, Butte, Mont.
D. C. Brown, San Luis Potosi, Mex.
W. H. Rettie, Yonkers, N. Y.
Mark B. Kerr, San Francisco, Cal.
H. W. Turner, San Francisco, Cal.
Frank M. Leland, Mackay, Idaho.
Sir Robert Hadfield, London, Eng.

Collier Cobb, Chapel Hill, N. C.
E. P. Fleming, Guayacan, Chile.
Albert Sauveur, Cambridge, Mass.
H. M. La Follette, La Follette, Tenn.
E. Fleming L'Engle, New York, N. Y.
H. Garlichs, St. Louis, Mo.
Henry Hay, London, England.

Edward J. Way announces that his agreement as consulting engineer in South Africa for the Anglo-French Exploration Co., Ltd., having terminated, he will engage in professional consulting work with headquarters at 135 Cullinan Buildings, Simmonds St., Johannesburg, South Africa.

W. L. Morrison has opened an office at Erie, Pa., and will engage in consulting work, specializing in the design and installation of electric furnaces and electrical equipment.

James C. H. Ferguson, for the past 13 years Pacific Coast Sales Agent of the Midvale Steel Co., of Philadelphia, has severed his connection with that company and is now the Pacific Coast Representative of the William Cramp & Sons Ship & Engine Building Co., of Philadelphia, with his office in the Monadnock Building, San Francisco.

Edward H. Coxe has been appointed President and General Manager of the Ideal Block Coal Co., Lily, Ky.

A. E. Borie has been elected Chairman of the Board of Directors; **Knox Taylor**, President; and **Henry M. Howe** and **V. Angerer** Vice-Presidents of the Taylor-Wharton Iron & Steel Co., High Bridge, N. J.

Richard Peters, Jr., has accepted a position with the Producers Coke Co., Uniontown, Pa.

E. L. Estabrook has accepted an engagement with **F. G. Clapp** to examine oil properties in China.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Position open as assistant superintendent at a steel plant. Practical experience in the manufacture of crucible and open-hearth steel and heat treatment of steel and alloy steels required. This position offers an excellent future. No. 9.

A company in Virginia desires a man with experience in Western mines to act as superintendent of its pyrite mine. No. 15.

Position open as instructor at a University in Middle West to teach the mechanical side of mining, such as hoisting, haulage, drilling, and mine surveying. Salary, \$1,500 to \$1,800. No. 16.

Experienced amalgamator wanted at a mill in Ivory Coast, Africa. An American speaking French preferred. No. 18.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Metallurgist of 18 years' experience in Mexico with large plants is open for engagement. Last position, superintendent of copper smelting works. Best line, copper and lead smelting, but also experience in mine valuation and ore treatment investigation work. Speaks Spanish. No. 87.

Mining engineer, aged 27, desires permanent position. Six years' experience in mine and development work in bituminous coal field. Was acting superintendent of a large mine past winter. No. 88.

Member, specializing in the treatment of refractory gold and silver ores, desires position as metallurgist. No. 91.

Member, aged 25, technical graduate, with experience as engineer and assistant superintendent of coal and lead mines, familiar with concrete construction, desires position as engineer or assistant mine superintendent. No. 92.

Graduate mining engineer, aged 24, with one year's graduate work in the metallography of iron and steel, will be available about June 1 for a position in a steel or manufacturing plant. No. 94.

Superintendent with 15 years' experience as mine surveyor, foreman, assayer, superintendent and manager, in the development and operation of gold, silver, and copper mines, open for engagement as superintendent, engineer and assayer, or assistant superintendent. Good knowledge of Spanish and 10 years' Mexican experience. No. 95.

Member, technical graduate, 36 years old, married, 10 years' experience as engineer and superintendent in Mexico and four years' as superintendent in United States, desires position as engineer or superintendent. No. 96.

THE INSTITUTE FORUM**WHAT DOES "MAGNETIC" MEAN?**

The word magnetic has several meanings. When used, as it usually is, without qualification one is often unable to tell which meaning is intended.

1. A body is magnetic which has power to attract iron magnetically. Such is a lodestone or magnet, either permanent or not. This is an active sense or meaning in that the body itself does the attracting or is magnetic.

2. A body is magnetic which may be made to attract iron magnetically, as the iron core of an electro-magnet when not magnetized by the current. This is a potentially active meaning, the body being capable of becoming a magnet.

3. A body is magnetic which has the property of being attracted by a magnet magnetically. Such are iron, cobalt, nickel, magnetic oxide of iron, magnetic alloys, etc. This is a passive meaning, the attracting being done by the actively magnetic body.

With these three meanings to choose from, one must usually make a guess based on previous knowledge as to which is the right one. Would it

not be desirable if the kind of magnetism in mind were to be designated as active, potentially active, or passive, as the case may be, or in some better way, to avoid creating doubt? As now used the word is applied indiscriminately to anything and everything which is magnetic in any of the three ways.

If the iron core of any electro-magnet be claimed to be magnetic because of some residual magnetism it may possess it would have to be called feebly magnetic, while potentially it is the most magnetic substance known.

What can our electrical members do to clear the situation?

HENRY D. HIBBARD.

SOUTHERN CALIFORNIA LOCAL SECTION

Executive Committee

THEODORE B. COMSTOCK, *Chairman*

SEELEY W. MUDD, *Vice-Chairman*

FREDERICK J. H. MERRILL, *Secretary-Treasurer*,
300 Germain Building, Los Angeles, Cal.

C. COLCOCK JONES

FRANK ROBBINS

The February meeting of the Southern California Local Section was held at the headquarters in the Chamber of Mines and Oil of Los Angeles on Feb. 17, C. Colcock Jones presiding. Fourteen members were present. The subject announced was Ore Sampling. W. M. Watts opened the discussion by a general outline of the important points to be covered in sampling a mine and the methods to be used in the various stages of the work. All the members present successively contributed details from their personal experience.

The March meeting of the Section was held at its headquarters on Mar. 15. Twelve members were present, with Dr. Theodore B. Comstock in the Chair. The subject announced was Secondary Enrichment and all present took part in the discussion, Frank Robbins contributing a short paper on some observations in Central America.

The next meeting will be held on Apr. 14, at which time the subject of Mine Accounting will be discussed.

F. J. H. MERRILL, *Secretary*.

ENGINEERING CONGRESSES AT SAN FRANCISCO, 1915

INTERNATIONAL ELECTROTECHNICAL COMMISSION
During week beginning September 6, 1915.

INTERNATIONAL ELECTRICAL CONGRESS
During week beginning September 13, 1915.

INTERNATIONAL ENGINEERING CONGRESS
During week beginning September 20, 1915.

INTERNATIONAL GAS CONGRESS
During week beginning September 27, 1915.

INTERNATIONAL ELECTRICAL CONGRESS, 1915

In offering you this opportunity to become a member of the International Electrical Congress, San Francisco, 1915, the Committee on Organization solicits your interest and support as well as your membership.

There have been ten electrical congresses of international scope, three of which have been held in America.

The forthcoming Congress is to be held in conjunction with the Panama-Pacific International Exposition, San Francisco, 1915. The American delegates to the most recent Congress, that of Turin in 1911, extended an invitation to the Congress to meet in San Francisco in 1915. This invitation was accepted, the necessary authority was granted by the International Electrotechnical Commission, and under the auspices of the American Institute of Electrical Engineers plans for the Congress have been effected and the organization work is being prosecuted vigorously.

It is the desire of the Committee on Organization that engineers the world over shall promote and participate in the success of this Congress. To that end the Committee pledges its utmost effort in the organization of the Congress and in making its valuable transactions available to members upon the most attractive terms practicable, and the Committee calls upon the electrical engineering profession for assistance and support in its undertaking.

Manuscripts will be received and papers will be printed in English, French, German, Spanish, and Italian. All papers printed in languages other than English will be accompanied by English abstracts.

Invitations are now being issued to engineers in all civilized countries to prepare papers for the Congress, and the Committee on Program will be glad to consider any papers which may be offered, acceptance of such papers, however, being contingent upon the usual considerations and upon the space limitations of the transactions.

The Committee on Organization extends to all engineers who are affiliated with engineering and scientific societies of standing, an invitation to apply for membership in the International Electrical Congress.

The membership fee is \$5 (New York Exchange), which entitles the member to the privileges of the Congress, to the general index, and to a choice of any four of the Section volumes of the Transactions. By payment of \$1 the volumes ordered will be delivered without further expense to the member, and by additional payment of 50 cents a volume they will be bound in cloth. Additional volumes may be had at \$1 each, bound in paper, or at \$1.50 each, bound in cloth, and the entire set of (probably) thirteen volumes will be delivered for \$11 if bound in paper, or for \$16 if bound in cloth. (Membership fee included.) Upon election to membership and receipt of payment, a membership card will be issued bearing upon its reverse side receipt for payment and statement of the volumes of transactions which have been chosen by the member.

The meetings of the 1915 Congress will be divided into twelve Sections and there will be (probably) thirteen volumes in the complete set of its transactions, as follows:

1. **Generation, Transmission, and Distribution.**—Central station and substation design, control and operation. Long-distance transmission of electric power.

2. **Apparatus Design.**—Generators, motors, and transformers. The rating of machinery.
 3. **Electric Traction and Transportation.**—City, surface, and rapid transit railways; interurban and trunk lines; electric vehicles, ship propulsion, mining railways, elevators, and hoists.
 4. **Electric Power for Industrial and Domestic Use.**—Factories, mills, refrigeration, heating devices, etc.
 5. **Lighting and Illumination.**—Arc and incandescent lighting; the science and art of illumination.
 6. **Protective Devices; Transients.**—Switches, circuit breakers; condensers; electrostatics; disruptive phenomena; high-frequency phenomena.
 7. **Electrochemistry and Electrometallurgy.**—Electrolytic and metallurgical apparatus and processes.
 8. **Telegraphy and Telephony.**—(a) All communication of intelligence by the use of wires. (b) Electromagnetic waves and radiotelegraphy and telephony.
 9. **Electrical Instruments and Electrical Measurements.**—Switchboard, portable, standard, and absolute instruments. Testing and standardization methods; absolute measurements.
 10. **Central Station Economics.**—Load factors, power factors, measurement of maximum demand and all problems affecting the economy of central stations; also rates, their regulation and legislation.
 11. **Electrophysics.**—Radioactivity; Röntgen rays; gas and vapor conduction; electron theory; constitution of matter, etc.
 12. **Miscellaneous.**—Such as history and literature of electrical engineering; symbols and nomenclature; engineering education and ethics.
 13. **General Congress Proceedings.**
-

INTERNATIONAL ENGINEERING CONGRESS, 1915

The attention of all engineers is called to the International Engineering Congress to be held in San Francisco in 1915 in connection with the Panama-Pacific Exposition, under the auspices of the American Society of Civil Engineers, American Institute of Mining Engineers, American Society of Mechanical Engineers, American Institute of Electrical Engineers, and the Society of Naval Architects and Marine Engineers.

The sessions will be held in 11 sections and the proceedings will be published in 10 volumes of about 500 pages each, with a somewhat smaller volume containing all general proceedings and an index digest of all papers.

All persons interested are invited to become members whether or not they may be able to attend the Congress.

The membership fee of \$5 will entitle holder to the volume of general proceedings and index digest and to any other single volume according to choice. Other volumes may be subscribed for according to a sliding scale depending upon the number of volumes taken.

The American Institute of Mining Engineers is an underwriter to the general guarantee fund for the necessary expenses of the Congress. Prompt action on the part of members in sending in their subscriptions to membership in the Congress will aid effectively in carrying on the necessary preliminary work without further present financial demands on the Institute for this purpose.

The Committee of Management of the Congress therefore specially requests all members of the Institute to respond promptly with their subscriptions.

EMMONS VOLUME ON ORE-DEPOSITS*A continuation of the "Posepny" Volume*

Comprising papers descriptive of ore-deposits and discussions of their origin, edited, with an introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

Contents

- Genesis of Certain Ore-Deposits. By S. F. EMMONS.
 Structural Relations of Ore-Deposits. By S. F. EMMONS.
 Geological Distribution of the Useful Metals in the United States. By S. F. EMMONS. Discussion, by JOHN A. CHURCH, ARTHUR WINSLOW, S. F. EMMONS, and WILLIAM HAMILTON MERRITT.
 Torsional Theory of Joints. By GEORGE F. BECKER. Discussion, by H. M. HOWE, R. W. RAYMOND, C. R. BOYD, and GEORGE F. BECKER.
 Allotropism of Gold. By HENRY LOUIS.
 Superficial Alteration of Ore-Deposits. By R. A. F. PENROSE, JR.
 Some Mines of Rosita and Silver Cliff, Colorado. By S. F. EMMONS.
 Genesis of Certain Auriferous Lodes. By JOHN R. DON. Discussion, by JOSEPH LE CONTE, S. F. EMMONS, G. F. BECKER, ARTHUR WINSLOW, W. P. BLAKE, and J. R. DON.
 Influence of Country-Rock on Mineral Veins. By WALTER HARVEY WEED.
 Igneous Rocks and Circulating Waters as Factors in Ore-Deposition. By J. F. KEMP.
 Consideration of Igneous Rocks and Their Segregation or Differentiation as Related to the Occurrence of Ores. By J. E. SPURRE. Discussion, by A. N. WINCHELL.
 Chemistry of Ore-Deposition. By WALTER P. JENNEY. Discussion, by JOHN A. CHURCH.
 Ore-Deposits near Igneous Contacts. By WALTER HARVEY WEED. Discussion, by W. L. AUSTIN.
 Ore-Deposition and Vein-Enrichment by Ascending Hot Waters. By WALTER HARVEY WEED.
 Basaltic Zones as Guides to Ore-Deposits in the Cripple Creek District, Colorado. By E. A. STEVENS.
 Geological Features of the Gold-Production of North America. By W. LINDGREN. Discussion, by W. G. MILLER and W. L. AUSTIN.
 Osmosis as a Factor in Ore-Formation. By HALBERT POWERS GILLETTE.
 Ore-Deposits of Sudbury, Ontario. By CHARLES W. DICKSON.
 Genesis of the Copper-Deposits of Clifton-Morenci, Arizona. By W. LINDGREN.
 Copper-Deposits at San Jose, Tamaulipas, Mexico. By J. F. KEMP.
 Magmatic Origin of Vein-Forming Waters in Southeastern Alaska. By A. C. SPENCER.
 Genetic Relations of the Western Nevada Ores. By J. E. SPURRE.
 Are the Quartz Veins of Silver Peak, Nevada, the Result of Magmatic Segregation? By J. B. HASTINGS.
 Occurrence of Stibnite at Steamboat Springs, Nevada. By W. LINDGREN.
 Summary of Lake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing Series. By C. K. LEITCH.
 Geological Relations of the Scandinavian Iron-Ores. By H. SJÖGREN.
 Formation and Enrichment of Ore-Bearing Veins. (With Supplementary Paper.) By GEORGE J. BANCROFT.
 Distribution of the Elements in Igneous Rocks. By H. S. WASHINGTON.
 Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits of the United States. By W. H. EMMONS.
 Cognate Papers.
 Bibliography of the Science of Ore-Deposits. By J. D. IRVING, H. D. SMITH, and H. G. FERGUSON.

The volume contains 1002 pages. Price, bound in cloth, \$5; in half-morocco, \$6. Both the Emmons and the Posepny Volumes on Ore-Deposits, bound in cloth, \$8; in half-morocco, \$10.

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LIBRARY ACCESSIONS

New Books on Mining

ACTION OF ACID MINE WATER ON THE INSULATION OF ELECTRIC CONDUCTORS. Tech. Paper No. 58, U. S. Bureau of Mines. Washington, 1913.

INFLAMMABLE GASES IN MINE AIR. Tech. Paper No. 39, U. S. Bureau of Mines. Washington, 1913.

PERMISSIBLE EXPLOSIVES TESTED PRIOR TO JAN. 1, 1914. Tech. Paper No. 71, U. S. Bureau of Mines. Washington, 1914.

SAMPLING AND EXAMINATION OF MINE GASES AND NATURAL GAS. Bull. No. 42, U. S. Bureau of Mines. Washington, 1913.

TREATISE ON THE AMERICAN LAW RELATING TO MINES AND MINERAL LANDS. Ed. 3, Vols. 1-3. By C. H. Lindley. San Francisco, 1914. (Gift of Author.)

[NOTE.—The first edition of this book, published in 1897, was eagerly taken by the public. A second edition, thoroughly revised and largely rewritten, appeared in 1903. Three years later, stereotype plates of the book, and the stock of copies on hand, were destroyed in the conflagration attending the San Francisco earthquake of 1906. But the author refused to have it reprinted without further amplification, and,

after the lapse of eight years, he now presents a third and final edition, including the numerous and important judicial decisions rendered since 1903, as well as the new statutes enacted during that period. The work comprises three volumes, containing 2,812 pages, and presenting in its logical arrangement, table of contents, list of cases, and full analytical index, a model of completeness and convenience, whether for perusal, for study, or for occasional reference. It would be foolish to compare this work with others in the same field. As was once said of the yacht "America" in a notable international race, Lindley is first—and there is no second. This verdict offends nobody; for even the authors of smaller treatises on the subject cordially recognize his book as a thesaurus of information and an authority never to be superseded. To the *Engineering and Mining Journal* of Apr. 18, 1914, I have contributed a more extended review of this book.—R. W. R.]

New Books on Metallurgy

DIE VEKOKUNG DER STEINKOHL BEI NIEDERER TEMPERATUR. Sammlung Berg und Hüttenmännischer Abhandlungen, pt. 138. Kattowitz, 1913.

DETAILS OF CYANIDE PRACTICE. By Herbert A. Megraw. McGraw-Hill Book Co., New York, 1914. (Gift of Publisher.) Price \$2.

[NOTE.—This is another of the useful books made by compiling articles in the *Engineering and Mining Journal*, only, unlike most of its predecessors, it is (with the exception of a single chapter) the work of a single contributor. The author, a member of the editorial staff of the *Journal*, already known through his *Practical Data for the Cyanide Plant*, here records the results of a tour among the cyanide-mills of North America, giving frankly his own observations and opinions of existing practice. The superiority of such a review over a symposium of different critics is, that all the descriptions bear the impress of one mind; contain similar data, similarly weighed and judged; and can therefore be compared with intelligence and ease. The book contains notes of plants and methods in the Cobalt and Porcupine districts of Canada, the Black Hills of South Dakota; Telluride and Cripple Creek, Colo.; Tonopah, Fairview, and Wonder, Nev.; Republic, Wash.; Grass Valley and Black Oak, Cal.; and Gold Road, Oatman and Wickenburg, Ariz. Numerous criticisms, evoked by the articles when they were first published, are given in the text, which is also illustrated with views of works and diagrams of flow-sheets.—R. W. R.]

ELECTRIC SWITCHES FOR USE IN GASEOUS MINES. Bull. No. 68, U. S. Bureau of Mines. Washington, 1913.

ELECTRO-THERMAL METHODS OF IRON AND STEEL PRODUCTION. By J. B. C. Kershaw, with an introduction by J. A. Fleming. London, 1913.

REPORT OF THE TESTS OF METALS AND OTHER MATERIALS. U. S. Ordnance Department. Washington, 1913.

New Books on Geology and Mineral Production

IGNEOUS ROCKS AND THEIR ORIGIN. By R. A. Daly. New York, 1914.

MAP OF THE KOLAR GOLD FIELD. Bangalore, 1914. (Gift of Mysore Geological Department.)

NEW ZEALAND OFFICIAL YEAR BOOK, 1913. Wellington, 1913. (Gift of New Zealand Government Statistician.)

OIL FIELDS OF CRAWFORD AND LAWRENCE COUNTIES, WITH MAPS. Bull. No. 22, Illinois State Geological Survey. Urbana, 1913.

THE MIDDLESEX AND MOUNT CLAUDE MINING FIELD, WITH MAPS. Bull. No. 14, Tasmania Geological Survey. Tasmania, 1913.

STATISTICS OF THE MINERAL PRODUCTION OF ALABAMA FOR 1911, 1912. Bull. No. 13, 14, Alabama Geological Survey. University, 1913.

ARGENTINE REPUBLIC. MINISTERIO DE AGRICULTURA. Memoria Ministerial de 1912. Buenos Aires, 1913.

Las Vetas con Magnetita (Martita) y las de Wolframita de la pendiente occidental del Cerro del Morro. Boletín No. 3. Buenos Aires, 1913.

Investigaciones hidrogeológicas de los valles de Chapalcó y Quehué y sus alrededores. Boletín No. 4. Buenos Aires, 1913.

Informe Preliminar sobre un Viaje de Investigación Geologica a las Provincias de Entre Rios y Corrientes. Boletín No. 5. Buenos Aires, 1913.

Apuntes Hidrogeologicos sobre el Sud-Este de la Provincia de Mendoza. Boletín No. 6. Buenos Aires, 1913.

Las Cales Cristalino Granulosas de la Sierra de Cordoba y sus Fenomenos de Contacto. Boletin No. 7. Buenos Aires, 1913. (Gift of Ministerio de Agricultura.)

New Books on Non-Metallic Minerals

LA SILICE ET LES SILICATES. By Henry Le Chatelier. Paris, 1914. (Gift of Author.)

[NOTE.—An extensive monograph by an Honorary Member of the Institute, in which are collected not only facts about natural silica and silicates, but data as to the chemical and physical properties of glass and pottery.—W. P. C.]

MUD-LADEN FLUID APPLIED TO WELL DRILLING. Tech. Paper No. 66, U. S. Bureau of Mines. Washington, 1913.

RADIUM, ITS PRODUCTION AND USES. By Sydney Fawns. London, 1913.

Company Reports

BROKEN HILL PROPRIETARY CO. Reports and Statements of Account for half year ending Nov. 30, 1913. Victoria, 1913. (Gift of Company.)

BROKEN HILL SOUTH SILVER MINING CO. Reports, Statements of Accounts, for half year ended Dec. 31, 1913. Melbourne, 1913. (Gift of Company.)

GOLDFIELD CONSOLIDATED MINES CO. Annual Report, 7th, 1913. Goldfield, 1913. (Gift of Company.)

PHELPS, DODGE & CO. Annual Report, 1913. New York, 1913. (Gift of Company.)

Trade Catalogues

CHICAGO PNEUMATIC TOOL CO., Chicago, Ill.

Bulletin No. 151. "Chicago Slogger" drills. Feb., 1914.

Bulletin No. 152. "Chicago Gatling" drills. Feb., 1914.

Bulletin No. 153. "Chicago Sinker." Feb., 1914.

Bulletin No. 154. "Chicago Stopper." Feb., 1914.

Bulletin No. 172. No. 5 Chicago Plug and Feather Drill. Feb., 1914.

CHICAGO STEEL TAPE CO., Chicago, Ill.

Descriptive circular of Chicago steel tape.

Descriptive circular of Architects' and builders' rod.

Descriptive circular of Chicago leveling rod.

Descriptive circular of Farmers' and drainage rod.

Descriptive circular of La Crosse leveling rod.

Descriptive circular of Mining leveling rod.

Descriptive circular of Self-computing leveling rod, Style "M."

Descriptive circular of Utility leveling rod.

DEISTER MACHINE CO., Fort Wayne, Ind. Catalogue describing concentrating machinery. 1914.

DENVER FIRE CLAY CO., Denver, Colo.

Case metallurgical furnaces. 1914.

Electroscope for determining radio-activity.

DRAEGER OXYGEN APPARATUS CO., Pittsburg, Pa. Description of Draeger Self-rescuer.

RUMSEY & CO., LTD., Seneca Falls, N. Y. Rumsey power pumps for industrial plants.

SANFORD-DAY IRON WORKS, Knoxville, Tenn. "Whitney" roller-bearing trucks.

SPARTA IRON WORKS CO., Sparta, Wis. Sludge bucket. April, 1914.

SUPPLEE-BIDDLE HARDWARE CO., Philadelphia, Pa. Bull., March, 1914. Monel metal.

SYMONS BROTHERS CO., Chicago, Ill. Pulsating screen.

TRENT, T. C. ENGINEERING CO., Los Angeles, Cal.

Catalogue No. 2. Cyanide machinery.

Bull. No. 6. T. C. Trent continuous system for hydrometallurgy especially adapted to cyaniding gold and silver ores.

WEBSTER MFG. CO., Tiffin, Ohio. Webster Method. March, 1914.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Mar. 10 to Apr. 10, 1914.

* *Members*

- ADAMS, DANIEL T., Mining.....1416 Westminster Bldg., Chicago, Ill.
 BLICKENSDECKER, FRANK CLEVELAND, Sampler..... Morenci, Ariz.
 CAVERS, THOMAS WILLARD, Met..... Copperhill, Tenn.
 CLAPP, CHARLES H., Geol..... Univ. of Arizona, Tucson, Ariz.
 CLARK, PATRICK T., Pres..... Bullwhacker Mine, Spokane, Wash.
 COMSTOCK, HARRY, Mining..... Mineville, Essex Co., New York.
 COOKE, JOSEPH H., Gen. Mgr..... Mono Dev. Co., Gisborn, Tooele Co., Utah.
 COOPER, DEXTER P., Civ. and Hyd. Engr..... 101 Park Ave., New York, N. Y.
 DeGOLYER, E. L., Geol..... Apartado 269, Tampico, Mexico.
 DESHAYES, ERNEST F., Mine Foreman..... Baldy, N. M.
 DOUGLASS, WILLIAM CRAWFORD, Min. Engr..... 107 Leonard Hotel, Butte, Mont.
 DOWNS, FLETCHER GARDNER, Min. Engr., Topographical Surveyor Instituto
 de Geologia y Perforaciones, 312 Calle Washington, Montevideo, Uruguay.
 DRESSER, JOHN ALEXANDER, Geol., Mgr., Lands Dept., Algoma Central &
 Hudson Bay Railway, Sault St. Marie, Ont., Canada.
 FAULKNER, ROBERT, Asst. Supt. Iron Ore Concentrator, Pa. Steel Co., Lebanon, Pa.
 FISCHER, SIEGFRIED, Instr. of Met., Lehigh Univ.,
 313 Cherokee St., So. Bethlehem, Pa.
 FLETCHER, A. R., Min. Engr., Care John Fletcher, 4 Hospital St.,
 C. P. R. Telegraphs, Montreal, Canada.
 FOOTE, GEORGE CLARK, Resident Director, Witherbee, Sherman Co.,
 Port Henry, N. Y.
 GOWING, FREDERICK A., Min. Engr..... 234 Carmel Ave., Piedmont, Cal.
 GRACE, WILLIAM L., Iron Ore Work..... 100 N. 8th St., St. Louis, Mo.
 GREENWAY, JOHN CAMPBELL, Gen. Mgr..... Calumet & Arizona Co., Warren, Ariz.
 HILBENZ, HANS, Met. Dir. of Fried. Krupp A. G., Essen, Friemersheim, Germany.
 HOFFER, ALLEN, Supt. Furnace..... 645 King St., Pottstown, Pa.
 KAFFER, STEPHEN L., Min. and Met..... Esqueda, Son., Mexico.
 KEMP, JAMES TAYLOR, Met., Concentrator Foreman, 261 Columbia Ave.,
 Palmerton, Pa.
 KOHL, R. WILLIAM, JR., Chief Engr., Spanish-American Iron Co.,
 Santiago de Cuba, Cuba.
 KRAEMER, FRANK JOSEPH, Asst. to Supt., Spanish-American Iron Co., Felton,
 Oriente, Cuba.
 LINDEMUTH, LEWIS B., Asst. Supt., Open Hearth Dept., Penn. Steel Co.,
 Steelton, Pa.
 LYON, FREDERICK, Min. Engr..... P. O. Box 385, Boston, Mass.
 MCQUIGG, CHARLES ELLISON, Teacher..... State College, Pa.
 MORY, SAMUEL ALBERT, Land and Mining Engr. and Draftsman,
 Box 56, Harmon Bldg., London, Ky.
 MATTESON, W. G., Min. Geol. and Pet. Engr., Dept. Geology, Lehigh Univ.,
 So. Bethlehem, Pa.
 NASON, STANLEY LEWIS, Mill Foreman..... Mineville, Essex Co., New York.
 NEILL, CHARLES PATRICK, Chairman Labor and Safety Com., American
 Smelting & Refining Co., 165 Broadway, New York, N. Y.
 NORCROSS, FRED STEPHENSON, JR., Min. Engr..... Greenwood, B. C.
 OATMAN, FRANKLYN WILLIAM, Min. Engr..... 2428 College Ave., Berkeley, Cal.
 PARSONS, ARTHUR C., Chief Chem..... Lackawanna Steel Co., Buffalo, N. Y.
 PARSONS, FLOYD W., Editor *Coal Age*..... 505 Pearl St., New York, N. Y.
 PICKRELL, EUGENE R., Met. Chem. and Assayer, 566 W. 162nd St., New York, N. Y.
 ROBERTSON, JOHN FERGUSON, Smelter Supt..... Coniston, Ont., Canada.
 RORK, FRANK CURTIS, Min. Engr., Mine Supt. Moose Mt., Ltd., Sellwood, Ont.,
 Canada.
 SAVAGE, HARLOW D..... American Arch Co., 30 Church St., New York, N. Y.

SKOGMARK, JOHN, Met. and Cons. Engr., Prime Western Spelter Co.,	Martins Ferry, Ohio.
SMALL, HORATIO LEVERETT, Geol., Care The American Consul, Rio de Janeiro,	Brazil, S. A.
SPICER, H. N., Met. Engr.....	30 Church St., New York, N. Y.
SWARVA, G. L., Miner.....	Treadwell, Alaska.
SWEENEY, HARRY P., Min. Engr.....	Care The Thomas Iron Co., Wharton, N.-J.
SWIFT, THEODORE VAN KLEECK, Min. Engr.....	Mineville, Essex Co., N. Y.
TAGGART, ARTHUR F., Asst. Prof. of Min. Engrg., Sheffield Scientific School,	Yale Univ., New Haven, Conn.
TOPPING, JOHN A., Chairman, Republic Iron & Steel Co., 17 Battery Place,	New York, N. Y.
VAN CAMPEN, FRANK RUMSEY, Mgr.....	Beatson Copper Co., Latouche, Alaska.
WALKER, LELAND ARTHUR, Mill Foreman.....	Rye Valley, Baker Co., Ore.
WATERHOUSE, GEORGE B., Met. Engr.....	Lackawanna Steel Co., Buffalo, N. Y.
WEINBERG, GEORGE SEMEN, Min. Engr., Kirpitcheny Pereoulouk No. 1,	St. Petersburg, Russia.
WERNECKE, LIVINGSTON, Min. Engr.....	4546 18th Ave., N. E., Seattle, Wash.
WHITE, DAVID, Geol.....	U. S. Geological Survey, Washington, D. C.
WOLFARD, MERL RUSKIN, Instr. Mech. Engr....	29 Everett St., Cambridge, Mass.

Associate Members

BARTHOLOMEW, WILLIAM FAYETTE, Broker....	50 Congress St., Boston, Mass.
SHAFFER, WILLIAM BARBER.....	152 South Broad St., Nazareth, Pa.
VAN RENSSELAER, ARTHUR MASON.....	715 Ocean Ave., New London, Conn.

Junior Members

BEHRE, HENRY AUGUST, Student.....	379 Temple St., New Haven, Conn.
COLE, ROBERT J., Student.....	207 W. Park St., Butte, Mont.
DOWNES, T. WATSON, Student.....	3rd and Conestoga Sts., Steelton, Pa.
DUNBAR, DOUGLAS MACDONALD, Student, 326 Wyandotte St., So. Bethlehem, Pa.	
PROTZELLER, RAYMOND J., Student.....	1221 3rd St., N. Catsaqua, Pa.

Change of Status—Associate to Member

SHIPP, ELLSWORTH MALTBY.....	2 Rector Street, New York, N. Y.
STEWART, JUDD.....	165 Broadway, New York, N. Y.

CANDIDATES FOR MEMBERSHIP

The following persons have been proposed during the period Mar. 10 to Apr. 10, 1914, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Harold Wallace Aldrich, Anaconda, Mont.

Proposed by E. P. Mathewson, C. D. Demond, Frederick Laist.

Born, 1884, South Dakota. 1898-1902, East Denver High School. 1902-06, Colorado School of Mines; E. M. 1906, Three months as Solution Man, Black Mountain Min. Co., Magdalena, Son., Mexico. 1906-07, Nine months as Assayer for Langridge Cyanide Plant, Boulder, Colo. 1907-13, Asst. Testing Engineer and Chemist for Anaconda Copper Min. Co., Anaconda, Mont.

Present position: Supt. of Leaching Plant, Anaconda Copper Min. Co.

Herbert M. Bacon, San Francisco, Cal.

Proposed by Ralph Arnold, W. R. Hamilton, Robert B. Moran.

Born, 1867, Greenwich, N. J. Public and Private Schools. 1881-82, Westtown Boarding School, Westtown, Pa. 1889-93, R. D. Wood & Co., Philadelphia. 1893-95, Mfgs. Mutual Fire Ins. Co., New York. 1895-1903, Inspector and Adjuster, various Fire Ins. Cos. of New York City. 1903-07, Banking Business. 1907-10, Devitt Tremble & Co., bankers, Chicago. 1910-14, Ford, Bacon & Davis, Engineers, New York City.

Present position: Engaged with Ford, Bacon & Davis of New York.

Pierre Francois Blik, Bolivia, S. A.

Proposed by H. F. Grondijs. (Two more signatures required).

Born, 1880, Rotterdam, Holland. 1893-98, General education at the "Hoogere Burgerschool" at Rotterdam. 1898-1903, Technical education at the Polytechnische School, Delft, Holland, with temporary stay at the Technical Univ. at Clausthal, Germany. 1903, Grad. as Mining Engineer. 1903, Asst. in Crystallography and Mineralogy at the Polytechnische School at Delft. 1904-06, Professor in Chemistry at the Naval School at Valparaiso, Chile. 1907, Manager of the Compania Estanifera de Berenguela, seat at Santiago, mines in Bolivia at Berenguela. 1908, Mgr. of the Empresa Minera de Japo, at Oruro.

Present position: 1909 to date, Mgr. of the tin and silver mill at Machacamarcas, of the Cia. Minera Oruro, Santiago.

Edward Browning, Cornwall, Pa.

Proposed by Harrison Souder, William Lloyd Wolfe, B. E. McKechnie.

Born, 1886, Camden, N. J. 1900-04, Episcopal Academy. 1904-08, Yale, Sheffield; B. S. 1909-11, Rand Mines, T. Lane Carter, French Rand. Mayer Bradford, W. W. Mein, Robinson Gold Mine. W. I. Honnold, Brakpans Mines. Rareris Mines, Butte, Mont.

Present position: Asst. to Supt., Cornwall Ore Bank Co., Cornwall, Pa.

DeNard Wilson Buckley, Wallace, Idaho.

Proposed by Rush J. White, John Gillie, B. R. Putnam.

Born, 1877, Philadelphia, Pa. 1896, Philadelphia Central High School. 1898, Cornell Univ., Certificate of Proficiency in Chemistry. 1901, School of Mines, Univ. of Mo.; B. S. 1901, Smelterman, Arizona. 1901-02, Millman, Montana, Madison Co. 1902-06, Assaying and Smelting, Butte, Mont. 1906, Engineering, Bingham, Utah. 1907-10, Assaying, Butte, Montana. 1910-11, Experimental work in smelting in New York City and Great Falls, Mont. 1911-13, Experimental work in milling, Mullan, Idaho.

Present position: Mill Supt., Stewart Mining Co., Wallace, Idaho.

Curtis Fairbairn Burt, Bingham Canyon, Utah.

Proposed by W. A. Barnes, H. D. Kinney, George O. Deahler.

Born, 1889, St. Ignace, Mich. 1908, Erie High School, Erie, Pa. 1906-07, Culver Military Academy, Culver, Ind. 1913, B. S., E. M., Michigan College of Mines. 1908-10, Practical underground work, Quincy Mining Co., Hancock, Mich. 1912, Practical underground work, Oliver Mining Co., Ishpeming, Mich.

Present position: 1913 to date, Engineer, Highland Boy Mine, Utah Cons. Mining Co., Bingham Canyon, Utah.

Scott David Gallaway, Iola, Kansas.

Proposed by Durward Copeland, H. A. Buehler, G. H. Cox.

Born, 1889, Nevada, Mo. 1906, Grad. Kemper Military School, Boonville, Mo. 1906-08, Missouri State Univ., Engineering, Columbia, Mo. 1908-09, Rolla School of Mines, Rolla, Mo. 1910-11, Mich. Univ., Engineering, Ann Arbor, Mich. 1911-12, Grad., Rolla School of Mines; B. S. 1909-10, Chemist, United Kansas Portland Cement Co., Iola, Kan. 1912-13, Custom Assayer, and Surveying, Nevada, Mo., and Iola, Kan.

Present position: Chemist, La Harpe Spelter Co., La Harpe, Kan.

William A. Castleton, Butte, Mont.

Proposed by Oscar Rohn, G. G. Vivian, J. H. Warner.

Born, 1882, Salt Lake City, Utah. 1909, Salt Lake Grade Schools. 1903, Grad. Salt Lake High School. 1905-07, Univ. of Cal., Berkeley. 1902, Daly West Mine, Park City, Utah. 1904-05, Dalton Lark Mine, Lark, Utah. 1907, Mammoth Mine, Kenneth, Cal. 1908, Valcaldia Mining Co., Blair, Nev. 1909-11, Utah Copper Co., Mill Foreman. 1912, Snowstorm Mining Co., Larson, Ida., designing mill and Mill Supt.

Present position: Mill Supt., East Butte Copper Min. Co.

Walter Barclay Chidester, Dedrick, Cal.

Proposed by H. A. Morrison, Francis A. Thomson, Thomas T. Read, and others.

Born, 1884, Memphis, Tenn. Grammar and high school completed in Tacoma, Wash. Grad. from Tacoma High School in 1903. Entered State College of Washington, Pullman, Wash., in same year. 1907, Grad., Wash. State College; B. S. 1904-06, Five months underground work in Morning, Hecla and Standard mines, Cœur d'Alene district, Ida. 1908-10, Millman in Sweeney and Morning mills of Federal Min. & Smelt. Co., at Sweeney and Mullan, Ida., Chairman for N. P. Ry. survey, N. D. 1910-11, Millman in Goldfield Cons. and Nevada Goldfield Reduction Works at Goldfield, Nev. 1911-12, Draftsman, engineer, assayer, Tonopah-Belmont Development Co., Tonopah. Supt. Dreisman mine, Tuolumne, Cal. 1913, Three months engineer and shift boss, Jim Butters mine, Tonopah, Nev.

Present position: Shift boss in 20-stamp cyanide mill, Globe Cons. Min. Co., Dedrick, Cal.

Adolph Clayton Clark, Perth Amboy, N. J.

Proposed by W. S. Harper, F. W. Allen, B. B. Thayer,

Born, 1866, Boston, Mass. Preparatory School and Anglo-French College, London, England, also 1½ years at school in Paris, France. 1893-1901, Asst. Supt. New England Electrolytic Copper Co., Central Falls, R. I.

Present position: 1901 to date, Supt. Raritan Copper Works, Perth Amboy, N. J.

Edward Hardy Clark, New York, N. Y.

Proposed by B. B. Thayer, W. S. Harper, J. W. Allen.

Born, 1864, St. Louis, Mo. General education in Missouri Schools.

Present position: 1896 to date, Manager of the Mining Interests of the Estate of Senator George Hearst.

Frederick P. Clark, Hammond, Ind.

Proposed by G. P. Hulst, W. S. Harper, J. W. Allen.

Born, 1876, London, Eng. International College, London, Eng. Institute of Technology, London, Eng. 1897-1902, Assayer, Raritan Copper Wks., Perth Amboy, N. J. 1902-05, Chemist, Arizona Copper Co., Clifton, Ariz. 1905-06, Chemist, Poland Mg. Co., Poland, Ariz. 1906-11, Chemist, Creston Colorado Co., Torres, Son., Mexico.

Present position: Chief Chemist, International Lead Refining Co., East Chicago, Ind.

John Vaughan Colston, Mineral, Va.

Proposed by Clarence J. London, F. Lynwood Garrison, R. H. Sanders.

Born, 1886, Iowa. 1900-02, Workshop College, Notts, Eng. 1903-06, Three Years, South Kensington Lectures and Examinations (Coal Min.), London, Eng. 1902-04, Mining pupil to M. Deacon, Sheepbridge Coal & Iron Co., Ltd., near Chesterfield, Eng. 1904-05, Asst. Mgr. Crown Farm Mine and Creswell Mine of the Bolsover Colliery Co., Bolsover, near Chesterfield. 1906, Surveyor, in charge of Underground Plots, Grassmore Coal & Coke Co., near Chesterfield. 1907-10, Pract. Mining, Staveley Coal & Iron Co., near Chesterfield. 1910-11, Chesapeake & Ohio Ry. Co., Construction Staff. 1911-13, Reconstruction of street car track, Virginia Railway & Power Co., Richmond, Va.

Present position: Asst. Mgr. and Engineer, Boyd Smith Mines, Inc.

M. J. Conover, Tonopah, Nev.

Proposed by W. H. Sirdevan, W. H. Blackburn, George R. Elder.

Born, 1887, Matawan, N. J. 1904-07, Blair Academy, Blairstown, N. J. 1907-11, Lafayette College, Easton, Pa.; E. M. 1912, Assayer and Surveyor, Kimberly Cons. Mines Co., Hilltop, Nev. 1913, Asst. Supt., Black Canyon Cons. Mines Co., Lovelocks, Nev. 1913, Millman, Jumbo Reduction Co., Bonnie Clare, Nev.

Present position: Engineering Staff, Tonopah-Belmont Dev. Co., Tonopah, Nev.

William Henry Coolidge, Boston, Mass.

Proposed by H. L. Smyth, George S. Raymer, Charles H. White.

Born, 1859, Natick, Mass. 1877, Entered Harvard College, being graduated from Newton High School. 1881, Grad., Harvard College, then attended Harvard Law School. 1885, Admitted to practice law in Mass. and U. S. Courts. 1885-1914, Lawyer, firm of Coolidge & Hight. Since 1895, much interested in mining matters, being director and counsel for U. S. Smelting, Refining & Mining Co., Island Creek Coal Co., American Zinc, Lead & Smelting Co., Alaska Gold Mining Co.

Present position: Lawyer.

Emilio Cortese, Sebaieh, Upper Egypt.

Proposed by John M. Cairns, Charles F. Rand. (One signature required.)

Born, 1856, Alexandria, Italy. 1877, School for Civil Engineers in Rome, Italy. 1879, Mining Engineer, Certificate of the Mining School of Paris, appointed Geologist, and then Royal Mining Engineer, in the Royal Mining Service of Italy. 1879-1902, Geologist, in the Geological Survey of Italy, and Royal Mining Engineer. 1902-04, Technical Manager Sulphur Mines in Italy. 1904-07, Technical Manager Coal Mines in Italy. 1907, Technical Manager Iron Mines of Elba and Blast Furnaces of Tollonica. 1901-02, Technical Manager Coal Mines in Venezuela. 1902-12, Consulting Engineer and Technical Manager, in Copper-Iron-Tin-Antimony mines, Italy, and granite and marble mines.

Present position: Director, Manager and General Manager, in the Phosphate mines at Sebaieh, Upper Egypt.

Charles Wilson DeWitt, Seikwan, Korea.

Proposed by J. L. Slater, Jr., J. S. Bradford, James J. Martin.

Born, 1890, Kalamazoo, Mich. 1896-97, Public School, Kalamazoo, Mich. 1897-1900, Public School, Chicago, Ill. 1900-04, Public School, Corning, Cal. 1906-09, Augusta Military Academy, Fort Defiance, Va. 1910-13, Polytechnic College of Engineering, Oakland, Cal.; degree, in Mining as Associate in Science. 1904-06, Mining, South American Development Co., Zaragosa, Rep. de Colombia, S. A. 1909-10, Clerk, Pacific Vinegar & Pickle Works, San Francisco, Cal.

Present position: Chemist and Mine Surveyor of Chilsan Mining Co., Seikwan, Korea.

Daniel Montgomery Drumheller, Jr., Republic, Wash.

Proposed by S. H. Richardson, L. K. Armstrong, F. A. Ross.

Born, 1890, Spokane, Wash. 1905-09, Spokane High School. 1909-13, Univ. of Cal., Mining Engineering. 1913, Engineer, Maple Leaf Coal Co., Maple Leaf, Alberta.

Present position: 1913 to date, Supt. at San Poil Cons. Co., Republic, Wash.

Edwin R. Eaton, Crown Point, N. Y.

Proposed by S. H. Hamilton, Kirby Thomas, N. V. Hansell.

Born, 1888, Crown Point, N. Y. 1905, Crown Point High School. 1906, Vermont Military Acad. 1910, Syracuse Univ. 1910-12, Gawny Diamond Drilling Co., Arnold Hill, N. Y. 1911, Hamilton & Hansell, Essex Mine. 1912-14, Manatawny Bessemer Ore Co. 1913, Hatch & Clute, Mineville District, N. Y.

Present position: Engineer for Manatawny Bessemer Ore Co.

Dr. Lazar Edeleanu, Berlin, Germany.

Proposed by Anthony F. Lucas, David T. Day, M. R. Campbell.

Born, 1861, Bucarest. 1883, B. S., Bucarest. 1887, Ph. D., Berlin, chemistry being the principal study. 1887, Member of Chemical Society, Berlin. 1890, Member of Physic Chemical Society, Bucarest. 1909, Member of Society of Naturalists, Moscow. 1888, Asst. Chemist in the Royal Artillery College, Woolwich, London. 1890, Chief Chemist Travaux Chimiques Univ., Bucarest. 1897, Dir. of the Chemical Laboratory, Ministre of Domains, Bucarest. 1906, Dir. of the Chemical Laboratory of the Geological Institute, Bucarest. 1904-11, Dir. of the Vega Soc.

Present position: Director of the Allgemeine Gesellschaft für Chemische Industrie, Berlin.

Luis Calvillo Espinosa, Pachuca, Mexico.

Proposed by Victor M. Braschi, Clive S. Newcomb, Ezequiel Ordenez.

Born, 1879, Zacatecas, Mexico. Escuelas Oficiales e Instituto de Ciencias de Zacatecas. Escuela Practica de Minas en Pachuca, Hdgo. Gradua do el 19 de Febrero de 1902 in Zacatecas, Zac., Mex. 1902, Supt. "Gran Cuadra Prodigio y Onx." Zacatecas. 1904, Chief Engineer the "Compania Restauradora del Mineral de Veta grande," Zac. 1906, Official Engineer of Zacatecas State. 1910, Asst. to Ing. Ezequiel Ordenez. 1913, Agent of the "Compania Exploradora El Encino y Onx," in Pachuca, Hdgo.

Present position: Asst. to Engineer Ezequiel Ordenez, on examination works.

Oscar N. Friendly, Park City, Utah.

Proposed by James Humes, Fred T. Williams, George D. Blood.

Born, 1884, Corvallis, Ore. 1907, B. S., Univ. of Cal. 1907-09, Asst. Engr. Daly-West Min. Co., Park City, Utah. 1909-13, Engr. Daly-Judge Min. Co., Park City, Utah, and Snake Creek Min. & Tunnel Co.

Present position: 1913 to date, Genl. Supt. Daly-Judge Min. Co., Genl. Supt. Mid-West Tunnel Co.

Paul Whitefield Gaebelein, Cornucopia, Ore.

Proposed by J. F. Kemp, Charles P. Berkey, Henry S. Munroe.

Born, 1888, Hoboken, N. J. 1901-05, High School education. 1905, Columbia Univ. 1909, School of Mines; E. M. 1909, Otisse Min. Co., Elk Lake, Ont. 1909, Crown Reserve Min. Co., Cobalt, Ont. 1909, Golden Cycle Min. Co. 1911, Colo. Springs, Colo., Cyanide Plant, various positions. 1911, Testing work. Examination. 1912, Work for private parties. 1912, Smuggler-Union Min. Co. 1912, Telluride, Colo., Cyanide Plant, Construction.

Present position: Mill Supt., Cornucopia Mines Co., Cornucopia, Ore.

James Owen Greenan, Fairview, Nev.

Proposed by Charles C. Starr, Frederick J. Siebert, Charles E. Locke.

Born, 1888, Taunton, Mass. 1904-08, Boston University; A. B. 1908-11, Mass. Inst. of Technology; B. S. 1909-11, Member of Mining Engineering Society of Mass. Inst. of Tech. 1911-12, Miner for Parnott Min. & Mill. Co., Central City, Colo.

Present position: 1912 to date, Engineer, Nevada Hill Mining Co., Fairview, Nev.

Haines Gridley, San Pedro, N. M.

Proposed by A. H. Case, Leo F. Arnold, E. H. Westlake.

Born, 1880, Elmira, N. Y. 1900-04, Cornell Univ.; C. E. 1904-12, Engineer, Ophir Hill Cons. Min. Co., Ophir, Utah.

Present position: 1912 to date, Supt. Santa Fe Gold & Copper Mg. Co.

Frank Iden Griffiths, Queensland, Australia.

Proposed by John W. Moule, E. Noel Goode, B. G. Patterson.

Born, 1877, Launceston, Tasmania. 1884-88, Collegiate Inst., Launceston, Tasmania. 1888-94, Launceston Church Grammar School, Tasmania. 1910, Metallurgy Course, International Correspondence Schools, Scranton, Pa. Received Diploma. 1913, One year Chemistry, Technical College, Mount Morgan, Queensland, Australia. 1895-96, Worked on Construction Mount Lyell Min. & Railway Corp. Reduction works. 1896-97, Smelter Shift Boss. 1897-11, Converter, Shift Boss and Foreman. 1912, General Foreman, Mount Morgan. 1913, Gold Mining Corp. Smelting and Converting Plant.

Present position: 1913 to date, Asst. Metallurgist, Mount Morgan Gold Mining Co.

Robert H. Gross, Boston, Mass.

Proposed by Horace V. Winchell, Oscar Rohn, J. H. Warner.

Born, 1864, Bloomington, Mich. District school in Michigan and Ohio, and public schools in Ravenna, O., terminating with the first year in the high school.

Present position: President of East Butte Copper Mining Co., of Butte, Mont. Pres. New River Co., and subsidiaries, of West Virginia. Sec. and Treas. of several other companies.

Harry Frank Guggenheim, New York, N. Y.

Proposed by Willard S. Morse, A. Eilers, Stephen Birch.

Born, 1890, New Jersey. Secondary education, New York City. 1907, Entered Sheffield Scientific School. 1910, Entered Pembroke College, Cambridge, Eng. 1913, Grad. Cambridge Univ.; B. A. Degree taken in Chemistry and Political Economy. 1907-10, Entered employ of American Smelting & Refining Co., Aguascalientes, Mexico.

Present position: Mining and Smelting.

Maurice H. Hare, Spokane, Wash.

Proposed by L. K. Armstrong, J. C. Haas, E. P. Spalding.

Born, 1869, Greensburg, Pa. 1874-94, Common and High Schools, Greensburg, Pa. 1906-14, Alluvial Mining, Idaho and elsewhere, Hydraulic and Dredging.

Present position: Pres., Idaho Placer Mines Co., Ltd.

Edmund Norris Hobart, Morenci, Ariz.

Proposed by Frank D. Rathbun, W. H. McLean, S. F. Shaw.

Born, 1888, Mankato, Minn. 1910, New Mexico School of Mines; B. S. in Min. Engr.; B. S. in Met. Engr. 1910-11, Assayer, Shannon Copper Co., Metcalf, Ariz. 1911-12, Asst. Engr. Amer. Smelting & Refining Co., Anganguo, Mich., Mexico. 1912-13, Mine Foreman, Mazapil Copper Co., Ltd., Concepcion del Ore, Zac., Mexico. 1913-14, Chief Engr., A. S. & R., Charcas, S. L. F., Mexico.

Present position: Mining Engr., Detroit Copper Mining Co., Morenci, Ariz.

Joseph Inch, Chile, So. America.

Proposed by Arthur W. Martin, Lester W. Strauss, S. H. Loram.

Born, 1887, La Serena, Chile. 1895-96, Private English School, La Serena, Chile, S. A. 1897-02, Liceo de Lombres de La Serena. 1903-05, Escuela Practica de Minería de La Serena, Chile, S. A. 1906, Foreman and Asst. Engr. Mina Amarilla Cerro Blanco, Copiapo, Chile. 1907-11, Mine Supt. Mina Coguimbana, Cerro Blanco, Copiapo, Chile. Sociedad de Minas i Fundiciones de Carrizal. 1912, Asst. Mgr. and Asst. Engr., El Salado mines, mill, cyanide, and hydraulic electric plant. The San Juan Mines Argentine, Ltd.

Present position: Manager Mine, San Jose Compania Minera del Cajon de Navarro.

John Vincent Kelly, Jalisco, Mexico.

Proposed by C. Barnes Hoadley, Charles F. Rand, Louis D. Huntoon.

Born, 1880, Buffalo, N. Y. 1905, Harvard Univ.; S. B. in Science. 1906, Harvard Univ.; S. B. in Mining. 1906-07, Asst. Engineer with Guantanamo Exploration Co., Cuba. 1907-08, Engineer with Tinbaden Coal & Coke Co., Tinbaden, Va. 1908-09, Engineer, Tintic Smelting Co., Silver City, Utah. 1909, Millman, McGill Concentrator, Ely, Nev. 1909-11, Mines, Bisbee, Ariz. 1911-13, Mines, Esperanza Min. Co., El Oro, Mexico.

Present position: Mine Supt., Faver Mining Co., Hostotipaquillo, Jalisco, Mexico.

Maurice Walter Kishman, Cripple Creek, Colo.

Proposed by Dorsey A. Lyon, Robert M. Keeney, Robert S. Lewis.

Born, 1885, Pueblo, Colo. 1903, Grad., Central High School. Pueblo, Colo. 1903-1906, Student of Mining Engineering, Colo. School of Mines, Golden, Colo. 1906-08, Summers spent as miner, Gold King Cons., Silverton Colo. 1908-10, Head assayer and chemist, Gold Prince Mines, Silverton, Colo. 1908-10, Mining Engineer, Western Inv. Co., Victor, Colo. 1910-12, Field Engineer, Costilla Estates Dev. Co., San Acacio, Colo. 1912-14, Mining Engr., Western Inv. Co., Victor, Colo.

Present position: Supt., American Flag Mine, Park City, Utah.

K. D. Koliassnikoff, Perm, Russia.

Proposed by R. Gilman Brown, D. P. Mitchell, T. J. Jones.

1902, Grad. Frieberg Mining Academy, Met. Engr. 1902-04, Chief Chemist Al-paefsky Steel Wks. 1904-07, Mgr. Kartaff Steel Wks. 1907-08, Mgr., Kasli Wks. of the Kyshtim Corp. 1908-11, Mgr., Kyshtim Iron and Copper Wks.

Present position: 1911 to date, Mgr., Karabish Plant, Kyshtim Corp., and Mgr. Srymanapk Valley of same concern.

Milton O. Knauss, Catasauqua, Pa.

Proposed by S. G. Valentine, C. G. Peattie, Leonard Peckitt.

Born, 1885, Macungie, Pa. General school education at Macungie public school. 1898, Started to work for Crane Iron Works, as dish washer in laboratory. 1899, Chemist, Allentown Rolling Mill Co. 1900, Asst. Chemist, Crane Iron Works. 1902, Promoted to Chief Chemist. 1909, promoted to Supt., Macungie furnace, Empire Steel & Iron Co., 1911, Supt., Topton furnace, Empire Steel & Iron Co. 1911, Supt., Crane Iron Works, Empire Steel & Iron Co.

Present position: Supt. Blast Furnaces, Crane Iron Works.

Edward Clarence Lee, Harrisburg, Ill.

Proposed by W. D. Owens, J. M. Humphrey, Charles Enzian.

Born, 1886, Stroudsburg, Pa. 1902, Stroudsburg High School. 1903, Stroudsburg High School, Commercial. 1904, Wyoming Seminary, College Prep. 1908, Penn. State College; B. S. 1913, Penn. State College; E. M. 1908, Transitman, Lehigh Valley Coal Co., Wilkes-Barre, Pa., and Chief of Party, Penn. State Highway Dept. 1909-13, Transitman, Lehigh Valley Coal Co., Wilkes-Barre, Pa.

Present position: Instructor in the Illinois Miners and Mechanics' Institutes, in charge of the schools in Harrisburg and Herrin.

Maurice E. Lombardi, Berkeley, Cal.

Proposed by M. L. Requa, P. R. Bradley, F. W. Bradley.

Born, 1878, Houston, Texas. 1900, Yale Univ.; A. B. 1904, Univ. of Cal.; B. S. 1901, Mariposa Mine, Cal., underground work. 1904, Mountain View Mine, Ore., Underground work. 1904 to date, Asst. Geologist, Southern Pacific Co.; Engineer, Kern Trading & Oil Co.

Present position: Supt. of Construction and Developments, Kern Trading & Oil Co.

John Munro Longyear, Jr., Cambridge, Mass.

Proposed by Scott Turner, H. L. Symth, L. C. Graton.

Born, 1889, Marquette, Mich. 1906-07, Mass. Inst. of Technology. 1908-09, Harvard College. 1909-12, Mich. College of Mines; B. S., E. M. 1913-14, Harvard Univ., Grad. School of Applied Science. 1912-13, Superior Copper Co., Houghton, Mich. 1913, Osceola Cons. Min. Co., Osceola, Mich. Employed as Efficiency Engineer at both the above properties. Left this employment on account of uncertainties arising from the strike situation. Expect to work in Lake Superior iron ranges after 1914.

Present position: Grad. Student in Geology, Harvard Univ. Grad. School of Applied Science.

William D. Mangam, Butte, Mont.

Proposed by John Gillie, E. H. Wilson, Fred T. Greene.

Born, 1877, New York, N. Y. Brown Univ. Univ. of Virginia, Law School. 1900-14, W. A. Clark, Jr., agent in charge of Mining and Milling.

Present position: W. A. Clark, Jr., Agent. Sec'y and Treas., Elm Orbe Mining Co., Sec'y and Treas., Timber Butte Milling Co.

William M. Manning, Spokane, Wash.

Proposed by L. K. Armstrong, F. A. Ross, Roy H. Clarke.

Born, 1875, Canada. 1898, S. P. S., Toronto. American Hill Min. Co. 1899, Second Relief. 1900-12, First Thought. 1903, Easter Sunday. Private work as Consulting, Hydraulic, etc.

Present position: Chief Engineer, International Power & Manufacturing Co.

Edward Francis McCrossin, Birmingham, Ala.

Proposed by C. A. Moffett, H. S. Geisner, Erskine Ramsay.

Born, 1887, Philadelphia, Pa. 1905, Grad. Birmingham High School. 1906, Finished Junior Year, E. M. Course, Ala. Polytechnic Institute. 1907-08, Condensed Course Mining Engineering, Ohio State Univ. 1908, Installing mining machinery, Robinson Manufacturing Co., West Virginia mines. 1909, Prospecting and surveying coal lands for H. F. De Bardeleben and Ala. Fuel & Steel Co. 1910-12, Division Engineer, Division Supt. and Asst. General Supt. Ore Mines and Quarries, Georgia Steel Co. (Subsidiary Southern Iron & Steel Co.) 1912-14, Consulting Engineer, operating personal iron ore mine. Reports and exploration properties in Alabama, Louisiana and Mexico.

Present position: Consulting Engineer.

James Agnitus McIlwee, Denver, Colo.

Proposed by W. L. Saunders, Thomas B. Stearns, George A. Howells.

Born, 1864, Glasgow, Scotland. School education in Scotland. Has made a study of the technical work of tunnel driving, acquiring a superior knowledge of tunneling efficiency when in personal charge of Cripple Creek Drainage Tunnel, Cripple Creek, Colo., 1908; 1910, Laramie-Poudre Tunnel, at Laramie, Colo. Now engaged in driving the Snake Creek Tunnel at Herber City, Utah, also in driving Rogers Pass Tunnel at Glacier, B. C., for the Canadian Pacific Railway. In all of this work had full charge and direction of the design of apparatus and process of driving. His method is an acknowledged success in speed and efficiency and it has been adopted by other tunnel drivers in the United States and foreign countries.

Present position: Contractor driving Canadian Pacific Railway Tunnel at Rogers Pass, Glacier, B. C.

Elwood S. Moore, State College, Pa.

Proposed by J. F. Kemp, H. Ries, H. D. Pallister.

Born, 1879, Ontario, Canada. 1904, Univ. of Toronto; B. A. 1907, Univ. of Toronto; M. A. 1909, Univ. of Chicago. Training has been in Natural Science, specializing in Geology and Mineralogy; Ph. D. 1904-05, Science Master, Pickering College. 1905-06, Science Master, Collingwood Collegiate Inst. 1904-10, Summers spent first as an assistant and during the latter five summers as chief of geological parties for the Ont. Bureau of Mines. 1912, Summer with the Canadian Geol. Survey. 1913, Traveled in B. C., Alaska, and Yukon. 1911, Traveled in Europe. 1907-09, Fellow in Geology, Univ. of Chicago.

Present position: 1909 to date, Prof. of Geology and Mineralogy and Head of the Department of Penn. State College.

Leland Bertley Mowry, Copperhill, Tenn.

Proposed by J. B. Risque, T. W. Cavers, M. A. Caine.

Born, 1889, Providence, R. I. 1904-07, Drury High School, North Adams, Mass. 1907-11, Cornell Univ.; B. C. 1911-13, Chemist for Tenn. Copper Co.

Present position: 1913 to date, Smelter Foreman, Tenn. Copper Co.

Gerald F. Murphy, Nova Scotia.

Proposed by Stanley N. Graham, W. F. Jennison, C. L. Cantley.

Born, 1886, Halifax, N. S. 1900-03, Ampleforth College, York, Eng. A. A., Oxford. 1903-07, Dalhousie Univ., Halifax; B. E. 1908, Post-Graduate course at Columbia (School of Mines). 1909-14, Instructor in Mining, Nova Scotia Technical College. Miscellaneous examinations through Nova Scotia.

Present position: Instructor in Mining, Nova Scotia Technical College.

Allan Franklin Muter, Mogollon, N. M.

Proposed by C. Colcock Jones, A. J. Anderson, C. O. Lindberg.

Born, 1876, Youngstown, Ohio. Rayen High School, Youngstown, O. Kenyon College, Gambier, O. Case School of Applied Science, Cleveland, Ohio; B. S. 1904, E. M. 1908. 1904-05, Supt. of Ramrod Mine, Chloride, Ariz. 1905-06, Assayer & Asst. Met., Union Smelting Co., Chloride, Ariz. 1906-07, Shift Boss for Altata Mining Co., and in charge of Electrostatic separators, Tenn. Min. Co., Chloride, Ariz. 1907-08, Supt. and Mgr., Klondike Cyanide Co. 1908-12, Engr. and Assayer office, Kingman, Ariz. 1912-13, Experimental plant, treating Mexican ores, Douglas, Ariz.

Present position: 1913 to date, Supt. mill, Ernestine Min. Co.

Willard H. Nash, Gleeson, Ariz.

Proposed by G. E. Kedzie, John E. Penberthy, W. G. McBride.

Born, 1885, Buffalo, N. Y. 1900, Grad. Buffalo Public Schools. 1904, Grad. Buffalo High School. 1910, Univ. of Ariz., Tucson, Ariz.; B. S. 1904-05, Draughtsman, David Bell Engineering Works, Buffalo, N. Y. 1906-07, Ore Dept. Selby Plant, American Smelting & Refining Co., San Francisco. 1911-12, Asst. on mine examination, Spurr & Co., on American Smelting & Refining Co. properties in Mexico. 1912-13, Assayer and Engineer, at Sierra Mojada, Coah., Mexico, with Compania Metalurgica Mexicana, one of Robert S. Towns properties.

Present position: Assayer and Chemist, Shannon Copper Co., Gleeson, Ariz.

Walter Neal, Jalisco, Mexico.

Proposed by C. Barnes Hoadley, Charles F. Rand, Louis D. Huntoon.

Born, 1881, Lancashire, England. Grammar school in San Francisco, Cal. Apprentice with Luis Falkenau School of Assaying in San Francisco. 1899-1904, Tombay Gold Mines Co., Telluride, Colo., Assayer, amalgamated-vannerman, shift boss, helper in mine office. 1904-06, El Oro Mining & R. R. Co., Ltd., El Oro, Mexico; shift boss and experimental cyanide work. 1906-12, Cia. Minera Las Dos Estrellas, El Oro, Mexico. Cyanide experimental work, Cyanide Supt., Asst. Mgr.

Present position: 1912 to date, General Manager of El Favor Mining Co., and Mololoa Mining Co.

Joseph P. Nolan, Butte, Mont.

Proposed by Frederick T. Greene, Alexander Leggat, E. H. Wilson.

Born, 1887, Ishpeming, Mich. 1905-08, Michigan College of Mines; B. S. 1909, With city engineer of Butte, Mont. 1909-12, A. C. M. Co., Butte & Boston Dept.

Present position: 1912 to date, Private Practice, Mining Engineer.

P. A. O'Brien, New Orleans, La.

Proposed by R. B. Stanford, Charles F. Rand. (One more signature required.)

Born, 1874, Victoria, Aust. 1889, finished public school. 1891, Two years grammar school, Donald, Vic. 1893, Two years mining school, St. Armand, Vic. 1901-02, Mining and Chemistry at Washington Agricultural College, Pullman, Wash. 1902-04, Broadway Mine, Mont. 1904, Rapid City Smelter. 1905-07, El Oro Min. Co., Mexico. 1907, Desert Mill, Tonopah Min. Co., Nevada. 1908-09, Goldfield Con. Min. Co., Nevada. 1910, El Vesuvia Mine and Mill, Nicaragua. 1911-12, Bonanza Mine, Nicaragua.

Present position: Engineer at large.

John Clark Pickering, New York, N. Y.

Proposed by R. T. Bayliss, Walter McDermott, Edward Hooper.

Born, 1882, New York City. Ordinary public school education in New York City and elsewhere; two years at C. C. N. Y.; Columbia School of Mines, from which I graduated in 1904 with E. M. 1904, Engr., with Cerro de Pasco Mining Co., Peru, S. A. 1905, Chief Engr., Compania de Minas de Morococha, Peru, S. A. 1906-07, Examining Engr., in Peru with Frank Klepetko of New York City. 1907-08, Chief Engr., El Oro Mining & Ry. Co., Ltd., of El Oro, Mex. 1908-09, Examining Engineer Exploration Co. of England & Mexico, Ltd., in Mexico. 1910, Special examinations for Exploration Co., Ltd., of London. 1910, Genl. Supt., Mexico Mines of El Oro, Ltd., El Oro, Mexico. 1911, Genl. Mgr., Buena Tierra Mining Co., Chihuahua, Mex.

Present position: 1912 to date, Genl. Mgr., British South Africa Company's Mines Development Co., Ltd., Rhodesia, South Africa.

Alfred Harold Ryder, Kuantan, Singapore.

Proposed by J. T. Marriner, T. A. Rickard, Scott Turner.

Born, 1874, Cheshire, England. 1890-95, Merchant Taylor School, Crosby, Liverpool. Oxford Local Examination Certif. Night classes. Certifs., Machine Construction, Mechanics, Steam, Electricity. Apprenticeship, 1890-92, Messrs. Ismay Nurie & Co., Liverpool, Engineer's repair shop. 1892-95, Messrs. David Roller & Sons, Liverpool, Engineers and Shipbuilders. 1897, Board of Trade Certificate. 1896-97, Fourth Engineer, S. S. "Alderley." 1897, Third and Second Engineer, S. S. "Saint Cuthbert." 1898, Messrs. S. Z. de Ferrauti. 1898-1904 Chief Engineer, Borneo Co., Ltd., Gold Mines, Borneo. 1904-05, Manager of Cyanide Plant, for above company. 1906-09, Part owner with Ben Howe, A. R. S. M. M. I. M. M., Gold mine in Borneo. 1910-11, Manila Trading Co., Ltd., Manchester, Reporting on Mining property in Manila.

Present position: 1911 to date, Chief Mechanical Engineer, Pahang Cons. Co., Ltd.

George O. Scarfe, Alleghany, Cal.

Proposed by William Hague, Robert H. Bedford, Arthur B. Foote.

Born, 1887, Canton, Tioga Co., Pa. 1901, Grammar school. 1901-05, High School. 1906-10, Completed Junior year at Univ. of Cal. 1905-08, Field Engineer, Pacific Gas & Electric Co.

Present position: 1910 to date, Field Engineer, Supt. of Construction, North Star Mines Co., Grass Valley, Cal., and at present Supt. of Middle Yuba Power Co., and Engineer for Tightner Mines Co.

William Allen Scheuch, New York, N. Y.

Proposed by Felix A. Vogel, A. M. Tweedy, C. Kirchhoff.

Born, 1889, Brooklyn, N. Y. 1896-1908, Public and High School, New York, N. Y. 1908-12, Columbia School of Mines; E. M. 1912-13, Chile Exploration Co., Chuquicamata, Chile, sampling and surveying.

Present position: 1913 to date, Engineer, Bates Iron Co., Florence Iron Co. of Wisconsin, General Briquetting Co.

Albert Heinrich Schroeder, Gelsenkirchen, Germany.

Proposed by C. H. Macnutt, William J. Lakeland, P. K. Lucke.

Born, 1876, Gelsenkirchen, Germany. 1897-1901, Universitat und Technische Hochschule, Berlin. 1896-97, Krupp Guss-stahlfabrik, Essen. 1901-03, Chemist, Guss-stahlfabrik, Gelsenkirchen. 1903-04, Asst. Supt. Cia. de Fierro y Acero, Monterey, Mex. 1904-05, Asst. Supt. Montezuma Lead Co., Santa Barbara, Chih., Mex. 1905-13, Asst. Gen. Supt., Cia. Minera de Penoles, Durango, Mex.

Present position: Sept. 1, 1913, to date, Smelter Supt., Burma Mines, Ltd., Namtu, Burma, India.

Walter Scribner Schuyler, Pike, Sierra Co., Cal.

Proposed by E. Chester Turner, Arthur DeW. Foote, Charles C. Derby.

Born, 1850, Ithaca, N. Y. 1866, Mil. Acad., West Point. 1870, Graduated. 1913, Retired from active service in the army. 1879-81, While still in the army was Supt. of the Murchie Mine, Nevada Co., Cal. 1880, Was a member of this Institute for several years but withdrew on account of absorption in other pursuits.

Present position: President and General Manager of the Sierra Alaska Mg. Co.

Victor Benno Seidel, New York, N. Y.

Proposed by W. L. Saunders, R. S. Carter, George A. Howells.

Born, 1882, Olbersdorf, Germany. 1900, Grad. from High School, Frankenstein. 1900-02, Two years and a half practical work in ore and coal mines (apprentice). 1902-04, O. S. School of Mines at Tarnowitz. 1904, Received Degree of Mining Engineer. 1904-05, Mining Engineer with large coal mines in Germany. 1905-06, Sirtaine Rock Drill Co., Germany. 1906-07, A. Beien, Hoisting Engineers, Germany. 1907-10, H. Flottmann & Co., Germany. 1910-12, J. Geo. Leyner Engineering Works Co., Littleton, Colo.

Present position: 1912 to date, Ingersoll-Rand Co., Sales Engineer.

George Fronheiser Stammler, Johnstown, Pa.

Proposed by F. W. Sperr, F. W. McNair, W. E. Hopper.

Born, 1887, Johnstown, Pa. 1905, Johnstown High School. 1908, Mich. College of Mines; B. S. and E. M. 1908-10, Instructor at Mich. College of Mines in the Dept. of Metallurgy and Ore Dressing. 1911, Mine Chemist, Shannon Copper Co., Metcalf, Ariz. 1912, Drafting and Construction work, Mine Mountain, Ltd., Sellwood, Ont. 1912, Chemist, Dominion Nickel Copper Co., Nickelton, Ont.

Present position: 1913 to date, Alloy Chemist, Cambria Steel Co., Johnstown, Pa.

Edwin Goodwin Steele, Dallas, Texas.

Proposed by Dyke V. Keedy, Carl F. Dietz, W. Spencer Hutchinson.

Born, 1874, Schoharie, N. Y. Public Schools; self taught in technical subjects. Mining engineering and ore dressing. Have been engaged for the past 16 years as joint inventor and in perfecting the Sutton, Steele & Steele system of ore concentration and mineral separation, during which time have been actively engaged in mining and mining engineering.

Present position: Secy.-Treas. Sutton, Steele & Steele, of Texas; and Sutton, Steele & Steele Manufacturing, Milling & Mining Co., of Colo.

Walter Livingston Steele, Dallas, Texas.

Proposed by Dyke V. Keedy, Carl F. Dietz, W. Spencer Hutchinson.

Born, 1866, Galveston, Texas. Public Schools, and Southwestern Univ., Georgetown, Texas. Have been engaged for the past 16 years as joint inventor and in perfecting the Sutton, Steele & Steele system of ore concentration and mineral separation, during which time have been actively engaged in mining and mining engineering.

Present position: Vice-President, Sutton, Steele & Steele, of Texas; and Sutton, Steele & Steele Manufacturing, Milling & Mining Co., of Colo.

Henry Moore Sutton, Dallas, Texas.

Proposed by Dyke V. Keedy, Carl F. Dietz, W. Spencer Hutchinson.

Born, 1862, Lexington, Ky. Public Schools; self taught in technical subjects. Chemistry and ore dressing. Have been engaged for the past 16 years as joint inventor and in perfecting the Sutton, Steele & Steele system of ore concentration and mineral separation, during which time have been actively engaged in mining and mining engineering.

Present position: Elec. Engr., Sutton, Steele & Steele, of Texas; and Sutton, Steele & Steele Manufacturing, Milling & Mining Co., of Colo.

John Clarence Taylor, Salt Lake City, Utah.

Proposed by C. W. Goodale, B. H. Dunshee, John Gillie.

Born, 1875, Oxford, Wis. 1893-94, Hamline Univ. 1896-1901, Grad. Univ. of Wisconsin; B. S. 1901-05, Engineer and Salesman, Sullivan Mach. Co. 1905-08, Engineer and Salesman, designing, selling and installing mining, hoisting, and pumping plants with Mine & Smelter Supply Co.

Present position: 1908 to date, District Manager, Denver Rock Drill Mach. Co.

Edward C. Thompson, Kennett, Shasta, Co., Cal.

Proposed by Howard F. Wierum, Redick R. Moore, C. E. Metcalfe, J. H. Kerwin.

Born, 1881, Dunlap, Iowa. 1901, B. S. Columbian Univ. (now George Washington), at Washington, D. C. 1902, M. E. Mechanical from above. 1905, LL. B. from National Law School, Washington, D. C. 1906-07, Furnace work at Humboldt, Ariz., Cons. Ariz. Smelt. Co. 1908 Furnace work at Sasco, Ariz., Southern Ariz. Smelt. Co. 1909-10, In charge of Roasting Dept. at Midvale Plant (Utah), United States Smelting, Refining & Mining Co. 1911-13, Experimental work at Kennett, Cal., Mathmoth Copper Min. Co. 1913, except four months, in charge of the smelter at Needles, Cal., Needles Min. & Smelt. Co. (also U. S. S. R. & M. Co.).

Present position: Assisting with experiments on Hall Process.

Lester S. Thompson, Congo Belge, W. Africa.

Proposed by S. H. Ball, P. B. Oliver, A. M. Peterson.

Born, 1889, New York, N. Y. 1905, General, elementary and preparatory schools. 1907-11, Columbia Univ., School of Mines; E. M. 1905, Practiced surveying. 1906-07, Practical mining, prospecting, etc., Western U. S. 1911, Practical mining, Cripple Creek, Colo. 1912-13, Engineer, Insp. Cons. Copper Co., Miami, Ariz.

Present position: Engineer, Prospector, Companhia de Perquisas, Minas de Angola, West Africa.

Stanley M. Walker, Denver, Colo.

Proposed by T. H. Proske, D. W. Brunton, F. M. Taylor.

Born, 1887, Denver, Colo. 1902-06, Tome School in Maryland. 1907-11, Colorado School of Mines. 1906-07, Teziutlan Copper Co., Estado de Pueblo, Mexico; filling various positions in mine, smelter and office. 1907-11, Summers spent underground in various mines in Colorado. 1911-12, Field and Construction Engineer, Nevada California Power Co., Los Angeles, Cal. 1912-13, Supt., Madonna Mine, at Monarch, Colo., and Gen. Mgr. of Monte Cristo mine, Ouray, Colo.

Present position: 1913 to date, Supt., Frisco Tunnel Co.,

Hugh C. Watson, New Orleans, La.

Proposed by R. B. Stanford, Charles F. Rand. (One more signature required.)

Born, 1878, Ouray, Colo. 1893, Finished Grammar School. 1897, Grad., Denver High School. 1901, Grad., Colorado School of Mines. 1901, Liberty Bell, Telluride, Colo. 1903, San Juancito, Honduras. 1905, Penoles Min. Co., Mapimi, Mex. 1907, American Metal Co., Bolivia, S. A. 1908, Goldfield, Nev. 1911, Mapimi, Mexico. 1912, New Monarch Min. Co., Leadville, Colo. 1913, Piz-Piz, Nicaragua, Central American Goldfields.

Present position: Gen. Supt., Central American Goldfields.

James Watts, Brazil, S. A.

Proposed by G. Chalmers, F. W. P. Chalmers, J. E. Davies.

Born, 1869, Cardiff. General and Technical education in Wales. 1909-12, Mining course with International Correspondence Schools, Scranton, Pa. 1885-90, Cardiff Steam Coal Co. 1890-1914, St. John del Rey Mining Co., Ltd., in various positions, Miner, Prospector with diamond drill, Mine Mechanic, Shift Boss, Asst. Mine Agent; for last six years Chief Mine Agent.

Present position: Chief Mine Agent, St. John del Rey Mining Co., Ltd.

George E. J. Wiessing, Oruro, Bolivia.

Proposed by H. F. Grondijs. (Two signatures required.)

Born, 1884, Amsterdam. 1902-08, Technical High School of Delft (Holland); degree of Mining Engineer. 1909, Six months at the School as assistant of the Professor of Coal Mining. 1909, Sambas (Borneo) as a gold explorer for the Algemeine Exploratie Maatschappij, The Hague, Holland. 1911, Supt., Colorado Mine of the Compania Minera de Oruro, near Oruro.

Present position: Supt. of the Colorado Mine of the Compania Minera de Oruro, near Oruro.

Harry L. Wilson, Jalisco, Mexico.

Proposed by C. Barnes Hoadley, Charles F. Rand, Louis D. Huntoon.

Born, 1877, Seymour, Texas. Preparatory school, Fishburne Military School. Univ. of Virginia; B. S. Mine engineering in the coal fields of West Virginia. Metal mining in Chihuahua, Mex. Cyaniding gold and silver ores in the various plants in Guanajuato, Dos Estrellas, Pachuca.

Present position: Metallurgist, El Favor Mining Co.

Tobias Wolfson, New York, N. Y.

Proposed by J. W. Allen, B. B. Thayer, W. S. Harper.

Born, 1864, Russia. No technical education. 1889, Lewisohn Bros., United Metals Selling Co., and Raritan Copper Works.

Present position: 1912 to date, President, Raritan Copper Works; Vice-Pres., United Metals Selling Co.

Horace G. Young, Cobalt, Ont., Canada.

Proposed by D. L. H. Forbes, R. B. Watson, Samuel W. Cohen.

Born, 1882, Osnabruck Center, Ont. 1898-1902, Preparatory Education in Morrisburg High School, Morrisburg, Ont., Canada. 1902-06, McGill Univ., Montreal, P. Q.; B. S. 1907-08, Salesman, Mussels, Ltd., Montreal, Selling Mining Machinery. 1909, Construction Engineer, King Edward Mine, Cobalt, building a concentrator. 1910-11, Manager, Hudson Bay Mines, Ltd., Cobalt, Ont. 1912-13, Manager, Tretheway Silver Cobalt Mine, Ltd.

Present position: Manager, Tretheway Silver Cobalt Mine, Ltd., Cobalt, Ont.

Associate Members

Harry Barry Byrne, Butte, Mont.

Proposed by Frederick T. Greene, John Gillie, B. H. Dunshee.

Born, 1872, Baltimore, Md. High School education. 1899-1901, Engineer on R. R. Construction. 1901-04, Butte Rep. of Paine Webber & Co., Boston, Mass., bankers and brokers, operating mines in a small way. Director of Corbin Copper Co., Butte and London Mining Co.

Present position: Mgr. Butte, Mont., branch of Paine, Webber & Co.

W. Hincle Smith, Philadelphia, Pa.

Proposed by F. Lynwood Garrison, B. B. Thayer, F. W. Bradley.

Born, 1861, Philadelphia, Pa. 1882, Grad. Towne Scientific School, Univ. of Penn.; B. S. Director of Utah Copper Co., Chino Copper Co., Ray Cons. Copper Co., Nevada Cons. Copper Co.

Present position: Director of above companies.

Daniel A. Welch, Butte, Mont.

Proposed by Frederick T. Greene, John Gillie, B. H. Dunshee.

Born, 1879, Missoula, Mont. Public Schools. 1894-95, College of Mont., Mont. 1896-97, Golden-Sceptre Gold Mining Co.

Present position: 1897 to date, Anaconda Copper Mining Co., International Smelting & Refining Co.

Junior Members

Theo. H. M. Crampton, Golden, Colo.

Proposed by F. W. Traphagen, Harry J. Wolf, Regis Chauvenet.

Born, 1890, New York City. 1904-08, Palmer Inst., Lakemont, N. Y. 1909-14, Colo. School of Mines, Golden, Colo. 1909, Pres. Gold Min. Co., Paris, Kern Co., Cal. 1909, Big Stick Gold Min. Co., Yavapai Co., Ariz. United Verde Copper Co., Smelter Work, Jerome, Ariz. Arizona Power Co., general work, Mayer, Ariz. 1909-10, Arizona Copper Belt Min. Co., Constellation, Ariz. 1911-13, Ceramic Investigation for Colorado Geological Survey; Tutor and Instructor in Colorado School of Mines.

Present position: 1910 to date, Student at Colorado School of Mines, Reports on Coal, Clay, Gold, Copper Properties. Perfection of a New High Temp. Gas Furnace.

Stephen Howard Daley, Jr., So. Bethlehem, Pa.

Proposed by Howard Eckfeldt, Siegfried Fischer, Jr., Joseph W. Richards.

Born, 1890, Hartford, Conn. 1906, Grad. Grammar School. 1910, High School. 1910-12, Trinity College. 1912, Class of 1915, Lehigh Univ. 1910-12, During summer months of the years mentioned, employed in the Engineering Department of the Hartford Electric Light Co.

Present position: Student, Lehigh University.

William Emery, Jr., Elkins Park, Pa.

Proposed by J. F. McClelland, L. W. Bahney, H. L. Wells.

Born, 1891, Williamsport, Pa. 1910-13, Yale Sheffield Scientific School, New Haven, Conn.

Present position: Student in Yale Graduate Mining School.

William August Hamann, Jr., New Haven, Conn.

Proposed by J. F. McClelland, L. W. Bahney, H. L. Wells.

Born, 1891, Mount Vernon, N. Y. 1897-1905, Public Schools, Mount Vernon, N. Y. 1905-06, High School, Mt. Vernon, N. Y. 1906-10, Taft School, Watertown, Conn. 1910-13, Sheffield Scientific School; Ph. B. 1913, Graduate School of Yale Univ.

Present position: Student in Yale Univ.

Marion Lothrop Hanahan, Columbia, S. C.

Proposed by Stephen Taber, Thomas L. Watson, J. S. Grasty.

Born, 1894, Charleston, S. C. 1910, Received diploma from High School of Charleston. 1910-14, Univ. of S. C.; expect to receive bachelor's degree in June, 1914. 1912, Worked in testing laboratory at plant of Standard Portland Cement Co., Leeds, Ala.

Present position: Student and Asst. in the Dept. of Geology, Univ. of South Carolina.

John Bernard Johnson, Golden, Colo.

Proposed by William R. Chedsey, F. W. Traphagen, Regis Chauvenet.

Born, 1891, Wheeling, W. Va. High School and Grade at Wheeling. 1909-10, Washington and Jefferson College. 1910-14, Colo. School of Mines. 1906-07, Summers, at National Tube Co. 1913, Summer, Pure Oil Co., Producers and Refiners.

Present position: Senior, Colorado School of Mines.

Samuel Zettler Krumm, Golden, Colo.

Proposed by F. W. Traphagen, Harry G. Wolf, William R. Chedsey.

Born, 1890, Columbus, Ohio. 1909-10, Student at Ohio State Univ. 1910-11, worked for the U. S. Gypsum Co., in Gypsum, Va., under A. J. Parkhurst (Supt.), as Tester and Mixer Boss. 1911-12, student at the Ohio State Univ. 1912-14, Student at Colorado School of Mines.

Present position: Student at Colo. School of Mines.

Herand Kalost Najarian, New Haven, Conn.

Proposed by L. W. Bahney, J. F. McClelland, C. T. Bradley.

Born, 1889, Marash, Turkey. Grad. Sheffield Scientific School, Yale Univ. (class of 1913). 1913, Engineering Dept. Iron Mines of Witherbee, Sherman & Co., Inc., Mineville, N. Y.

Present position: Grad. Student in Mining Engineering, Yale Univ.

Arthur Fuller Truex, Rolla, Mo.

Proposed by G. H. Cox, H. A. Buehler, C. R. Forbes.

Born, 1886, Newark, O. 1904, Rochester, N. Y., Public Schools. 1904-08, Univ. of Rochester; B. S. Summers, 1905-08, with Rochester Park Commission. 1909-12, Rodman, New York State Eng. Dept. Barge Canal.

Present position: 1912 to date, Student at Missouri School of Mines.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Mar. 10 to Apr. 10, 1914. This list, together with the list published in *Bulletin* No. 88, April, 1914, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1914, and brings it up to the date of April 10, 1914.

AGNEW, WILLIAM C.	218 N. 15th Ave. E., Duluth, Minn.
BERGER, W. F. B.	Colo. Nat'l Bank, Denver, Colo.
BETTS, ANSON G.	Stackhouse, N. C.
BLAIR, A. F.	Box 1830, Seattle, Wash.
BROWNING, WILL C.	Magma Copper Co., Superior, Ariz.
CAIRNS, JOHN M.	Drummond Gardens, Anniesland, Glasgow, Scotland.
CARLETON, JAMES G.	Cia. Huanchaca de Bolivia, Pulacayo, Bolivia.
CARLYLE, E. J.	608 Cedar St., Anaconda, Mont.
CHARLES, W. W.	346 South Ardmore Ave., Los Angeles, Cal.
CHOATE, WAYNE	1002 Hollingsworth Bldg., Los Angeles, Cal.
CLARKE, ROY H.	642 Peyton Blk., Spokane, Wash.
CLELAND, GEO. A.	700 Journal Bldg., Portland, Ore.
COLLINS, A.	Box 144, Seattle, Wash.
CORKILL, EDWARD T.	Copper Cliff, Ont., Canada.
COX, HERBERT B.	Ringwood Co., 50 Church St., New York, N. Y.
COYLE, JOHN A.	418 N. St. Clair St., Pittsburg, Pa.
CRARY, CHARLES N.	Kimberly, Nev.
DELANO, FREDERIC A.	1414 Transportation Bldg., "Dearborn" & Harrison Sts., Chicago, Ill.
DRAPER, CARL H.	Care Merquital Mining Co., Apartado 77, Guadalajara, Jal., Mexico.
DRESSER, CLARENCE G.	R. F. D., Burch, Globe, Ariz.
ELDRID, BYRON E.	50 East 41st St., New York, N. Y.
ELLIOTT, ROY H.	6435 Hillegass Ave., Oakland, Cal.
ELMER, WILL W.	518 N. E. 26th St., Portland, Ore.
EMLAU, H. S.	Spring Lake, Mich.
EMMONS, NEWTON W.	415 Rogers Bldg., Vancouver, B. C., Canada.
EU, S. H.	1 Eu Swee Guat Str., Kuala-Lumpur, F. M. S.
FITCH, MAX B.	603 Central Bldg., Los Angeles, Cal.
FLEMING, EDWARD P.	Harvard Observatory, Cambridge, Mass.
FROEHLING, HENRY.	1800 Hanover Ave., Richmond, Va.
GARD, IRVING R.	Care Mecca Colliery Co., Eagle, W. Va.
GIPPS, F. G. de V.	Mt. Shamrock G. M., via Maryborough, Queensland, Aust.
GREIT, A. J.	Apartado 263, Matanzas, Cuba.
GREENE, FRED TURRELL	316 State Savings Bank Bldg., Butte, Mont.
GREENSFELDER, NELSON S.	721 So. E. St., Tacoma, Wash.
GRIFFITHS, J. NORTON.	3 Central Buildings, Westminster, London, S. W., England.
HAAS, J. C.	Care Michener's Camp, Salesville, Mont.
HAMILTON, HARRY T.	Moctezuma Copper Co., Nacozari, Son., Mexico.
HEROULT, P.	37 Rue Cortambert, Paris, France.

- HUDSON, A. W. P. O. Box 657, Douglas, Ariz.
 IMHOFF, ALEX. Box 104, Battle Mountain, Nev.
 IONIDES, S. A., Min. and Met. Engr. 802 Equitable Bldg., Denver, Colo.
 JOHNSON, C. B. Hotel Marseilles, Broadway & 103d St., New York, N. Y.
 JUDSON, WILBER. Instructed to hold all mail.
 KANO, SHINICHI. Fukuoka Komusho, Fukuoka, Japan.
 KAWAMURA, TAKESHI. Mining Dept., Mitsubishi Co., Tokyo, Japan.
 KEPNER, ROSS B. 353 E. Pearl St., Los Angeles, Cal.
 KIMBALL, E. B. 284 Mountain Ave., Piedmont, Alameda Co., Cal.
 KURIE, F. M. Key West Apartment, Los Angeles, Cal.
 LAVERY, V. M. Box 85, Llanerch, Pa.
 LAWRIE, FRANK C. Nanchital, Puerto, Mexico. Ver.
 LAWTON, N. O. Lawton, Mich.
 LEPLNE, EDWARD F. 406 W. Onondaga St., Syracuse, N. Y.
 LOUX, CARL H. Care Ottawa Mine, Slocan City, B. C., Canada.
 MALCOMSON, A. B., JR. 2435 Morris Ave., New York, N. Y.
 MANN, WILLIAM SEWARD. 910 Donaldson Bldg., Minneapolis, Minn.
 MANNING, J. F. Holkol, Whang Hai Prov., Chosen, Korea.
 MARSH, R. 1532 West 7th Ave., Seattle, Wash.
 MARSH, ROBERT, JR. Care E. Bennett, 165 Broadway, New York, N. Y.
 MARSHALL, S. M., Asst. General Manager, A. J. Haws & Sons, Ltd., Johnstown, Pa.
 METCALF, FRANK A. Box 168, Juneau, Alaska.
 MINSCH, W. C. Cinco Mines Co., Magdalena, Jal., Mexico.
 MORRISON, WALTER I. Marine Bank Bldg., Erie, Pa.
 MOTTER, W. D. B., JR., Benson Mines Co., Benson Mines, St. Lawrence Co., N. Y.
 NAGEL, FRANK J. No. 3 Fisk Apts., El Paso, Texas.
 NICHOLS, H. W. Field Museum of Natural History, Chicago, Ill.
 NICHOLSON, FRANK, Care London City & Midland Bank, Ltd.,
 5 Threadneedle St., London, E. C., England.
 OF, CHARLES. Cumanayagua, via Cienfuegos, Cuba.
 PAYMAL, GEORGE W. 1621 Vallejo St., San Francisco, Cal.
 PLATT, EDWIN H. 932 Equitable Bldg., Denver, Colo.
 PORTUGAL, JOSEPH H. Swarthmore Lodge, Bryn Mawr, Pa.
 PRENTICE, RUSSELL. P. O. Box 580, Vancouver, B. C.
 REECE, P. P. Melcher, Marion Co., Iowa.
 RICE, JOHN A. Silver Center, Ont., Canada.
 ROBERTSON, J. D. 1403 Syndicate Trust Bldg., St. Louis, Mo.
 ROBINSON, HARRY O. Calle Madrid 15, Mexico City, Mexico.
 SANDOVAL, P. P. O. Box B, Nogales, Ariz.
 SCHOFIELD, W. D. Aspen, Colo.
 SHIRAS, WM. MCGILVRAY. 5746 Aylesboro Ave., Pittsburg, Pa.
 SLATER, J. L., JR. 1064 Gilbert St., San Francisco, Cal.
 SMITH, C. ALBERT, "Forminiere-Kasai Diamonds," Kinchasa, Congo Belge,
 West Africa.
 SMITH, E. PERCY. Wallace, Idaho.
 SMYSER, A. E. 239 So. Fairmount St., E. E., Pittsburg, Pa.
 SPERRY, R. S. Douglas, Ariz.
 SPROAT, A. D. Care Guanajuato Cons. Min. & Mill. Co., Guanajuato, Gto., Mexico.
 STEEL, A. A. Parma, Ida., Route 1.
 STEWART, J. B. The American House, Bordentown, N. J.
 STINES, NORMAN C., Polevskoj savod. St. Mramorskaja, Government of Perm, Russia.
 STRAUSS, LESTER W. Casilla 1174, Valparaiso, Chile, So. America.
 STRAW, CHARLES A. No. 1 Willard Place, Hudson, N. Y.
 STROHECKER, J. W. Garfield, Wash.
 TAYLOR, JAMES. "Cartref" Brierley St., Mosman, N. S. W., Aust.
 TAYLOR, S. A. 506 2nd National Bank Bldg., Pittsburg, Pa.
 THOMAS, KIRBY. 43 Exchange Place, New York, N. Y.
 THROPP, J. E., JR. Care Dr. Wm. M. Irvine, Mercersberg, Franklin Co., Pa.
 TONKIN, S. J. 614 S. 2d St., McAlester, Okla.
 TORRES, F. L. 3019 Tulorosa St., El Paso, Texas.
 TRISCHKA, CARL. P. O. Box 2322, Bisbee, Ariz.
 UNDERWOOD, A. J. Bland, N. M.
 WADSWORTH, M. E., Dean, Prof. of Geol., etc, School of Mines,
 Univ. of Pittsburg, Pittsburg, Pa.
 WAITE, HENRY M. City Hall, Dayton, Ohio.
 WATT, ARTHUR P. St. Louis Smelting & Refining Co., St. Francois, Mo.

WETHEY, A. H.	Lemoore, Kings Co., Cal.
WHITE, R. T.	Care Caucasus Copper Co., Ltd., Dzansoul, Russia.
WILLIAMS, RALPH B.	89 Federal St., Salem, Mass.
WILSON, J. BOWIE.	Wentworth Court, Elizabeth St., Sydney, N. S. W., Aust.
WORTH, JOHN G.	Sante Fe, N. M.
YANG, CHO.	Hanyang Iron & Steel Wks., Hanyang, China.
YEANDLE, WILLIAM H., JR.	Care Guillermo Brockmann, 2a Capuchinas No. 55, Mexico City, Mexico.
ZELLER, H. P.	P. O. Box 121, Youngstown, Ohio.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name.	Last Address of Record, from which Mail has been Returned.
ADAMS, RALPH E.	Amer. Smelters Securities Bldg., Velardena, Mexico.
AUER, CHARLES I.	Estacion de Pedrisena, Dur., Mexico.
BRYCE, R. A.	Ross M. M. Co., Silverton, Colo.
CALDWELL, F. B.	Candelaria Cons. Mex. Mine Co., San Dimas, Dur., Mexico.
CLAUDET, H. H.	Rossland, B. C., Canada.
COOK, EDWARD H.	Berimoo, via Bonalas, Dur., Mexico.
DISBROW, W. F.	1108 Foster Bldg., Denver, Colo.
GOETTER, FREDERICK B.	Mexican Candelaria Co., San Dimas, Dur., Mexico.
LAVERY, V. M.	Care S. A. Stephens, Real Estate Trust Bldg., Philadelphia, Pa.
LAWRENCE, T. J.	Topia, Dur., Mexico.
MARSTON, P. L.	P. O. Box 826, Tonopah, Nev.
RUBEL, MILTON L.	Pilones Min. Co., Topia, Dur., Mexico.
SCOBAY, J. C.	3424 Colfax Ave. B., Denver, Colo.
SIMONDS, E. H.	1108 Crocker Bldg., San Francisco, Cal.
SMITH, F. G. D., Mgr.,	Santo Domingo Silver Min. Co., Batopilas, Chih., Mexico.
TRUAX, SEWALL.	Highland Park, Ill.
WALLACE, ROBERT.	Lovelock, Nev.
WENIGER, RUDOLF.	Ojuelo, Dur., Mexico.
WHEELWRIGHT, O. W.	Madison, Wis.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period March 10 to April 10, 1914:

Date of Election.	Name.	Date of Decease.
1912	*Barager, George W.	January —, 1914.
1883	*Patch, Maurice B.	December 3, 1913.
1913	*Ranft, William Q.	March 12, 1914.
1886	*Westinghouse, George	March 12, 1914.

BIOGRAPHICAL NOTICE

Frederick Elwell Woodbury, Vice-President of the Newport Mining Co., was killed by an accident at Ironwood, Mich., on Jan. 21, 1914.

Mr. Woodbury, the only son of Leander S. Woodbury and Mary Davis Woodbury, was born at Nashua, N. H., Apr. 7, 1862. In 1870 his family moved to Burlington, Vt., where he attended the public schools, graduating with honors from the High School in 1880.

After a year in the Civil Service he went to the Lake Superior copper country, where his father was then engaged as Superintendent of Motive Power for the Calumet & Hecla Mining Co.

Mr. Woodbury entered the service of the same company in the Civil Engineering Department, remaining there until 1884. He then spent two years as a special student at the Massachusetts Institute of Tech-

* Member.

nology. Returning to Calumet, he was employed as Assistant Mining Engineer by the Calumet & Hecla for about two years, when he was transferred to their smelting works as Chemist, where he remained until 1890. From 1890 to 1892 he was employed by Ferdinand Schlesinger, of Milwaukee, as Mining Engineer at the Chapin Mine, Iron Mountain, Mich. He then returned to the Calumet & Hecla, where for about four years he did special work as an expert in milling and concentrating.

From 1896 to 1898 he was again associated with Mr. Schlesinger as Mine Manager at Ironwood. In 1898 he accepted a position as expert in charge of experiments at the Calumet & Hecla Stamp Mills. His work there resulted in greatly modifying and improving milling practice in the Lake Superior copper country. The Woodbury system of concentration was developed during this period; by this system, unlike previous practice and in opposition to the then accepted theories, the mixed sizes of ores are treated, thus largely eliminating the preliminary sizing. For this work he designed the Woodbury slime classifier, middling and bull jigs, now very largely used throughout the country. The Woodbury mortar-discharge jig, an ingenious device for removing large pieces of native copper from the stamp mortars, is among his valuable contributions to the development of the copper industry.

In 1900 Mr. Woodbury finally severed his connection with the Calumet & Hecla and renewed his connection with Mr. Schlesinger, which continued until his death. During these years he was technical adviser in all the various Schlesinger enterprises. He took an active part in the development of the Newport mine. He was an important factor also in the development of the coke-oven plant of the Milwaukee Coke & Gas Co. from the time of its inception. Perhaps the most important work which he undertook in this connection was the design and installation of a plant for the preparation and mechanical loading of foundry coke into box cars. The successful solution of the problems met with was specially difficult, due, partly, to local conditions, and because of Mr. Woodbury's determination to improve upon all plants which up to that time had been in use for the purpose and which, in his opinion, only partly accomplished the desired results. After much experimental work, it was clear to him that it would be necessary to abandon practically everything which had been used and to begin anew. The development work covered several years, during which time many unexpected obstacles were encountered which would have spelled failure to a man of less determination. However, in accepting each new, unsatisfactory development in the light of a lesson, success finally attended his efforts, and he gained for himself the reputation of having developed the most complete mechanical arrangement for preparing and loading foundry coke in the United States.

He made careful investigations of many mining properties in this country, Canada, and Mexico, for which work he was especially well fitted. Under his supervision preliminary plans for the development of a coal property in Kentucky had recently been completed.

Mr. Woodbury was married Dec. 29, 1887, to Helen Grant Duncan, who survives him.

Mr. Woodbury's personality was well described by one who knew him intimately for many years: "He was, as a young man, intellectually keen, with a zest for life, yet generous, considerate, and of a peculiarly clean mind. The passing years with their responsibilities and trials

seem to have developed the strong fiber of this man without producing anything of harshness. His character was marked by a most unusual blending of gentleness and strength. The world needs such men and his untimely loss is a great one; still he lived much and well, in the best sense, in his allotted time. He did much more both professionally and personally for his generation than is given to most men to accomplish in many more years. Those who knew him best most fully appreciate his worth."

Mr. Woodbury became a member of the Institute in 1904.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

LOUIS D. HUNTOON, *Chairman*.
 ARTHUR S. DWIGHT, *Vice-Chairman*.
 THOMAS T. READ, *Secretary*, Woolworth Bldg., New York, N. Y.
 E. MALTBY SHIPP, *Treasurer*.
 GEORGE F. KUNZ, W. D^r L. BENEDICT.

Boston

ROBERT H. RICHARDS, *Chairman*.
 ALBERT SAUVEUR, *Vice-Chairman*.
 AUGUSTUS H. EUSTIS, *Secretary-Treasurer*, 131 State St., Boston, Mass.
 TIMOTHY W. SPRAGUE, HENRY A. WENTWORTH.

Columbia

F. A. THOMSON, *Chairman*.
 GEORGE W. RODDEWIG, *Vice-Chairman*.
 LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
 R. S. McCAFFERY, S. H. RICHARDSON.

Puget Sound

JOSEPH DANIELS, *Chairman*.
 J. N. POTT, *Vice-Chairman*.
 I. F. LAUCKS, *Secretary-Treasurer*, 95 Yealer Way, Seattle, Wash.
 A. F. BLAIR, CHESTER F. LEE.

Southern California

THEODORE B. COMSTOCK, *Chairman*.
 SEELEY W. MUDD, *Vice-Chairman*.
 FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 300 Germain Bldg., Los Angeles, Cal.
 C. COLCOCK JONES, FRANK ROBBINS.

Colorado

FRANK BULKLEY, *Chairman*.
 C. LORIMER COLBURN, *Secretary-Treasurer*, 614 Ideal Bldg., Denver, Colo.
 S. A. IONIDES, JAMES M. McCLAVE, DAVID G. MILLER.

Montana

E. P. MATHEWSON, *Chairman*.
 FRANK M. SMITH, *Vice-Chairman*.
 D. C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.
 JAMES L. BRUCE, OSCAR ROHN.

San Francisco

S. B. CHRISTY, *Chairman*.
 H. C. HOOVER, *Vice-Chairman*.
 ABBOT A. HANKS, *Secretary-Treasurer*, 630 Sacramento St., San Francisco, Cal.
 F. W. BRADLEY, C. W. MERRILL.

Pennsylvania Anthracite Section

R. V. NORRIS, *Chairman*.
 W. J. RICHARDS, CHARLES F. HUBER,
 RUFUS J. FOSTER, DOUGLAS BUNTING,
 EDWIN LUDLOW, FRANK A. HILL,
 CHARLES ENZIAN, JOHN M. HUMPHREY.

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Turbo Blowers for Blast-Furnace Blowing

BY RICHARD H. RICE, WEST LYNN, MASS.

TURBO blowers for blast-furnace blowing have now been in use for some years, and a review of the experience gained and the present state of progress may be interesting.

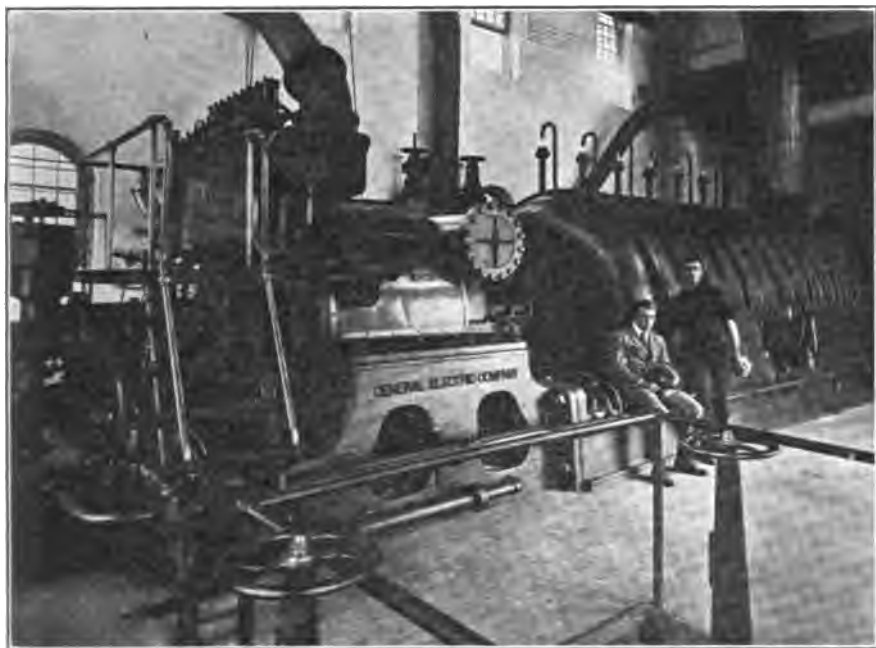


FIG. 1.—STEAM TURBINE DIRECT CONNECTED TO AIR COMPRESSOR.

The first blast-furnace blowing engine of this type to be installed in America was built by the General Electric Co. and is installed at the Oxford Furnace of the Empire Iron & Steel Co. This machine was put into operation March 8, 1910. A duplicate unit is situated at the furnace of the Northern Iron Co. at Port Henry, N. Y., which unit was put into operation June 30, 1910 (Fig. 1).

These two units consist of six-stage compressors driven by four-stage Curtis turbines, operating at variable speeds depending on furnace condi-

tions, the maximum speed being 1,950 rev. per minute. The rated volume is 22,500 cu. ft. of actual free air per minute.

Since then, besides these units, there have been built, installed, and put in operation by the General Electric Co., two 25,000 and four 45,000 cu. ft. machines; and by other builders, one 20,000, two 40,000, and one 50,000 cu. ft. machines.

Therefore, the total number of blast-furnace blowing engines of this type on which experience is available is 12.

Besides these, the General Electric Co. has built, installed, and has in operation 23 multi-stage turbo-driven air and gas compressors.

Method of Rating—Air Measurement

At the outset a radical difficulty presented itself. The rating of a reciprocating steam or gas blowing engine is usually based on displacement and this method is usually the basis of the determination (for practical purposes) of the amount of air being blown at any time. For the turbo blower, actual measurement of the air is the only criterion. If any other standard is used, an assumption is at once necessary as to the actual relation of the standard to the facts. The ratio between displacement and actual air delivered varies with many factors, as follows:

Clearance space in air cylinders (as pressure of air delivered increases more air is necessary to fill this space, decreasing air actually delivered).

Leakage past air pistons (increases with increase of air pressure).

Leakage past inlet and discharge valves (tends to increase with increase of air pressure).

Insufficient area of inlet valves (prevents cylinder from filling on suction stroke).

High temperature of air at beginning of compression stroke.

Insufficient area of discharge valves (prevents cylinder from completely discharging).

Waves in discharge pipe (same result as insufficient discharge valves).

Inertia of valves (has same general effect as insufficient valve area—in addition causes slip or back blow on return stroke).

Improper timing of mechanically driven valves.

From these causes, the actual air delivered may and often does amount to only 70 or 80 per cent. of the displacement.

On account of these discrepancies between rated and actual operating conditions, the first turbo blowers installed were found to be too large for the work. They were rated and proportioned to deliver normally 22,500 cu. ft. of free air per minute at 15 lb. normal or average pressure, with a maximum pressure capacity of 30 lb. As a matter of fact neither engine has ever blown over 18,000 cu. ft. per minute.

It was soon found that the steadiness of blast and perfect control which are the prime characteristics of this apparatus resulted in an increased output from the furnace and an improved quality; and these facts

have been noted in all similar installations where comparative data or experience can be had. The chief difficulty is to prevent overblowing of the furnace, since it is difficult for furnace men to think in terms of the new rating, and until they do, the furnace is always supplied with too much wind.

Difficulties Experienced

A number of difficulties have been experienced with these machines which are worth cataloguing.

The first difficulty met with was pulsation, or fluctuation of pressure, which occurs at volumes of say one-half normal rating and less (varying with the blast pressure). This pulsation consists of pressure fluctua-

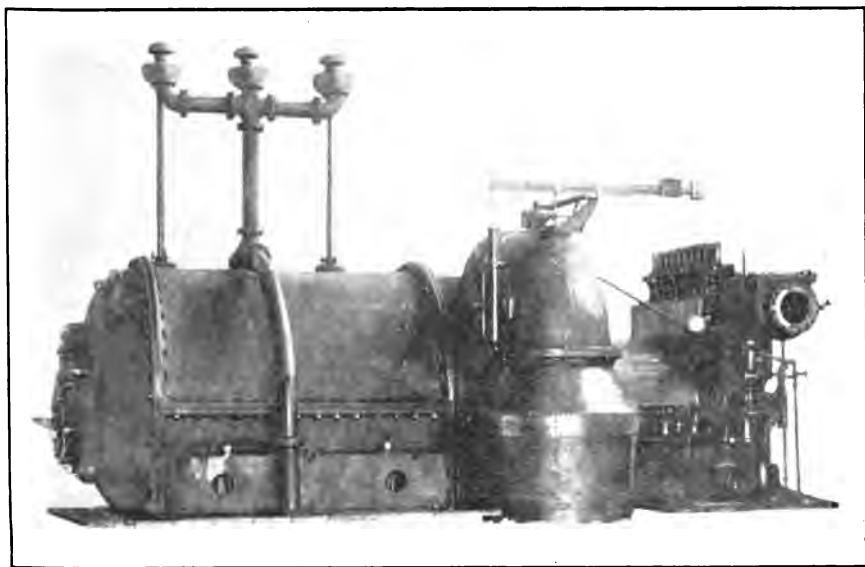


FIG. 2.—CENTRIFUGAL COMPRESSOR DIRECT CONNECTED TO STEAM TURBINE.

tions or waves of one or more pounds magnitude, and having a wavelength period of from one-half to one minute, depending on the volume of blast main and stoves which may be connected to it. This phenomenon has been entirely overcome in the machines which I have designed by the use of a blast gate (butterfly valve) in the suction pipe, between the air governor and the compressor. It is found that a very moderate throttling by this blast gate is sufficient to eliminate pulsation entirely (Fig. 2).

A difficulty met with in the first units built consisted in expansion of the bores in the hubs of the impeller wheels at maximum speeds, due to

insufficient allowance for forcing fits.¹ A mere statement of the difficulty indicates the proper remedy, which when applied entirely eliminated the difficulty.

Some of the turbines used have given trouble by throwing off buckets. This difficulty was finally traced to steam impulses corresponding to the free period of the buckets, and was eliminated by a change in the steam nozzles.

Southern furnaces, using leaner ores, which are besides of a sticky nature, and also using rather poorer cokes than their Northern prototypes, require more wind, and a higher pressure of blast. This severer service has brought out a weakness in impeller design. The entrance portion of some impellers, although of a stiff and rigid shape, has been found subject to cracks and breakage due to reed-like vibrations in the air currents, and these parts have been greatly strengthened and stiffened.

An accident to the turbine of a 50,000 cu. ft. compressor at Birmingham, due to the bursting of a turbine wheel, has been clearly proved to have been due to cracks produced in heat treatment, and not due in any way to defective design or inherent weakness. This machine will shortly be re-installed with all possibility of similar difficulty removed.

Relative Steadiness of Blast

I have frequently asserted that the blast from a turbo blower was appreciably steadier than that from a reciprocating machine, but this statement has often been denied by blast furnace men, who have contended that the pulsations due to intermittent discharge from air tubs were smoothed out by the frictional resistances of blast main and stoves.

In order to set this question definitely at rest a delicate measuring instrument devoid of inertia was devised and constructed and records of pressure conditions at the tuyères of a blast furnace were taken. Fig. 3 shows diagrammatically the apparatus which was used in making these tests.

The object of the apparatus is to show small variations on top of a heavy pressure (18 to 20 lb.) at the tuyère; it being well known that the ordinary instruments for measuring pressures of this magnitude have so much inertia, and are so far from being delicate, that all the variations, which it was the object of these tests to measure, are dampened out. The apparatus used is a differential recording gauge, consisting of a pressure diaphragm in a closed chamber, one side of which can be subjected to a constant pressure of desired amount, and the other subjected to the variable pressure which it is desired to measure, by means of a large pipe

¹ Sanford A. Moss: Increase of Bore of High Speed Wheels by Centrifugal Stresses, *Transactions of the American Society of Mechanical Engineers*, vol. xxxiv, p. 895 (1912).

capable of instantaneously transmitting the fluctuations. Motions of the diaphragm are transmitted to a delicate pen through a mechanism which is devoid of inertia, and the pen makes a record on a rapidly moving band of paper.

The diaphragm used was a thin copper disk about 14 in. in diameter, motions of which were taken from the center through a very light rod

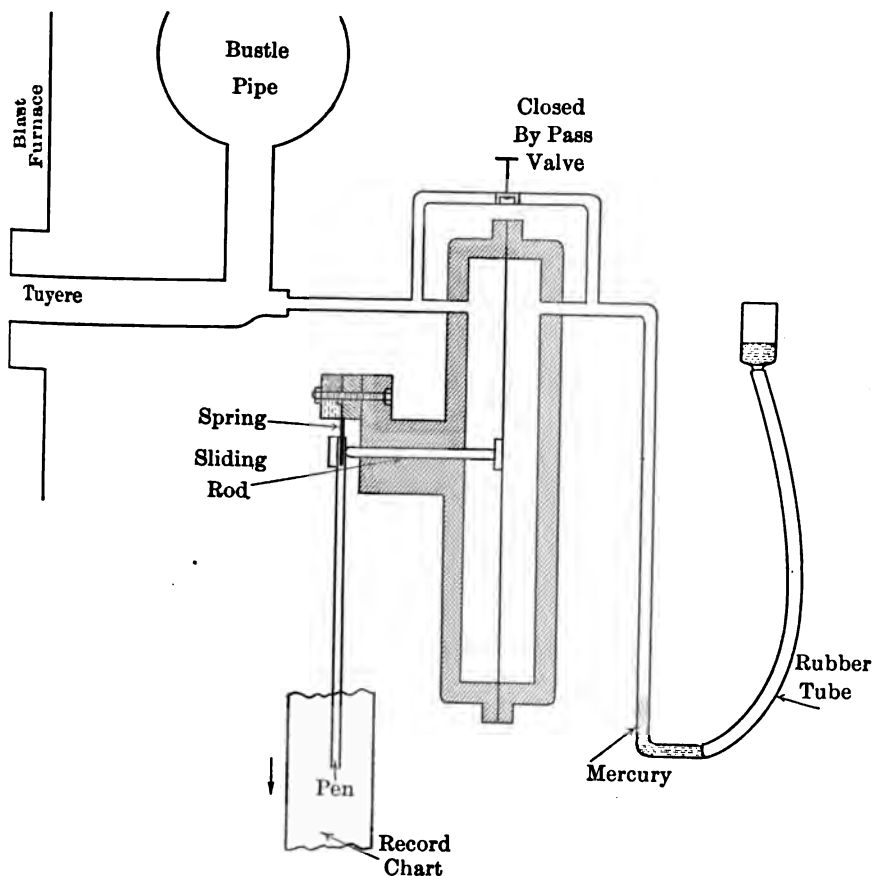


FIG. 3.—DIFFERENTIAL RECORDING PRESSURE GAUGE FOR MEASUREMENT OF PRESSURE VARIATIONS IN BLAST-FURNACE TUYÈRE.

sliding freely through a close-fitting bushing without appreciable leakage. This rod was pressed between the diaphragm on one end and the pen lever on the other, the fulcrum of the pen lever being a flat steel spring, so arranged as to press on the rod and through it upon the diaphragm, even with the greatest pressure fluctuation. The chamber on the back of the diaphragm was connected through a pipe of considerable size to a mercury U, consisting of a rubber tube, on the top of which was a reservoir for

mercury. The back and front sides of the diaphragm were connected by means of a by-pass pipe and valve, and the apparatus was operated as follows:

The by-pass valve was opened, connecting the two sides of diaphragm and equalizing the pressure. As soon as the pressure had been equalized, the by-pass valve was closed, and the pressure on the back of the dia-

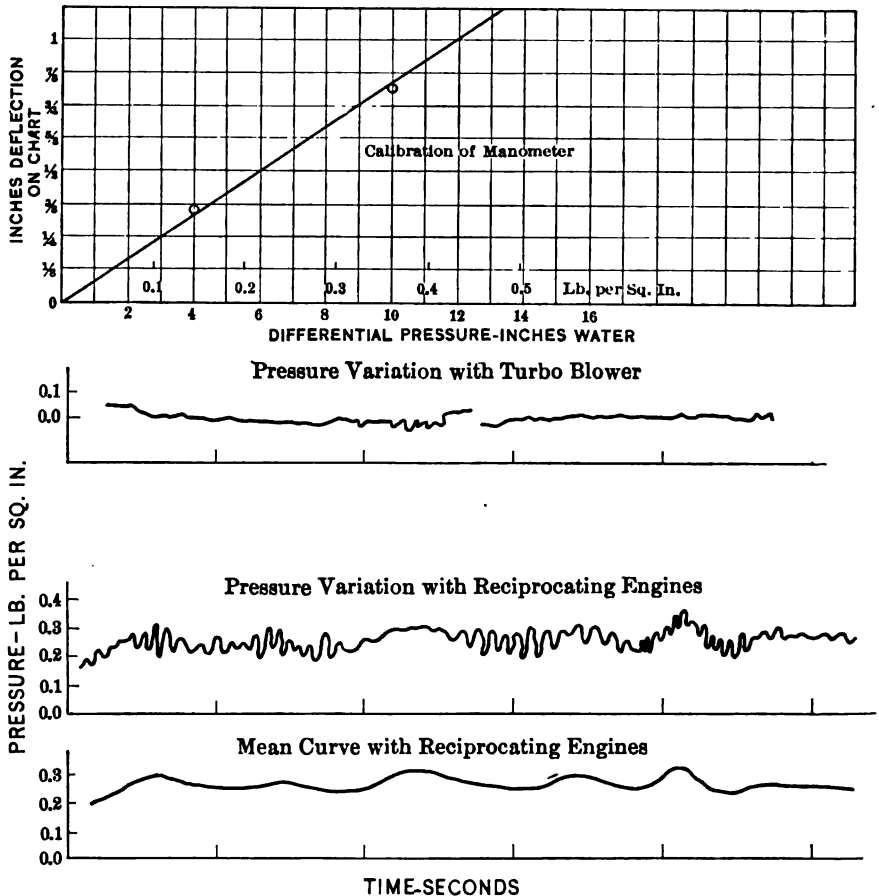


FIG. 4.—CURVES OF PRESSURE VARIATION IN BLAST-FURNACE TUYÈRE.

phragm was adjusted by raising or lowering the mercury reservoir. As soon as this adjustment was complete the mercury reservoir was clamped in position. It was necessary that the by-pass valve, piping, and mercury tube be absolutely tight, and that the pipes containing the mercury be so large that the motion of the diaphragm did not appreciably change the capacity on the constant pressure side of the system.

The motion of the diaphragm was very small, the tension of the

pen spring on the diaphragm was considerable, and the inertia of the moving parts was as small as possible, resulting in a moving system, which was, to all intents and purposes, dead beat.

When operating the apparatus the chart was moved past the pen by hand, and the time interval was marked on the chart by an observer using a stop watch, marks being made at every 5 sec.

The records shown on Fig. 4 were made on a furnace working on Southern iron, and at the time it was blown with reciprocating engines there were four engines in use, two of which had steam cylinders 84 in. in diameter and air cylinders 84 by 60 in. stroke, while two had steam cylinders 44 in. in diameter and air cylinders 84 by 60 in. stroke. All were of the vertical long crosshead type, equipped with Corliss air-inlet valves, and modified Reynolds type discharge valves, and were operated at an average speed of about 40 rev. per minute, the total displacement being about 60,000 cu. ft. of air per minute.

It will be noted that the curve obtained when operating in this manner is complex and apparently consists of a primary wave, and a secondary wave. Below the observed curve has been drawn one showing the primary wave, produced by tracing through the mean points of the secondary waves. These primary waves may be due to the discharge of the reciprocating cylinders, while the secondary waves may be caused by the fluttering of the discharge valves. In contrast to these is shown a curve which shows the pressure variations observed with the turbo blower.

Between the bustle pipe, where the observations were taken, and the engines, was a blast main about 1,000 ft. long and of large diameter, as well as the usual stoves. The use of this long and large blast main and four engines resulted in smaller fluctuations than would have been experienced with shorter blast main and with only one or two engines in use. It is hoped at a later period to make pressure observations of this arrangement also.

The method of calibrating the instrument was to produce deflection on the diaphragm by admission of air pressure to one side thereof so as to give a constant pressure difference, which was measured by a water U-tube, noting the deflection of the pen corresponding to these pressures, the chart given in Fig. 4 showing the result of this calibration.

Advantage of Steady Blast

It is believed that the superior steadiness of blast pressure from the turbo blower results in an increase of production, because the width of the melting zone is decreased, while the loss of fuel and ore due to the dancing of the particles on the blast and consequent attrition and blowing away as dust of such particles is reduced. That this is so is evidenced by the

reduction of the amount of dust caught by the dust catchers, which has been found in all such installations to be a very considerable percentage.

Since the width of the melting zone is reduced and all conditions of blast made more uniform, the blast can be used at a higher temperature.

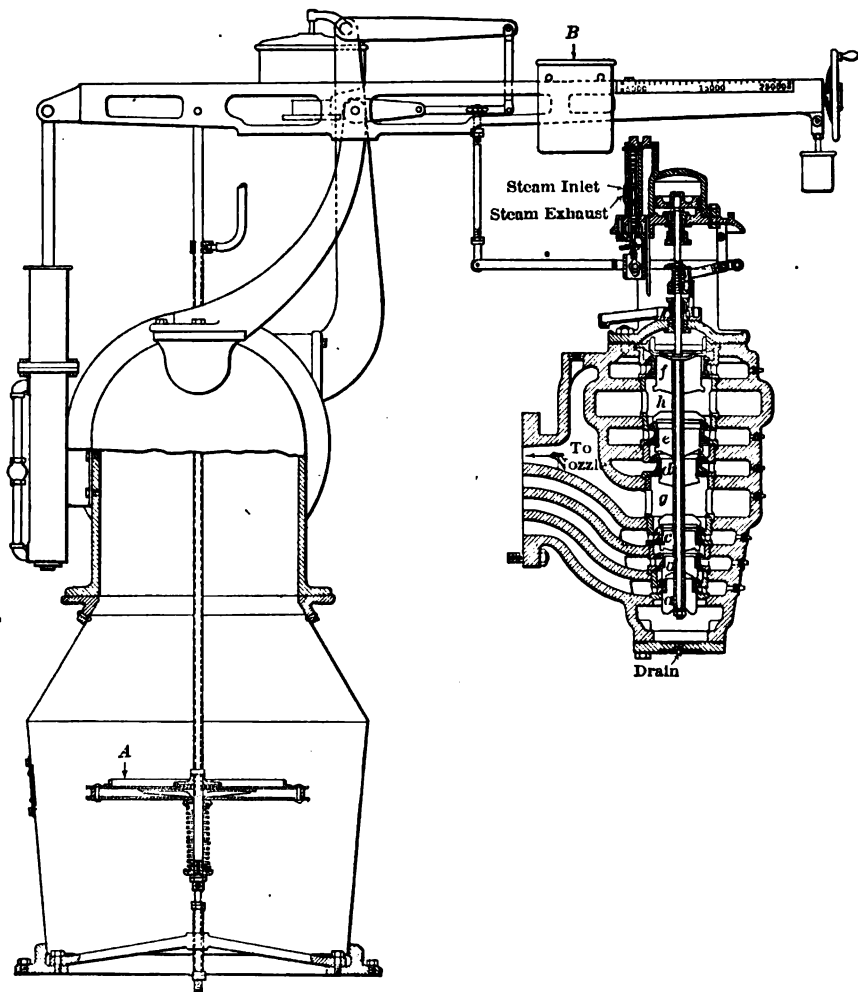


FIG. 5.—CONSTANT VOLUME GOVERNOR FOR CENTRIFUGAL COMPRESSORS.

In practice, the use of turbo blowers, properly governed to measure the rate of blowing with accuracy, has the result of appreciably increasing the amount of iron produced, and this with a more uniform quality. This increase of production is accomplished without increase in the total coke charged and therefore the coke ratio is reduced.

There is evidence that this increase of production and decrease of coke ratio may amount to 10 per cent.

Governing Air Flow

The requirements for a governor to control properly the rate of blowing are ample power and accuracy.

The method used in apparatus built by the General Electric Co. is shown in Fig. 5. It consists of a disk, *A*, riding on the ingoing air

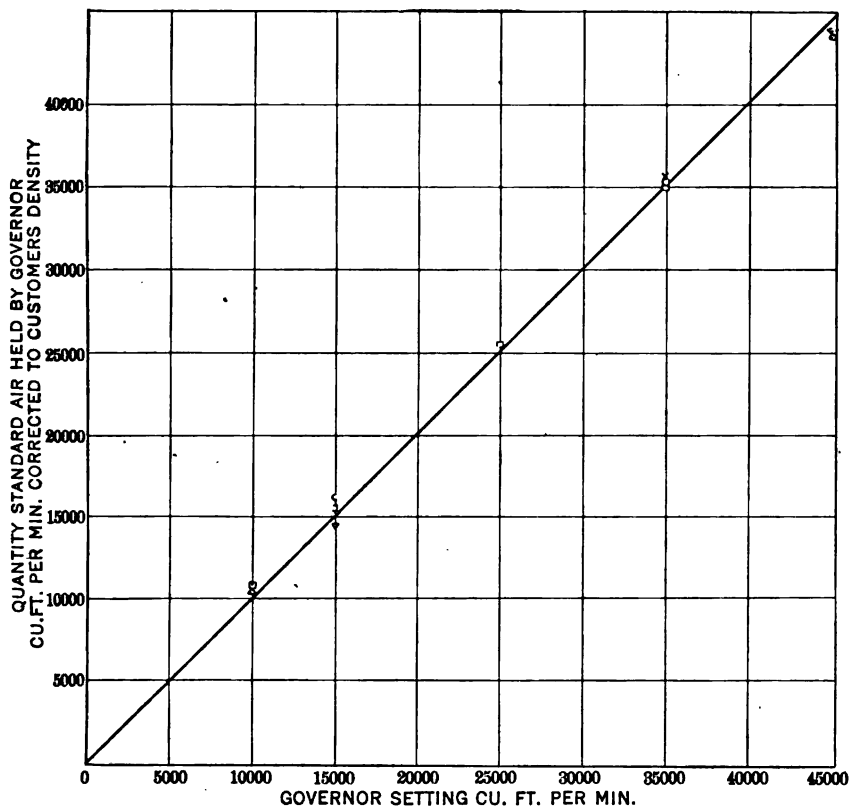


FIG. 6.—CALIBRATION CURVE OF CONSTANT VOLUME GOVERNOR.

in the center of a frustro-conical portion of the inlet pipe. Movements of the disk are transmitted direct to the valve gear, and ample power is secured by the use of very moderate dimensions. The angularity of the walls of the conical pipe is determined by the desired accuracy of regulation. No packing is necessary and the device is therefore practically frictionless and wear-proof.

Differing rates of flow are secured by moving a weight, *B*, along a scale

beam, which is graduated in cubic feet of free air per minute, and carefully calibrated at the factory before shipment. Fig. 6 gives a calibration curve of such a governor and shows the accuracy of the indications.

General Design of Turbo Blowers

Several methods have been used to balance end thrust in the air compressor. The method which I have used is to make all impellers double-entrance, without shroud rings, as shown in Fig. 7. While this method somewhat complicates the coring of the castings, it results in the maximum simplicity and durability of the rotor and eliminates any necessity for the use of a balancing piston, while the clearance packings

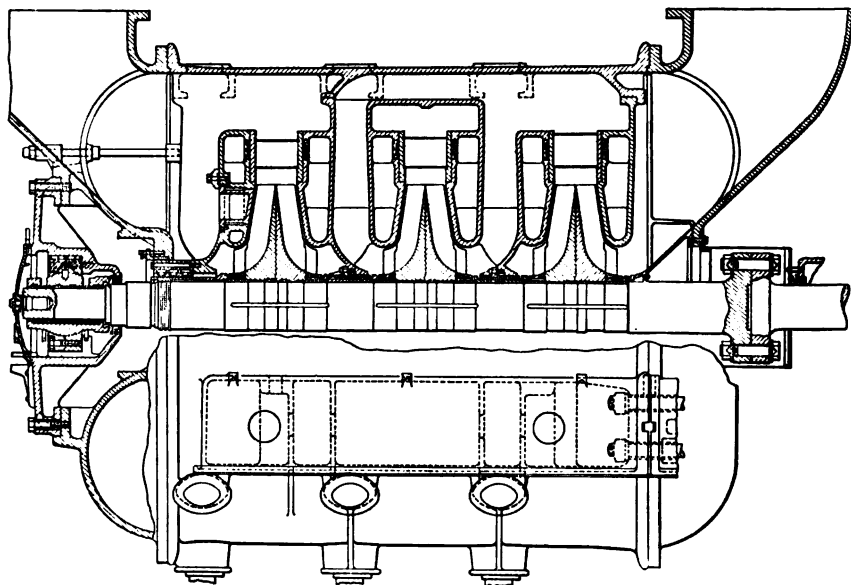


FIG. 7.—THREE-STAGE COMPRESSOR.

are on the shaft diameter and hence may have a large clearance with small leakage. Water jacketing is used in the diaphragms so as to cool the air while undergoing compression and in its passage from stage to stage.

Complete accessibility is secured by splitting casings and diaphragms on a horizontal plane. The turbine is similarly treated.

The locating bearing for the entire system of turbine and compressor is at the turbine end, and is so constructed that the axial position of the shaft with its turbine wheels and air impellers may be changed while the unit is in operation. (Fig. 8.)

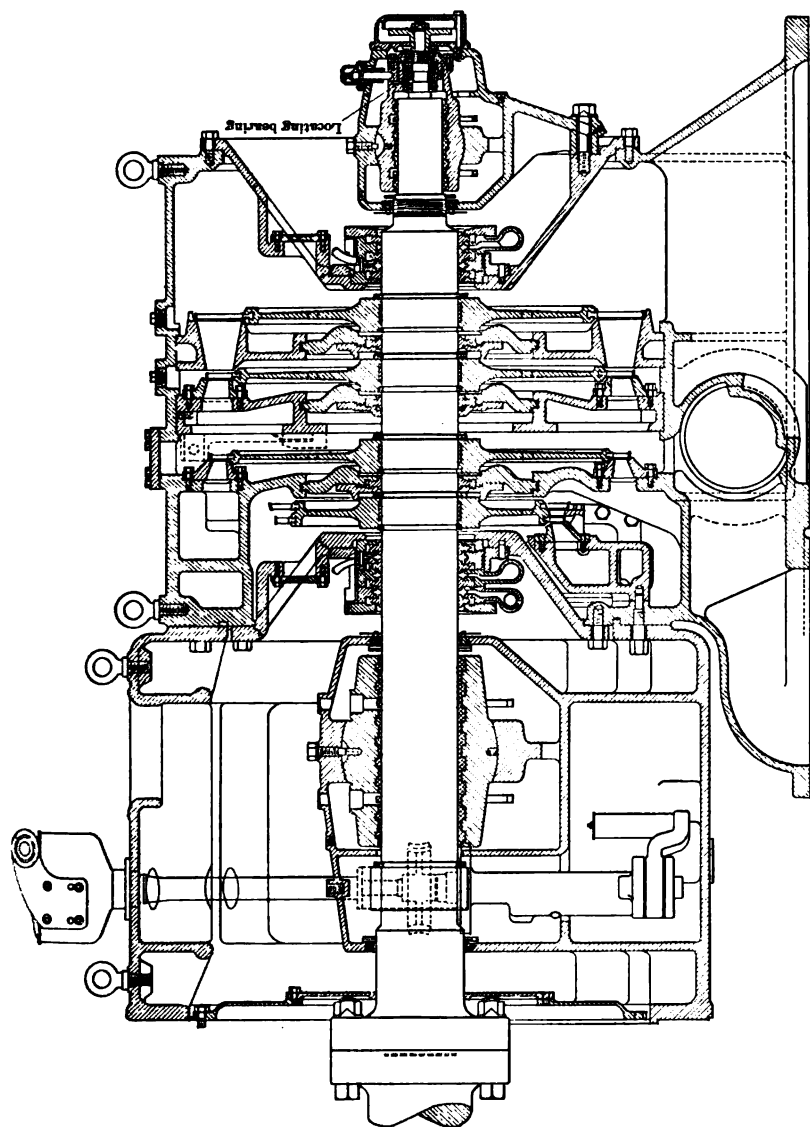


FIG. 8.—5,000-H.P. RIGID-FRAME STEAM TURBINE.

Comparative Efficiency

The relative thermal efficiency of this apparatus as contrasted with other forms of blowing engines is somewhat difficult to determine owing to lack of data. The reciprocating steam blowing engine has never been tested under circumstances which give any reliable basis for comparison, owing to the difficulty, amounting to impossibility, of accurately measuring the actual amount of air delivered per unit of time. The same thing holds true to a greater extent for the gas engine, since it is not only necessary to measure accurately the amount of air delivered for

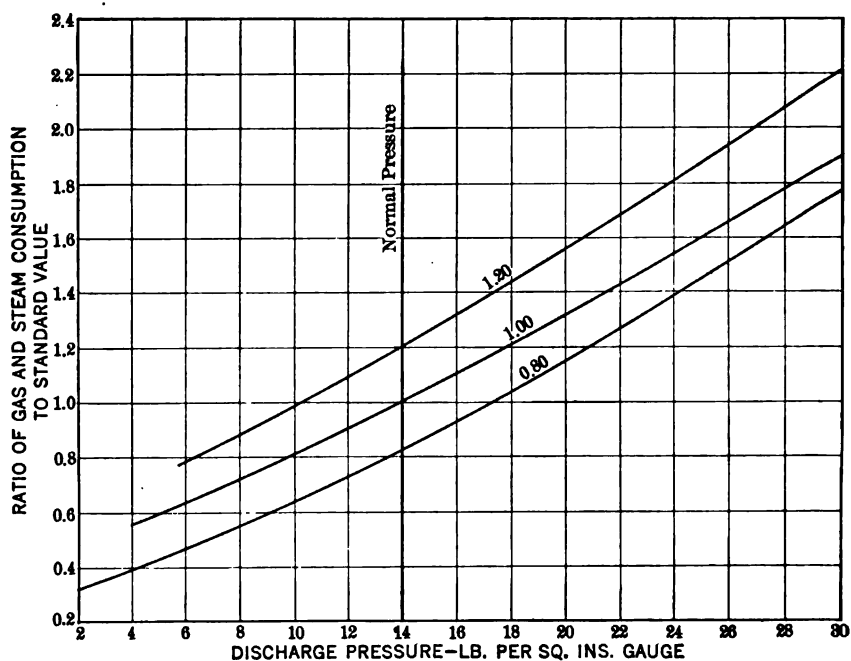


FIG. 9.—GAS AND STEAM CONSUMPTION OF TURBO COMPRESSOR WITH CONSTANT VOLUME GOVERNOR.

Figures on curves show fraction of rated quantity.

any length of time, but also the gas consumption. Both these measurements are subject to the same error. The source of this error is the fact that in all known methods of air metering, the indications vary as the square of the velocity of flow, and hence the mean of the readings is not the mean rate of flow. For this reason the readings of Venturi meters, Pitot tubes, etc., may give rates of flow as high as 40 per cent. in excess of the true value, depending on the amount and character of the fluctuations. We have, therefore, to fall back upon assumptions as to the ratio

of actual to displacement air, which are usually, due to this source of error, too favorable to the reciprocating engine.

From other data we can determine the comparative efficiency of turbine and steam or gas engine. These data are derived from tests of similar machines driving electric generators, on which much accurate information is available.

By observations of temperatures at inlet, between stages, and at discharge, together with accurate measurement of quantity and discharge pressure, and by determination (also quite accurate) of leakage and radiation losses, the efficiency of the compressor may be closely determined. This has been done, and curves showing the performance have been published.²

For the purposes of this discussion curves have been prepared (Fig. 9) showing the gas consumption of a turbo blower when blowing at various rates and against various blast pressures, from which the amount of gas needed under various conditions may be quickly determined. The gas consumption is of course that necessary to operate the plant.

Comparison of Steam-Engine, Steam-Turbine, and Gas-Engine Installations

In comparing the relative merits of these three types of blowing apparatus, the only proper method is to base the computation upon the best present practice for each.

The steam practice in blast-furnace installations is hopelessly behind the times, and in no way comparable with that of central power stations supplying electric light and street-railway service. In order to place the turbo blower on a fair basis for comparison with other types, it must be surrounded with the most efficient auxiliaries for steam generation, production of vacuum, etc., and blast-furnace engineers must make up their minds to abandon existing practice and follow in the footsteps of designers of electric power stations. To this end boilers must be installed capable of generating steam at 200 lb. pressure and with 200° superheat.

Where ordinary forms of gas burners are used under boilers it is not advisable to clean the gas, since the sensible heat, amounting to about 8 per cent. of the total heat in the gas, is available and utilizable. It will, however, be necessary to clean the boilers frequently. It is the practice abroad, and should be the practice here where efficiency is an object, to use pre-mixing burners, by the use of which the air can be accurately regulated and the best efficiency obtained. In this case it is advisable to clean the gas, otherwise the dust may slag on the tubes owing to the high furnace temperature.

² Richard H. Rice: Commercial Application of the Turbine Turbo-Compressor, *Transactions of the American Society of Mechanical Engineers*, vol. xxxiii, p. 381 (1911).

Furthermore, the practice of using a single jet condenser connected with a number of units, and receiving steam from as many as may be in operation, must be abandoned in favor of a condenser for each unit, preferably of the surface type, in order that the best possible vacuum may be realized for each unit unaffected by any misadjustment or improper operation of its neighbor. The use of a single turbo blowing engine per furnace has become standard practice on account of the small dimensions, lower cost, and higher efficiency per unit of capacity of large turbo blowers and the short time required to start a spare unit, to say nothing of the reduced cost of installation and operation of single units as contrasted with multiples. It is, therefore, desirable to lay out the turbo-blower house on the unit system, each blower having its own boilers, condenser, pumps, and other auxiliaries, with suitable spares. These various units will of course be cross connected by suitable pipes so that they may be used in all desirable combinations. In this way maximum flexibility is combined with the best efficiency.

Taking this method of installation as a basis for the turbo blower, we shall naturally use for the gas engine an installation of two units per furnace, since this plan appears to find favor with progressive gas-engine builders; and we shall also use similar arrangements of horizontal compound steam engines as the basis for the reciprocating steam-driven type.

For the boilers, we shall use the largest units possible to obtain, preferably two boilers per blowing unit, and of such type that the radiation losses from the brick work are a minimum, while the possibilities of decrease of efficiency due to lodgment of dust from the gas are as small as possible. The superheaters should be built in the same setting as the boilers for simplicity of manipulation.

Performance of Reciprocating Steam Engines

For plants as now being installed it is customary to use not over 150 lb. gauge pressure and not over 27 in. vacuum, and it must be admitted that this type of blowing unit is not well adapted for higher pressures, temperatures, or vacua in blast-furnace work. Therefore, we must assume a steam consumption for such engines of not less than 13 lb. per indicated horse power under blast-furnace conditions, corresponding to 14.15 lb. per shaft horse power.

As we shall see later, this assumption does not need to be exact, as the steam consumption would have to be reduced to a figure not attained in the best engines ever built for any purpose, with the highest degree of superheat ever used and with all conditions favorable for the steam engine (as they are not in blast-furnace work), to put this type on equal terms with the other as regards fuel consumption.

The data necessary for determining the gas consumption of this type of blowing engine are as follows:

Production of pig per day of 24 hr.....	450 tons.
Coke per ton.....	2,240 lb.
Displacement per pound of coke (avg. value)	65 cu. ft. per min.
Theoretical horse power required to compress	
100 cu. ft. of air to 15 lb.....	5 h.p.
Diagram factor for air tub ³	0.94
Mechanical efficiency of air tub ⁴	0.92
Displacement air required to produce 450 tons per day:	

$$\frac{450 \times 2,240 \times 65}{24 \times 60} = 45,500 \text{ cu. ft. per minute.}$$

Shaft horse power required to blow this air:

$$\frac{5 \times 455}{0.94 \times 0.92} = 2,630.$$

Boilers (coal fired) of large size have recently given 80 per cent. efficiency in electric power stations,⁵ and even a small boiler without special attention has given recently 82 per cent. at the plant of the General Electric Co. at Lynn. For blast-furnace use, however, we must expect lower values, but 70 per cent. efficiency is readily attainable in practice with proper attention, or 75.5 per cent. with pre-mixing burners and clean gas. I shall assume the same efficiency, due to the same care, in case of turbo blowers and of reciprocating steam engines.

In the case of gas burned under boilers we have additional available heat due to the fact that we can utilize the sensible heat contained in the gas at high temperature. The amount of heat per cubic foot of gas per degree Fahrenheit is 0.0182 B.t.u., and allowing gas for boilers at 500° F. and for engines at 60° F., gives 8 B.t.u. sensible heat per cubic foot difference.

For this reason I take

	B.t.u.
Amount of heat in gas to gas engine.....	95
Amount of heat in gas to boilers.....	103

The heat per pound of steam is determined as follows:

	B.t.u.
Initial total heat	1,194.6
Final heat of liquid.....	83.1

1,111.5

³ Freyn and Bacon: Discussion, *Transactions of the American Society of Mechanical Engineers*, vol. xxxiii, p. 407 (1911).

⁴ *Idem*, p. 408.

⁵ J. W. Parker: Operation of Large Boilers, *Journal of the American Society of Mechanical Engineers*, p. 33 (February, 1914).

Hence we have the gas for boilers for reciprocating blowing engines:

$$\frac{2,630 \times 14.15 \times 1,111.5}{103 \times 0.70} = 574,000 \text{ cu. ft. per hour.}$$

Performance of Gas Blowing Engines

In the case of the gas-engine driven blowing engine we use the same data as above, to find the shaft horse power required to blow the same quantity of displacement air. We also use the value of a horse power hour, which is

$$\frac{33,000 \times 60}{778} = 2,545 \text{ B.t.u.}$$

The thermal efficiency of the gas engine has been assumed in recent discussions⁶ at 21 per cent., based on tests which were made with great care. Using this efficiency we shall consume

$$\frac{2,545}{0.21} = 12,120 \text{ B.t.u. per shaft horse power in the gas engine.}$$

As above, we assume 95 B.t.u. per cubic foot of gas as the available heat for gas-engine purposes; hence we have for the gas required by an engine of this type to produce pig at the rate of 450 tons per day

$$\frac{2,630 \times 12,120}{95} = 335,000 \text{ cu. ft. of gas per hour.}$$

Performance of Turbo Blower

For the turbo blower the use of displacement air measurement no longer holds; and I have shown that the ratio between displacement and actual air is a variable, not a constant. The experience gained from the large number of units installed where air-governor measurements are available shows a variation in the air consumption per pound of coke from a minimum figure of 45 cu. ft. to a maximum of 55, depending upon local conditions, such as quality of coke, tightness of air valves and blast mains, etc. This experience teaches that 50 cu. ft. per pound is a safe and conservative figure for plants where all sources of loss are properly looked after and removed. Furthermore, I have shown the steadiness of the air current and consequent decrease of coke ratio due to the increased iron production resulting from this; however, I shall not in this discussion take any account of the effect of this, leaving it to future determination when more data are in my possession. We now have, for the turbo blower, the following data:

⁶ H. J. Freyn: *Bulletin of the American Iron and Steel Institute*, p. 55 (1913).

Production of pig per day.....	450 tons
Coke per ton, as before	2,240 lb.
Actual air per pound of coke.....	50 cu. ft.

$$\text{Total air per minute } \frac{450 \times 2,240 \times 50}{24 \times 60} = 35,000 \text{ cu. ft.}$$

As is well known, the friction loss in blast mains, stoves, tuyères, and through valves is less for steady than for fluctuating flow. In the case of steam pipes it is common practice to make the pipes to steam turbines much smaller than in the case of similar pipes to steam engines. Also, the steady flow of air from a turbo blower maintains better conditions in the furnace and results in a freer flow through the charge.

For this reason, a lower blast pressure is needed with this type of blowing apparatus. This lower blast pressure of course decreases the power and gas consumption of the turbo blowers, but no account is taken of this item in the following computations:

We then have

Theoretical horse power required to compress 100 cu. ft. of air to 15 lb.....	5
Making use of 200 lb. steam pressure, 200° super-heat, and 28.5 in. vacuum (conditions actually obtaining in a number of electric power stations) we shall have steam consumption of turbine used as driver, pounds steam per shaft horse power....	9.35
The shaft efficiency of the compressor has been assumed as 70 per cent., a result derived from many actual tests of such apparatus as safe and conservative.	

We then have

$$\frac{5 \times 350}{0.70} = 2,500 \text{ shaft horse power.}$$

The heat supplied to the steam by the boilers will be

	B.t.u.
Initial total heat.....	1,307.5
Final heat of liquid.....	60.1
Heat supplied by boilers per pound of steam....	1,247.4

We can now determine the gas required for the boilers, which is

$$\frac{2,500 \times 9.35 \times 1,247.4}{103 \times 0.70} = 404,000 \text{ cu. ft. of gas per hour.}$$

We now have the following comparative results:

Gas Consumption in Cubic Feet per Hour

Reciprocating Steam Engine	Reciprocating Gas Engine	Turbo Blower
574,000	335,000	404,000

The above table shows that in order that the reciprocating steam blowing engine should equal the turbo blower in gas consumed it would be necessary to reduce its steam consumption to slightly over 8 lb. per indicated horse power per hour, a result manifestly impossible to accomplish with any known practical apparatus.

I shall now discuss these figures in reference to the operation of a steel plant containing eight blast furnaces and ample use for electric power.

It is usually assumed that a 450-ton blast furnace produces 150,000 cu. ft. gas per ton of iron, or

$$\frac{150,000 \times 450 \times 8}{24} = 22,500,000 \text{ cu. ft. gas per hour}$$

in a plant of eight blast furnaces.

Of this 40 per cent. is consumed in stoves and leakage, leaving 60 per cent., or

13,500,000 cu. ft. gas per hour

available for blowing and the production of electric power.

I shall discuss two types of installations:

- A—Blowing station with gas-engine blowing engines.
Electric station with gas-engine driven electric generators.
- B—Blowing station with steam-driven turbo blowers.
Electric station with steam-driven turbo generators.

In the gas blowing station will be installed 20 gas-blowing engines. In the turbo-blower station there will be 10 turbo blowers.

In the gas-electric station, where the units will be of 10,000-kw. capacity, there will be six units.

In the turbo-electric station eight 7,500-kw. units will suffice.

The comparative results on this basis are given in the following table:

*Comparison of Turbine and Gas-Engine Driven Blowing and Electric Plants
for Eight 450-ton Blast Furnaces*

	A	B
1. Cubic feet of gas per hour required for blowing eight furnaces as per previous work for one furnace.....	2,680,000	3,232,000
2. Subtract (1) from 13,500,000 (the total gas available) to give cubic feet per hour of surplus gas.....	10,820,000	10,268,000
3. B.t.u. per cubic foot of gas (including sensible heat to boilers).....	95	103
4. B.t.u. per hour in gas for electric station (surplus gas).....	1,028,000,000	1,056,000,000
5. B.t.u. per kilowatt-hour (see next table).....	18,820	25,200
6. Kilowatts producible in electric station from surplus gas.....	54,600	41,900
7. Total kilowatt capacity installed in electric station.....	64,000	60,000
8. Cost of electric station per kilowatt installed (using lowest figure ⁷ for gas electric station quoted in any publication but believed to be far too low for the ordinary purchaser).....	\$68.06	\$50.00
9. Total cost of electric station.....	\$4,355,840.00	\$3,000,000.00
10. Shaft horse power of blower per furnace (see above).....	2,630	2,500
11. Shaft horse power of blowing house as installed..	26,300	25,000
12. Cost of blowing station (lowest published ⁸ for gas engine) per horse-power (rated) installed—actual figures for steam station.....	\$74.50	\$50.00
13. Total cost of blowing station.....	\$1,959,400	\$1,250,000
14. Total cost of plant, (9) plus (13).....	\$6,315,240	\$4,250,000

In order that the turbo-electric station shall produce the same total kilowatt output as the gas-electric station it will be necessary to make, by coal burning

$$54,600 - 41,900 = 12,700 \text{ kw.}$$

To generate this will require

$$12,700 \times 25,200 = 320,000,000 \text{ B.t.u. per hour from coal.}$$

Tons of coal per hour at 24,000,000 B.t.u. per ton

$$\frac{320,000,000}{24,000,000} = 13.33$$

At \$1 per ton, as in the Pittsburg district, this means a cost for coal of \$13.33 per hour.

Cost of this make-up coal per year at \$1 per ton:

$$365 \times 24 \times 13.33 = \$116,770$$

⁷ Arthur West: Discussion, *Bulletin of the American Iron and Steel Institute*, p. 65 (1913).

⁸ *Idem*, p. 66.

I shall assume, as is done⁹ at the Gary works of the U. S. Steel Corporation, 15 per cent. fixed charges for the gas-engine plants, and, as is the common custom¹⁰ in large turbo-electric plants, 11.5 per cent. fixed charges for the turbo blower and electric plants.

Then we have

	A	B
Interest, depreciation, and taxes, per cent. . . .	15	11.5
Total fixed charges.	\$947,286	\$488,750

Operating Costs of Blower House.

Gas cleaning.	35,200	
Plant operating and maintenance, labor and supplies.	100,100	83,000
Water ¹¹	8,800	12,000
Total blower operating costs.	\$144,100	\$95,000

Operating Costs of Electric Station.

Gas cleaning.	\$105,600	
Plant operating and maintenance, labor and supplies.	298,600	227,500
Water ¹¹	26,400	36,000
	\$430,600	\$263,500

Total operating costs and fixed charges per year.	\$1,521,986	\$847,250
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The difference between these operating costs

$$\$1,521,986 - \$847,250 = \$674,736.$$

The cost of make-up coal to bring up the output of the turbo-electric station to that of the gas-electric station at \$1 per ton was found above to be \$116,770.

⁹ H. J. Freyn, Gas Engine in Modern Blast Furnace and Steel Plants, *Bulletin of the American Iron and Steel Institute*, 1913, p. 47 (1913).

¹⁰ This figure is customarily allowed by large public service corporations and is made up as follows:

	Per cent.
Depreciation of machines, 5 per cent., and buildings, 3 per cent., averaging.	4½
Interest.	5
Taxes and insurance.	2
	11½

¹¹ The costs of water for the steam blower house and electric station "B" are taken high for extreme cases. Usually they would be greatly lower than the figures given.

It will be seen that the above difference in operating costs and fixed charges will not be wiped out until the cost of coal has risen to

$$\frac{\$674,736}{116,770} = \$5.78 \text{ per ton.}$$

The gas-plant costs have been based on an average kilowatt load substantially equal to that produced by the average gas. Actually this could not be obtained in the gas plant without some producers banked as reserve for periods of poor gas and peak loads. The cost, fixed charges, fuel costs, maintenance and operating charges of such a bank of producers have been wholly omitted. Such an arrangement would not be necessary in the case of a steam plant, as the boilers furnish sufficient heat-storage capacity and can be forced to any desired amount without additional plant. The cost of this reserve capacity must be added to the cost of plant, heretofore estimated, to put the two plants on an equality and will operate to increase the advantage of the turbine stations over the gas engine and increase the limiting cost of coal at which the two types become equal in operating costs.

The above figures take no account whatsoever of the improvement in quality and quantity of output from the furnace due to the use of the turbo blower, which will be all gain over the above figures.

The figures for the gas engine are based on the lowest obtainable. I believe them much too low for the average plant.

The figures for the turbo-blower and turbo-generator plants are taken on a very conservative basis.

Conclusions

From the foregoing the following conclusions are justified:

Turbo blowing engines give a steadier blast and a better measurement of blast than any other form of blowing apparatus for blast furnaces.

Steadier and better measured blast gives better conditions in the furnace, which leads to increased output and improved quality.

Turbo blowers operate with less attention and are more flexible in service than any other form of blowing apparatus for blast furnaces.

The gas consumption of the reciprocating steam blowing engine is so high as compared with that of a turbo blower, that it is not possible with any form of blowing engine in sight to come within reasonable range of the turbo blower in this respect.

Taking the best figures published by gas-engine manufacturers as representing the efficiency of the gas engine in practice and comparing them with the turbo-blower station in which modern steam practice is availed of to secure the efficiencies which are being obtained in turbo-electric stations, with reasonable allowances for the difference in fuels,

the gas consumption of the turbo blower is only 20 per cent. in excess of the gas blower.

Comparing blast-furnace and steel-works installations involving a plant containing eight blast furnaces with ample use for surplus gas and contrasting gas blowing and gas-electric stations with turbo blowing and turbo-electric stations (using the best figures given by any gas-engine advocates for the gas engine, and usual and conservative figures for the steam stations), the saving in operating costs and fixed charges by the use of the steam plant is so great as to make it necessary that coal should rise to \$5.78 per ton in order that the cost of make-up coal to give the same outputs in the steam-electric and gas-electric stations could wipe out this saving; while with coal at present and usual prices, the saving by use of the steam turbine in such stations is very large.

The steam practice of our blast furnaces and steel works is hopelessly behind the times and needs entire revision, but with modern economical steam practice applied to such plants, the turbine-driven plant can be operated at lower cost with less stoppage and loss of product, and with a larger product for the same furnace of better quality, than by any other prime mover now in use for the purpose.

In practice, with a load factor of 75 per cent. in the electric station and 80 to 85 per cent. in the blowing house, the gas units will be operating below their maximum efficiency; and owing to their low mechanical efficiency and lessened thermal efficiency at light loads, will suffer considerably more under these conditions than the turbo blowers, which are designed for best efficiency at average operating conditions, or than the turbo-electric units, which have a remarkably good efficiency at light loads. Hence the above comparisons are unduly favorable to the gas-engine stations. Furthermore, neither gas nor turbo-electric stations will generate sufficient power for the requirements of the steel mills to which they will be an adjunct; and the deficiency must be made up by additional units, the fuel for which must be coal. With gas-electric power units, gas producers must be installed; and since the requirements of a steel plant for power are very variable and the power station must be always ready to supply the demand these producers will be banked for considerable periods; while the turbo-electric plant, by reason of the heat-storage capacity of boilers, will be always ready to meet the demand with little or no banking. Therefore, this part of the plant must inevitably be steam driven.

For smaller plants with fewer furnaces the advantage of the turbine-driven units becomes more and more considerable; and for plants with less than four furnaces, or for plants where use for the surplus gas does not exist, the turbo blower is without competition in adaptability to the blowing of blast furnaces by reason of its flexibility, quick adjustability to different furnace conditions, automatic control of pressure to maintain

a constant rate of flow into the furnace and steadiness of blast, resulting in improved furnace conditions.

APPENDIX

Computation of British Thermal Units per Kilowatt-Hour for Gas-Electric and Turbo-Electric Generating Stations

1. Gas-Electric Station

Per Cent.

Maximum shaft efficiency.....	25
Average shaft efficiency during operation at good unit load factor.....	21
Average generator efficiency.....	93
Average over-all efficiency, $0.21 \times 0.93 =$	19.5
Allowing 8 per cent. of average load for auxiliaries, net average over-all efficiency = $19.5 \div 1.08 \dots$	18.1

II. Turbo-Electric Station

	Maximum Efficiency	Average Efficiency at Good Unit Load Factor
Boiler.....	0.80	0.70
Generator.....	0.95	0.93
Turbine.....	0.75	0.70
Available energy (200 lb. per sq. in., 200° F., 28.5 in. vacuum.....)		398.3 B.t.u.
Heat added to steam in boiler.....		1,247.4 B.t.u.

$$\text{Over-all efficiency} = \frac{398.3 \times 0.70 \times 0.70 \times 0.93}{1,247.4} = 1.46 \text{ per cent.}$$

Allowing 8 per cent. of average load for auxiliaries,

net average over-all efficiency = $1.46 \div 1.08 = 13.5$ per cent.

Hence the following figures, for electric stations with good load factors such as would occur in such plants, allowing for all auxiliaries, are per kilowatt-hour:

Gas-engine station.....	$3,412 \div 0.181 = 18,820$ B.t.u.
Steam-turbine station.....	$3,412 \div 0.135 = 25,000$ B.t.u.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor¹ American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

The Apex Law in the Drumlummon Controversy

BY CHARLES W. GOODALE, BUTTE, MONT.

(New York Meeting, February, 1914)

THE principles and theory on which the U. S. mining law of 1872 was based are well understood, and have been discussed at great length by many writers. The papers by Dr. R. W. Raymond¹ in the *Transactions* of the American Institute of Mining Engineers have covered the subject very fully, with explanations of practical results in many adjudicated cases. It is believed that the following instance of the application of the law in recent years will have some weight in a discussion of proposed changes in our mining laws.

The litigation between the St. Louis Mining & Milling Co., and the Montana Mining Co., Ltd., a British corporation, over portions of the Drumlummon lode covered more than 20 years, and resulted in a judgment for damages in favor of the St. Louis Co. of such an amount that the Montana Co. declined to pay the judgment, and therefore lost the entire property.

The Drumlummon lode claim in the Marysville district, Mont., was located in 1876 by Thomas Cruse. When he staked out his claim, he assumed that the discovery on the Drumlummon vein, and another showing of mineral about 200 ft. distant, were on the same vein, and the center line of the claim was given a course of N. $57\frac{1}{2}^{\circ}$ E. Development of the Drumlummon vein soon showed that its strike was about N. 15° E., and that the other outcrop was on the North Star lode; so it was inevitable that the Drumlummon vein would cross one or both side lines of the location, and the Montana Co. found it advisable to acquire adjoining claims on both sides of the Drumlummon, in order to avoid possible litigation over apex rights. They bought the Marble Heart, but would not purchase the Hopeful (see Figs. 1 and 2), at the terms proposed for the reason that exploration in the Cruse level had shown the vein to be barren in that region. The owners of the Hopeful were sinking an incline from that claim in 1889, and the Montana Co. enjoined them, contending

¹ The Law of the Apex, *Trans.*, xii, 387 (1883-84).

The Law of the Apex, Appendix, *Trans.*, xii, 677 (1883-84).

Lode-locations, *Trans.*, xv, 272 (1886-87).

End-lines and Side-lines in the U. S. Mining Law, *Trans.*, xvii, 787 (1888-89).

Imaginary Boundaries, *Trans.*, xviii, 182 (1889-90).

that they were so near to the Cruse level that the holes when blasted would make an opening into that level. Before firing, they ran up the American flag as a signal to all the world that they had the English company "dead to rights," and would soon prove the extent of its trespass. They heard the shots go off, but as no smoke came up, one of them went down to investigate. Hearing nothing from him, another went down, found the first man overcome by powder fumes, got him into the bucket, and after getting in himself, signaled to hoist. Before the bucket reached the surface, he also became unconscious, and what was expected to be a triumph, came near being a tragedy. It was fortunate that the flag did not have to be lowered to half-mast.

The St. Louis claim was located Sept. 28, 1878, as an extension of the Drumlummon. There is abundant evidence from the location notice, and from other facts, that its side lines were straight, as shown by dotted lines on Fig. 2; but the discovery vein of the St. Louis proved disappointing when opened, and, the 9-Hour location having been made in the meantime by William Robinson, July 26, 1880, on a promising discovery of ore, the St. Louis owners, in surveying their claim for patent in July, 1881, ran their lines so as to take in the 9-Hour discovery. When Robinson objected, they made angles in their side lines, leaving out the shaft, but by such a narrow margin (about 10 ft.), that when they applied for patent, he put in an adverse claim. On the 7th of March, 1884, he was induced to withdraw his suit, the St. Louis owners agreeing to convey to him, on the issuance of their patent, a portion of the ground described as follows:

"Commencing at a point from which the center of the discovery shaft of the Nine-Hour lode bears south 39 degrees 32 minutes east, said course being at right angles to the boundary line of the St. Louis lode, between corners 2 and 3, fifty feet distant, thence north 50 degrees 28 minutes east on a line parallel to the aforesaid boundary line of the St. Louis lode claim, between corners 2 and 3, 226 feet to a point on the boundary line of the St. Louis lode between corners 1 and 2; thence south 20 degrees 28 minutes west along said boundary between corners 1 and 2, 60.5 feet to corner No. 2 of St. Louis lode; thence 400.31 feet to corner No. 3 of St. Louis lode; thence north 46 degrees 10 minutes west along the line of boundary of St. Louis lode, between corners 3 and 4, 30 feet to a point; thence north 50 degrees 28 minutes east along a line parallel to the boundary line of St. Louis lode, between corners 2 and 3, 230 feet to the point of beginning, including an area of about 12,844.50 square feet *together with all mineral therein contained.*"

As this would have the effect of moving the St. Louis line 40 ft. farther away from his discovery, he felt secure in his mineral rights to the 9-Hour vein. The territory covered by this contract was known through all the years of subsequent litigation as the "Compromise Strip." (See Fig. 2.) The locator had only developed the property to a limited extent, when he sold it to the Montana Co.

Going back now to the beginning of the litigation with the St. Louis M. & M. Co.—a suit was started on Oct. 14, 1890, by that company

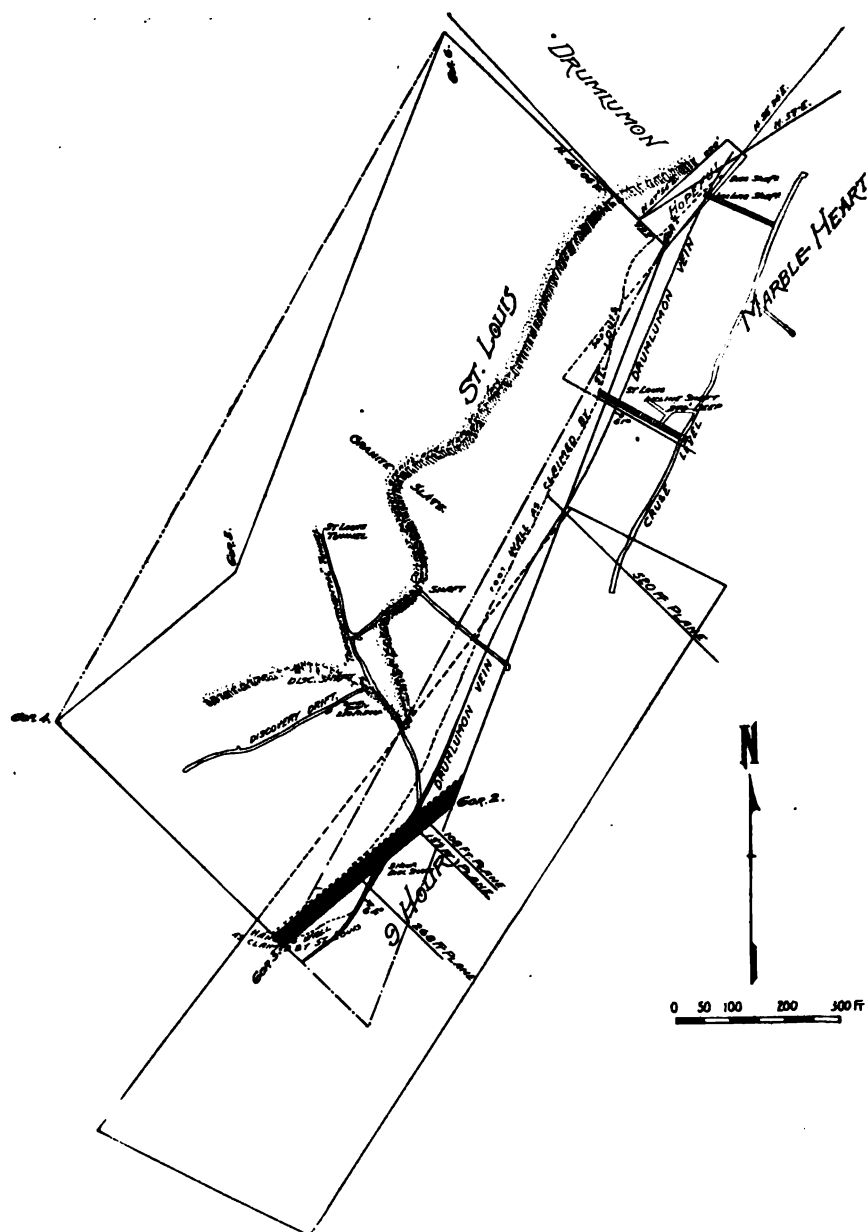


FIG. 2.—DIAGRAM, SHOWING ST. LOUIS AND 9-HOUR, WITH PARTS OF DRUMLUMMON AND MARBLE HEART LOCATIONS, OUTCROP OF DRUMLUMMON VEIN, ETC.

against the Montana Co. for trespass and \$2,000,000 damages. An incline shaft had been sunk on the St. Louis claim to a depth of about 370 ft. (Fig. 2), in an effort to find the Montana Co.'s workings, as it was known that the Cruse level had been driven several hundred feet south of the Drumlummon end line, and into the Marble Heart claim. This shaft was started as close as possible to the side line of the St. Louis, and made parallel to the known dip of the Drumlummon vein. The geologists of the plaintiff asserted that it was sunk on the foot-wall of the Drumlummon vein, which, they said, entered the St. Louis claim at its north end line with a width of about 70 ft., varying from that to about 30 ft. on its course southerly; and they claimed the Jubilee and Jubilee No. 2 ore shoots in virtue of apex rights to which they were entitled by the alleged fact that they had the foot-wall in the St. Louis claim, the hanging-wall being in a junior location, the Marble Heart. The case came on for trial April 17, 1893, in the U. S. District Court in Helena, and five weeks later, the jury gave a verdict in favor of the defendant, thus supporting its geologists in their statement that the Drumlummon in its apex through the region in question was a narrow vein, not exceeding 3 ft. in width, existing between two well-defined walls and with a distinctive barren filling. The nature of this vein matter is shown below, where also will be found a partial analysis of the country slate or hornstone, both *within* the wide vein as claimed by the plaintiff's geologists (A), and *outside* of it (B).

	Barren Vein Matter 1 ft. to 3 ft. wide Per cent.	(A) Foot-wall Hornstone Per cent.	(B) Hanging-wall Hornstone Per cent.
SiO ₂	23.	58.2	62.5
FeO	2.7	3.4	3.4
CaO	33.7	7.9	5.2
MgO	5.2	1.8	2.3
Al ₂ O ₃	4.6	18.0	18.5

It will be seen there is very little difference in the composition of the two samples of hornstone; but the plaintiff insisted that what the defendant called the whole vein, was only the hanging-wall gouge, and that the rock between this and the plaintiff's alleged foot-wall, from 30 to 70 ft. distant, was "broken, brecciated, recemented vein matter." The defendant pointed to many places where the stratification planes of the slate were strongly in evidence, and asserted that wherever they were indistinct this condition could be explained by their proximity to the contact with diorite.

The plaintiff was undoubtedly influenced in his exaggerated claim for \$2,000,000 by the name, "The Jubilee," given to the ore shoot in

controversy by the defendant. The name might have implied that the ore was of extraordinary richness, but it was in fact based upon the incident that it was discovered June 21, 1887, the day when all Englishmen were celebrating the Golden Jubilee of Queen Victoria, the fiftieth anniversary of her accession to the throne.

A second suit, based upon the same facts as in the case just mentioned, but covering a later period, was dropped.

And now we come to another chapter of the war between the two companies, the battle ground being shifted to the extreme south end of the mine. (See Fig. 3.)

After the purchase of the 9-Hour claim, the Montana Co. began active exploration of the ground both in the discovery shaft and in deep levels. By the summer of 1893 developments had shown that the vein would cross the east side line of the St. Louis claim as patented; and on June 19, 1893, the St. Louis Co. commenced an action to restrain the Montana Co. from working any part of the vein, the apex of which was in the St. Louis claim as patented, and for \$10,000 damages, for ore extracted within the boundaries of the compromise ground. A temporary order was issued, restraining the Montana Co. from sinking its apex shaft upon the compromise ground near the boundary between that ground and the 9-Hour claim. The case was removed to the Federal Court, where a new complaint was filed Sept. 16, 1893, claiming \$200,000 damages for the ore then extracted. For 100 ft. or more along the surface, and to a depth of about 50 ft., the ore was richer than had been found in any other workings of the Drumlummon vein; and the fight for its possession became very bitter. The Montana Co. claimed the ground in controversy by virtue of the agreement to convey the compromise ground. The action being in the Federal Court, it could not maintain its right without the full legal title to the ground, and thereupon it commenced an action to compel the St. Louis Co. to deed to it the compromise ground, in accordance with the agreement which had been entered into with Robinson and his associates. Pending the proceedings in this "specific-performance suit," the proceedings in the Federal Court were suspended. The specific-performance suit dragged its length along until the year 1898, when the Supreme Court of the United States affirmed the decision of the State District and Supreme Courts, requiring the St. Louis Co. to convey the compromise ground in accordance with its agreement, which was done, and a deed was executed July 1, 1898, embodying the description as hereinbefore set forth.

The patent of the St. Louis claim was issued in July, 1887, and William Robinson, or his successor in interest, the Montana Co., should have received a deed promptly, but it was only given after eleven years of annoying and expensive litigation.

In November, 1898, the St. Louis Co. filed its first amended complaint

in the United States Circuit Court, claiming the right to the ores *beneath the compromise ground* by virtue of its ownership of the apex of the vein within the limits of the St. Louis claim, wholly outside of the compromise ground, from a point on the east side line of the St. Louis claim between corners Nos. 1 and 2, 520 ft. distant from corner No. 1 of the St. Louis claim to a point on the west side line of the compromise strip distant 108 ft. from the intersection of the west side line of the compromise ground with

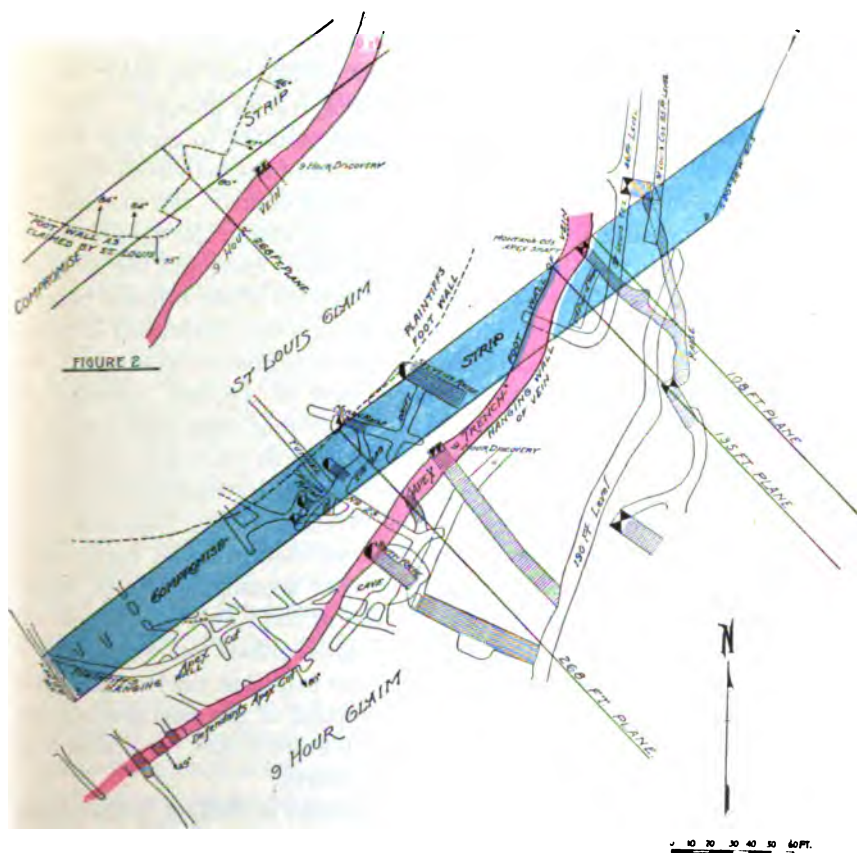


FIG. 3.—WORKINGS IN ST. LOUIS AND 9-HOUR LOCATIONS.

the east side line of the St. Louis claim, running from corners Nos. 1 and 2, where the hanging-wall of the vein began to cross from the St. Louis ground into the compromise ground, and for an additional portion of the said vein for a distance of 25 feet to the point where the foot-wall of the vein passed out of the east side line of the St. Louis lode claim into the compromise ground. These points of departure of the hanging-wall and

foot-wall from the St. Louis ground into the compromise ground became known as the 108-ft. and the 133-ft. planes.

The case was not reached for trial until 1899. On June 24, 1899, shortly before the commencement of the trial, the plaintiff amended its complaint, alleging that since the filing of the original complaint and up to and including the time of the filing of the second amended complaint, the defendant had extracted an additional amount of ore of the value of \$400,000, making the total damages claimed \$600,000. The trial was had before Judge Knowles in the U. S. Circuit Court in August, 1899, and resulted in a verdict for the plaintiff in the sum of \$23,209 for ore extracted north of the 108-ft. plane, or from a vein which had its apex entirely within the St. Louis claim outside of the compromise ground. The District Judge held that in order to entitle the plaintiff to recover it must have within its own surface lines the entire apex of the vein from which the ore was extracted. Both parties sued out writs of error, the St. Louis Co. upon the ground that it was entitled to the ore within the vein to the extent that it had any of the apex within its surface lines, and the Montana Co. on the ground that the deed conveyed to it all of the mineral beneath the compromise ground, and that as the verdict embraced damages only for ores extracted from beneath the compromise ground, it was entitled to a judgment. The Circuit Court of Appeals reversed the case on the St. Louis Co.'s writ of error, holding that that company, as the owner of the senior location, was entitled to recover to the extent that it had any of the apex within the surface of its ground, and affirmed the judgment to the extent of \$23,209, overruling the Montana Co.'s contention that the deed conveyed the ore beneath the compromise ground, where such ore was found in a vein apexing partly or wholly outside of the deeded ground. This action of the Court of Appeals was reversed by the Supreme Court of the United States on writs of error sued out by the Montana Co.; and that court held that the reversal of the judgment of the Circuit Court as to one portion of the case reversed the entire case, and remanded the case to the Circuit Court for a new trial, without deciding any of the points in controversy.

The case was again tried in the Circuit Court in Helena, before Judge Hunt, District Judge, sitting as a Circuit Judge, in May and June, 1905. Judge Hunt, following the ruling of the Circuit Court of Appeals, held that the plaintiff was entitled to recover for the ores extracted between the 520-ft. and the 133-ft. plane, and a verdict was rendered in the sum of \$195,000. Writs of error were again sued out, and the case was reviewed by the Circuit Court of Appeals. It then went to the Supreme Court of the United States, which, on Jan. 14, 1907 (204 U. S. 204), rendered its opinion that the deed to the compromise ground conveyed all of the mineral beneath the compromise ground. As the decision of that point might end the litigation, none of the other questions were directly passed upon.

But in his opinion, Mr. Justice Brewer raised, for the first time, the suggestion that the deed to the compromise ground carved out a section from the vein.

When the mandate was filed in the Circuit Court, the Montana Co. moved to dissolve the injunction, which had been granted at the time of filing the original complaint, restraining it from extracting ores beneath the compromise ground, basing its motion upon the decision of the Supreme Court of the United States that the deed to the compromise ground conveyed to the Montana Co. all of the ores beneath its surface. On March 30, 1907, Judge Hunt, District Judge, sitting as a Circuit Judge, granted the defendant's motion and dissolved the injunction. The St. Louis Co. then applied to the Supreme Court of the United States for a modification or an explanation of its opinion relative to the construction of the deed, but the Supreme Court denied the application, without any further opinion. The St. Louis Co. then, on Aug. 28, 1907, made an application for leave to amend its complaint, so as to claim damages for ores extracted from the vein on its dip after it had passed through the compromise ground, placing its damages at \$500,000 for ores extracted from that portion of the vein on or about the 30th day of June, 1893, and for \$500,000 for ores extracted from that portion of the vein between June 30, 1893, and the date of tendering this amended complaint. It also alleged that the foot-wall of the vein was at all points either in the compromise strip or the St. Louis claim, and passed out at some indefinite point across the south end line of the St. Louis claim as patented; the contention of counsel for the St. Louis Co. being that under the Supreme Court decision, the St. Louis Co. owned all ore in the vein after it had departed from the compromise ground on its dip, so long as any part of the apex of the vein lay within either the compromise strip or the St. Louis claim.

This proposed amended complaint not only extended the surface area in which was embraced the apex of the vein from which the ore was alleged to have been extracted, but carried the place of trespass into the depth of the mine and into stopes from which, many years before the filing of this amended complaint, ore had been taken, assumed by the complainant to be of the same high value as that taken from the compromise ground near the surface. The proposed amended complaint also set forth the fact that the Montana Co. had worked out the remaining veins belonging to it, had ceased its operations in Montana, and was in an insolvent condition, having no property within the jurisdiction of the Court, except the practically dismantled mining plant. The plaintiff asked for an injunction on the equity side of the court, restraining the Montana Co. from mining upon the compromise ground, notwithstanding the decision of the Supreme Court of the United States that the ore beneath that ground belonged to the Montana Co., basing its right to an injunction

upon the allegation of insolvency and inability of the Montana Co. to respond in damages in the event that the St. Louis Co. recovered judgment; and that the St. Louis Co. must necessarily recover judgment in some amount, because the portion of the vein from which the ores had been extracted was, by the decision of the Supreme Court, awarded to the St. Louis Co.

This construction of the decision of the Supreme Court was assailed by the attorneys for the Montana Co. upon the ground that the declaration of Mr. Justice Brewer in the opinion of the Supreme Court, that the effect of the compromise deed was to carve out a section of the vein, leaving the remaining portion unconveyed, referred only to the portion of the vein on its dip where all or a part of its apex lay within the St. Louis claim, and had no reference to the vein after its apex had wholly passed into the compromise strip, and that the Supreme Court by its refusal to modify or amend its mandate had practically so determined. Objection was also made to the filing of this amended complaint upon the ground that by removing the point of departure of the foot-wall from the 133-ft. plane to the south end line of the St. Louis claim, there was brought into the litigation a trespass upon an area of ground not embraced in the original ground, and that, as to that portion of the damages claimed, the statute of limitations had barred recovery. These contentions were finally overruled by the Circuit Court, and on Dec. 2, 1907, leave was given to file the third amended complaint. The defendant answered; and on Jan. 10, 1908, Judge Hunt issued an injunction restraining the Montana Co. from extracting ores from beneath the compromise strip pending the litigation. This injunction was unique in the history of mining litigation in Montana, in that it restrained the Montana Co. from extracting ores from the portion of the ground awarded to it, in order that these ores might be preserved to answer any judgment which might be recovered against the Montana Co. for trespass upon the portion of the vein which, it was claimed, the Supreme Court had awarded to the St. Louis Co. An appeal to review this action of Judge Hunt was taken to the Circuit Court of Appeals for the Ninth Circuit, and on March 2, 1909, the action of the Circuit Court was affirmed.

The case then came on for trial again before Judge Hunt in 1909, commencing June 14 and ending August 11. Before and during the trial, extensive explorations were made upon the surface ground, and the case was bitterly contested by the experts of the two parties as to the location of the foot-wall. At the close of the testimony on behalf of the Montana Co., the St. Louis Co. amended its third amended and supplemental complaint and replication, by withdrawing the allegation of the compromise ground as a part of the St. Louis claim, and substituting in lieu thereof the allegation, contained in the former complaints, that the compromise ground was and always had been a part of the 9-Hour claim. The com-

plaint was further amended by alleging that the apex of the vein in controversy passed entirely into the compromise ground at the point designated as the 268.6-ft. plane. This amendment, which was strenuously objected to by counsel for defendant, and was made after the granting of a sweeping injunction upon the construction of the Supreme Court decision, urged by the St. Louis Co.'s counsel, and after defendant had been compelled to prepare its defense of the cause to meet such legal theory, and after all of the evidence in chief of the St. Louis Co. had been introduced upon this theory—permitted the St. Louis Co. to make a complete somersault in its legal position, and seek a recovery upon new allegations of fact, upon which it confessedly could not have recovered under the complaint as it stood up to the time when the Montana Co. was required to present its evidence. Under the pleadings as they stood up to this time, a portion of the apex of all the vein in controversy lay within the compromise strip, and the compromise strip "was and always had been a part of the *St. Louis* claim," which part had been conveyed to the predecessors of the Montana Co. by a conveyance prior to that by which the remainder of the St. Louis claim had been conveyed by the same grantors to the St. Louis Co. Under the law the effect of the first deed—that conveying the compromise strip—was to convey all of the vein on its dip so long as any part of the apex lay within the compromise strip. By this last amendment, the St. Louis company was permitted to take the position that the compromise strip "was and always had been a part of the *9-Hour* claim," thus presenting a question of rights on the dip of a vein, the apex of which was divided between the junior *9-Hour* claim and the prior St. Louis location, thereby presenting entirely different questions both of law and fact. By this last amendment, the controversy was narrowed down to the question, whether the foot-wall passed into the compromise ground at the 133-ft. plane or at a point farther south, designated as the 268-ft. plane. The jury found that the foot-wall entered the compromise ground at the 133-ft. plane (as contended by the witnesses of the Montana Co., who maintained that the apex of the vein was only about 10 ft. wide, as against the assertions of the St. Louis witnesses that its width was 55 ft.), and awarded the St. Louis Co. damages for the amount of ore extracted from the vein after it had departed from the compromise ground, aggregating 1,912 tons, valued at \$237,470.40. The court had instructed the jury that if they found in favor of the plaintiff, they should include in their verdict interest at 8 per cent. per annum on the value of the ore extracted in 1893. As nearly 16 years had elapsed, the interest amounted to considerably more than the value of the ore. From the above amount was deducted the value of 218 tons of ore taken by the St. Louis Co. from beneath the compromise strip, amounting to \$34,341.38, including interest—making the verdict \$203,129.02. Upon a writ of error the judgment was reviewed by the Circuit Court of Appeals, where

it was affirmed (183 Fed. 51), and a petition for a writ of certiorari was denied by the Supreme Court of the United States on March 6, 1911. The Montana Co. decided to offer no further resistance, and in due course the property was sold by the sheriff to the plaintiff.

In the foregoing chronology of the litigation and analysis of court proceedings, Messrs. E. C. Day, C. F. Kelley, and L. O. Evans have kindly given the writer great assistance.

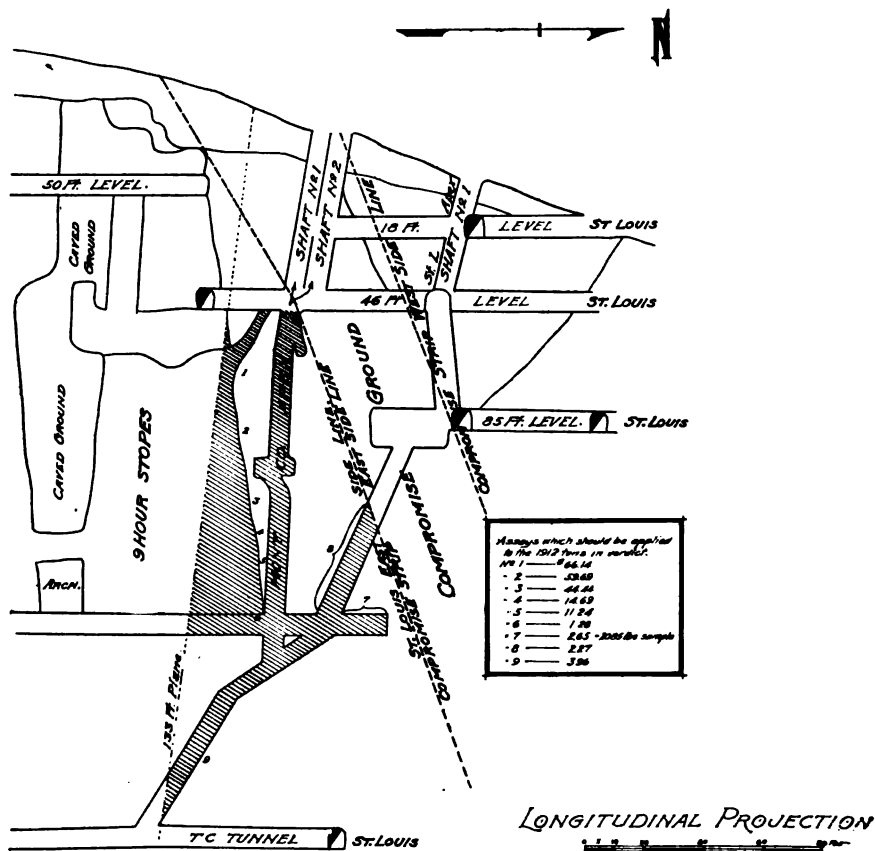


FIG. 4.—LONGITUDINAL PROJECTION OF 9-HOUR WORKINGS.

Reviewing this history, we find that the litigation in its various phases required in the aggregate about five months in the four trials on questions of fact, and that it reached the State Supreme Court once, the Circuit Court of Appeals five times, and the Supreme Court of the United States four times. Furthermore, much time was occupied by the courts in many preliminary hearings.

The jury by their verdict showed that they did not accept the wide-
vein theory of the plaintiff, for, if they had done so, they would have

awarded him damages for the proceeds of 31,592 tons of ore, instead of 1,912, which was about the amount admitted by the defendant as having been extracted by it under its own surface, but from that part of the vein having its apex partly within the St. Louis claim, west of the compromise strip. The award of \$237,470.40 must have placed an original value of about \$55 per ton on the ore. Fig. 4 shows the extent of the excavation made by the Montana Co. north of the 133-ft. plane and east of the compromise ground, and nearly all of this work was done in and beyond the extreme northern limits of the pay ore. Many samples were taken, before the trial, from the ore remaining in these workings, and the assays were submitted in court, giving what was thought to be ample proof that the material extracted did not return the company a profit of \$15 per ton. Samples had also been taken as the work progressed; but the samplers and assayers of that time were either dead or out of reach, and the company could not prove its records by personal testimony. Furthermore, believing fully in the validity of its title to the compromise ground and to all the mineral therein contained, and that no adverse rights could be successfully claimed beyond this strip and within the 9-Hour location, as patented, the defendant had not taken the precaution to run the ore in question through the mills by itself, thereby placing on record its bullion yield. Moreover, there was a metallurgical advantage in working it with ore from other parts of the mine.

The advocates and defenders of our mining law insist that in no way, except by giving the discoverer of a vein the right to follow it on its dip under adjoining ground, can he realize the full benefits to which he is entitled. But in this case, the plaintiff was not suing for any rights directly pertaining to his discovery vein. His demands were based entirely upon an incidental vein, the existence of which was unknown to him when he made the St. Louis location; and this vein only skimmed along his side line. Yet the grantees of the actual discoverer of the lode in question finally lost the entire property as a result of these demands. With no development along the vein, the strike in the 9-Hour discovery led the locator to believe that in compromising with the St. Louis, he would have the apex of the lode within his claim from end to end; and, according to all considerations of equity and justice, the worst that should have happened in consequence of his lack of knowledge of the course of the vein, should have been the loss of the ore included within the vertical boundaries of the St. Louis claim, excluding the compromise ground. But what happened? By the application of the apex law, he was deprived of all ore in the vein he had discovered up to the 133-ft. plane, and was adjudged to be liable in damages to the St. Louis Co. for more than \$200,000, covering 1,912 tons of ore which he had extracted from under his own surface. Furthermore, if the jury had accepted the wide-vein theory of the St. Louis witnesses, and had given it extralateral ownership to all

the ore up to the 268-ft. plane, the 9-Hour locator would have had *no rights in his discovery shaft*, and would have been a trespasser, and liable in damages, when he took a little sack of ore from there to the assayer, so that he could support the affidavit in his location notice, to the effect that he had made a valid discovery.

Can any one describe an instance where such injustice has resulted in the operation of the mining laws of other countries? With extralateral rights abolished, underground rights would be settled quickly on mathematical planes and years of litigation, with attending costs, would be avoided.

DISCUSSION

FREDERICK T. GREENE, Butte, Mont.—Mr. Goodale's very excellent history of the Drumlummon litigation educes certain reflections, which, while not obviously germane to his subject, are allied to the practical operation of mining lawsuits.

Most apex litigation necessitates many intricate problems involving the intersections of plane surfaces with one another. A very simple problem of this kind is indicated in Fig. 4. The vertical plane through the St. Louis side line cuts the vein and the line of intersection is represented by a bent line—bent, because there is a change in the dip of the vein producing a like change in the line of intersection. In consequence, a jury, probably composed of men who never even heard of descriptive geometry, is required to decide as to the cubical contents of this irregular polyhedron. In this particular instance the necessary knowledge is of the simplest, but there have been cases in which the question of damage involved some very intricate geometrical problems. In the solution of such problems it is essential that the ability to mentally depict the three dimensions of a body in their proper relations be invoked. In my own experience I find that many engineers who have had a course in descriptive geometry lack the power to correctly place the third dimension in its relation to the other two. Now, if *they* cannot comprehend a problem involving these points how much less capable must be the average juror. Nor is it at all probable that the average District Court Judge is much more successful in this than the juror, and the result of this failure, and of the fact that many other of the problems involved in "apex suits" are likewise incomprehensible to the judge or juror, is, that the attitude assumed by both judge and jury becomes that of a referee in a game; the victory goes to the side which "plays the game" according to the rule which appeals to that particular judge or set of jurors.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

Data Pertaining to Gas Cleaning at the Duquesne Blast Furnaces

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(New York Meeting, February, 1914)

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It is the object of this paper (1) to deal with the elements in blast-furnace gas from the standpoint of their importance, and the part they are to play in future consumption, and (2) to give detailed information about the construction and operation of a plant designed among the first in this country for washing gas on a large scale.

HISTORY

The conservation and use of blast-furnace gas dates back to the earlier history of pig-iron manufacture. Before this period all the gas was wasted, but the advent of the hot-blast stove and steam blowing engines brought a demand for additional fuel, and the natural source of this was waste gas at the tunnel heads. The regulation of the intensity of the hearth temperature by the preheating of the air has been discussed on many occasions, and the value of the waste gas as a substitute for coal is a definite problem, so only a passing mention may be made of these applications, each being a subject requiring a separate study. The principal fuel in these instances is carbon monoxide together with small percentages of hydrogen and hydrocarbons. The heated air, entering at the tuyères and encountering a great excess of incandescent carbon, combines with sufficient carbon to form the lower carbon oxide and leaves the tuyère area in this condition at a temperature of 3,100° to 3,200° F. This hot, unsaturated gas containing 38 per cent. monoxide passes upward through the stock, giving up its sensible heat, and is partly oxidized to CO₂ by the ore until it leaves the top with an average composition of 12 per cent. CO₂ and 25 per cent. CO. The moisture in the blast will contribute about 3.5 per cent. of hydrogen by its decomposition, and the evaporation of the water in the stock also accounts for about 35 grains of moisture per cubic foot of gas. The hydrocarbons come from the volatile matter in the coke. The gas issues from the top at a velocity varying with the volume of air blown and the size of the tops and outlets, and its temperature is usually from 300° to 400° F. Its constitution also varies with the fuel practice of the furnace. The problem of handling the gas would be a comparatively easy one if the gas leaving the furnace did not carry along mechanically a considerable quantity of what is known as flue dust, as well as a material which afterward sublimes and is extremely hard to remove.

CONSTITUTION OF BLAST-FURNACE GAS

Blast-furnace gas can then be considered as composed of: (1) true gas, (2) moisture, and (3) solids (consisting of (a) solids, and (b) volatile material which sublimes or is soluble in washing water).

Table I shows the average analysis of clean gas for engine purposes at the Duquesne Works for the year 1912.

True Gas

Carbon Monoxide and Dioxide.—As mentioned in the preceding paragraph, the gas relations of CO to CO₂ vary with furnace conditions and practice.

TABLE I.—*Analysis of Blast-Furnace Gas Delivered to the Gas Engines*
Year 1912

Month	Percentage by Volume										Specific Gravity
	CO ₂	O ₂	CO	CH ₄	H ₂	N ₂	CO/CO ₂	B.t.u.		Weight per Cubic Foot Pounds	
								Gross	Net		
Jan.....	12.3	0	25.6	0	3.5	58.6	2.08	94.4	92.7	0.076636	1.00493
Feb.....	12.2	0	25.8	0	3.4	58.6	2.11	94.8	93.1	0.076661	1.00526
Mar.....	12.1	0	25.7	0	3.2	59.0	2.12	93.8	92.2	0.07676	1.00652
Apr.....	12.6	0	25.3	0	3.9	58.2	2.01	94.8	92.8	0.07649	1.00303
May.....	12.4	0	25.7	0	4.3	57.5	2.07	97.4	95.2	0.07614	0.99844
June.....	12.5	0	25.8	0	4.6	57.1	2.06	98.7	96.4	0.07596	0.99608
July.....	12.1	0	26.0	0	4.7	57.1	2.10	99.8	97.5	0.07566	0.99208
Aug.....	12.0	0	26.3	0	4.2	57.5	2.20	99.0	96.9	0.07602	0.99685
Sept.....	12.2	0	26.0	0	4.5	57.3	2.30	99.0	96.8	0.07592	0.99554
Oct.....	12.1	0	26.0	0	3.8	58.1	2.15	96.7	94.8	0.07635	1.00118
Nov.....	12.5	0	25.6	0	3.6	58.3	2.05	94.8	93.0	0.07666	1.00525
Dec.....	12.5	0	25.4	0	3.4	58.7	2.03	93.5	91.8	0.07679	1.00595
Avg.....	12.4	0	25.6	0	3.9	58.1	2.06	95.8	93.8	0.07627	1.00013

Period over which samples were taken averaged 7 hr. 30 min. (approx.).
Calorific value and weight per cu. ft. of gas at 62° F. and 30 in. Hg.
Specific gravity referred to air.

Hydrogen.—The amount of hydrogen varies with the moisture in the blast and the moisture which is decomposed from the water in the stock if, by a slip, the incandescent coke comes in contact with the wet stock. An example of this variation in hydrogen content is given from an experience during a slip as follows:

It was noticed that, following heavy slips on No. 6 furnace, the engines back fired badly. These back fires followed in the length of time occupied by the gas in passing through the system from the furnace to the engines. Several sample tubes were placed in readiness and, on a similar slip, gas samples were taken; the results are given on p. 4. Water to the extent of 8 per cent. of the ore weight was used at this time. These samples were checked three times in the laboratory. The engines back fired and prematured very badly during this slip.

Hydrocarbons.—The amount of hydrocarbons depends on the volatile matter in the coke; some are soluble in the washing water. An analysis of the water of condensation from the gas mains shows that, without a doubt, other organic matter was carried away in the scrubbing. A ferro-cyanide coloration is noticeable on the bottom of the scrubbers, although analysis will not show it.

*Analyses of Blast-Furnace Gas taken at the Inlet to No. 9 Scrubber during
a Slip on No. 6 Furnace*

I	Percentage by Volume		II ^a	Percentage by Volume
Carbon dioxide.....	12.9		Carbon dioxide.....	16.8
Oxygen.....	0.1		Carbon monoxide.....	23.8
Carbon monoxide.....	19.8		Methane.....	0.2
Methane.....	1.6		Hydrogen.....	9.2
Hydrogen.....	38.8		Nitrogen.....	50.0
Nitrogen.....	26.8			
B.t.u. (62° F. and 30 in. Hg), 209.5			B.t.u. (62° F. and 30 in. Hg), 110.8.	

^a Taken immediately after No. I and at the same place.

Water of Condensation.—This water has a strong nauseating smell and taste. It is clear and colorless, with the following analysis, after filtering off 2.9 parts per 100,000 of ferric oxide, Fe_2O_3 :

	Parts per 100,000
Total solids.....	467
Inorganic solids.....	213
Organic solids	254

Iron is combined with an organic acid, is oxidized by air and precipitated. No evidence of cyanides was observed.

Moisture

The amount of moisture in the raw gas varies with the water in the stock. The use of Mesabi ore has made the addition of water a necessity in most cases of hard-driven furnaces in order to prevent dust being blown over, and this water, of course, is evaporated and carried by the gas. The effect of the water in the gas is shown in the following tables and calculations for different conditions and assumes the use of gas free from dust.

Inspection of Table II will disclose the fact that, per pound of dry fuel gas consumed, clean gas at 70° F. moisture saturated at 70° F. gives an available heat of 79.51 per cent. products of combustion cooled to 600° F. Raw gas at 400° F. and 35 grains moisture follows with available heat of 77.03 per cent. products of combustion cooled to 600° F. Partly washed and cooled gas at 125° F. gives an available heat supply of 74.33 per cent. products of combustion cooled to 600° F.

From a calculated thermal standpoint, we conclude, as evidenced by the table under inspection, that washed gas cooled to 70° F. gives the largest percentage of available heat. This is followed by hot un-

TABLE II.—Heat Available per Pound of Dry Blast-Furnace Gas Under Various Conditions of Initial Temperature and Moisture Content. Combustion Air Constant Condition

	Washed Blast-Furnace Gas at 70° F. with Moisture Saturated at 57° F.				Washed Blast-Furnace Gas at 125° F. with Moisture Saturated at 125° F. Combustion Air at 70° F. with Moisture Saturated at 57° F.				Unwashed Blast-Furnace Gas at 400° F. with 35 Grains Moisture per Cu. ft. Gas at 62° F. and 30 in. Hg. Combustion Air at 70° F. Moisture Saturated 57° F.			
	Products of Combustion Cooled to 400° F.	Products of Combustion Cooled to 500° F.	Products of Combustion Cooled to 600° F.		Products of Combustion Cooled to 400° F.	Products of Combustion Cooled to 500° F.	Products of Combustion Cooled to 600° F.		Products of Combustion Cooled to 400° F.	Products of Combustion Cooled to 500° F.	Products of Combustion Cooled to 600° F.	
TOTAL HEAT EXPENDED ABOVE 32° F.												
Fuel Gas.												
Sensible heat in dry fuel gas, B.t.u.....	9.128	9.128	9.128		22.449	22.449	22.449		99.61	99.61	99.61	
Sensible and latent heat in moisture, B.t.u.....	16.682	16.682	16.682		94.420	94.420	94.420		81.27	81.27	81.27	
Latent heat of combustion, 1 lb. dry fuel gas, B.t.u.....	1,253.800	1,253.800	1,253.800		1,253.800	1,253.800	1,253.800		1,253.800	1,253.800	1,253.800	
Total, B.t.u.....	1,279.610	1,279.610	1,279.610		1,380.669	1,380.669	1,380.669		1,434.68	1,434.68	1,434.68	
COMBUSTION AIR.												
Sensible heat in dry air, B.t.u.....	6.292	6.292	6.292		6.292	6.292	6.292		6.292	6.292	6.292	
Sensible and latent heat in moisture, B.t.u.....	7.729	7.729	7.729		7.729	7.729	7.729		7.729	7.729	7.729	
Total, B.t.u.....	14.021	14.021	14.021		14.021	14.021	14.021		14.021	14.021	14.021	
Total heat expended, B.t.u.....	1,293.630	1,293.630	1,293.630		1,394.69	1,394.69	1,394.69		1,448.70	1,448.70	1,448.70	
TOTAL HEAT IN PRODUCTS OF COMBUSTION ABOVE 32° F.												
Sensible heat in dry products of combustion, B.t.u.....	144.65	183.14	228.69		144.65	183.14	228.69		144.65	183.14	228.65	
Sensible and latent heat in moisture, B.t.u.....	31.41	33.89	36.37		117.24	123.12	129.24		94.16	99.07	104.08	
Total heat in products of combustion, B.t.u.....	176.06	217.03	265.06		261.89	306.26	357.93		238.81	282.21	332.77	
HEAT BALANCE.												
Total heat expended, B.t.u.....	1,293.630	1,293.630	1,293.630		1,394.69	1,394.69	1,394.69		1,448.70	1,448.70	1,448.70	
Total heat lost in products of combustion, B.t.u.....	176.06	217.03	265.06		261.89	306.26	357.93		238.81	282.21	332.77	
Total heat available per lb. dry fuel gas consumed, B.t.u.....	1,117.57	1,076.60	1,028.57		1,132.80	1,088.43	1,036.76		1,209.89	1,166.49	1,115.93	
Total heat available per lb. dry fuel gas consumed, per cent.....	86.39	83.23	79.51		81.22	78.04	74.33		83.52	80.31	77.03	
Total heat dissipated in products of combustion, per cent.....	13.61	16.77	20.49		18.78	21.96	25.67		16.48	19.69	22.93	

washed gas under furnace-top conditions carrying 35 grains moisture. The lowest percentage of available heat is shown to exist for washed gas cooled to 125° F. Under conditions of this table it may be interesting to note that thermal equilibrium, raw unwashed gas against washed gas, approximates a temperature midway of 70° and 125° F.

Solids

The solids contained in the gas consist of, first, the mechanically carried particles, of varying degrees of fineness and, second, a volatile substance which is carried as a gas and finally deposited or which may pass through the entire system.

Mechanically Carried Flue Dirt.—The table following shows the analysis of flue dust, using 90 per cent. Mesabi ores, for the year 1912:

Analysis of Downcomer Dust for Year 1912

Month	SiO ₂	Fe	Mn	Phos	H ₂ O	Al ₂ O ₃	Bases	Unflux
January.....	7.30	44.03	0.74	0.060	14.50	2.34	1.90	7.74
February.....	8.26	40.58	0.69	0.070	14.20	2.69	2.38	8.57
March.....	8.18	38.48	0.80	0.068	21.30	2.77	1.64	9.31
April.....	10.36	36.29	0.61	0.066	15.60	2.67	2.47	10.56
May.....	11.90	37.71	0.68	0.082	14.30	2.78	2.78	12.61
June.....	9.66	40.88	0.65	0.071	10.10	2.74	2.38	10.02
July.....	8.89	38.37	0.58	0.102	10.60	2.57	3.54	7.92
August.....	8.01	35.01	0.68	0.101	17.50	2.46	2.73	7.74
September.....	8.66	40.28	0.70	0.118	12.20	2.86	2.85	8.67
October.....	9.34	40.29	0.56	0.093	13.20	2.58	2.25	9.67
November.....	6.27	38.95	0.52	0.054	14.40	2.17	1.51	6.93
December.....	8.77	40.58	0.50	0.071	14.90	2.66	2.17	9.26
Average.....	8.80	39.29	0.64	0.080	14.48	2.61	2.38	9.03

Chemical and Physical Characteristics of Flue Dust.—The gas from the furnace should deposit the heavier material of the flue dust in the dust catcher and pockets, and only the lighter material should pass to the scrubbers. In order to determine if this is true, we took samples of dust from the dust catcher and also samples of the sludges from the settling basins, to ascertain the qualities—both physical and chemical. We first endeavored to sieve the material, after drying, but with no success, and then washed it through screens, weighing that which remained. The screens ran up to 300-mesh, and the tests show that 92 to 95 per cent. of the sludge from the basin passed through the 300-mesh screen. This material is also the richest in metallic yield, showing 41 per cent. iron. It is noticeable that only a low percentage of fine dust, that is through a 300-mesh screen, remained in the dust catcher—18 per cent. in

one case and 5 per cent. in another. This may have been carried down in contact with larger particles; that is, the fine particles would stick on the larger ones, but would be separated by water. I would mention that very interesting data would be revealed if ore were wet sieved instead of dry sieved: a more uniform and more accurate representation of the actual physical condition would then be had. An ore which we sieved dry, ran 9 per cent. through 100-mesh, and the same sample ran 24 per cent. through 100-mesh when wet sieved.

Wet Sieve Analysis

Dust direct from No. 6 dust catcher

Weight grams	Per cent. weight	Remaining on
0.60	0.79	20
2.80	3.71	40
3.10	4.10	60
6.55	8.67	80
2.90	3.84	100
19.60	25.94	150
18.60	24.62	200
7.60	10.06	300
13.80	18.27	Through 300

Total....75.55

Dust direct from scrubber

Weight grams	Per cent. weight
0.17	0.26
0.20	0.31
0.29	0.45
0.20	0.31
0.81	1.25
1.63	2.52
1.40	2.16
60.00	92.74

64.70

Wet Sieve Analysis

Dust direct from scrubber

Weight grams	Per cent. weight	Remaining on
		20
0.05	0.08	40
0.25	0.40	60
0.15	0.24	80
0.15	0.24	100
0.32	0.51	150
0.70	1.12	200
1.45	2.33	300
59.30	95.08	Through 300
		Through 200

Total....62.37

From No. 2 furnace dust catcher

Weight grams	Per cent. weight
18.8	2.62
165.8	23.13
113.9	15.89
175.3	24.45
31.8	4.43
138.2	19.26
34.8	4.85
.....	
.....	
38.2	5.35

Analysis of Flue Dust Direct from Scrubber Water. Wet Sieving

	On 150	On 200	On 300	Through 300
Silica.....	10.85	14.02	13.85	15.00
Iron.....	20.23	26.11	28.89	41.11
Manganese.....				0.72
Alumina.....				7.64
Lime.....				1.55
Magnesia.....				0.15
Ignition loss.....	53.42	39.91	34.55	15.92

Analysis of Settling from Settling Basin

	No. 1 Bay	No. 5 Bay
Silica.....	13.53	11.92
Iron.....	43.00	45.75
Alumina.....	8.00	6.09
Ignition loss.....	11.35	10.00

Settling Properties of Flue Dust.—Bearing in mind the finely divided condition of the dust, as shown in the sieve tests, it will be interesting to note the rate of settling of this material. We have a double settling basin. The water passes from the collecting sewer beneath the scrubbers and flows into one section, filling it with dirt. The dirty water is then diverted to the other section, and the first cleaned by means of a grab bucket, the material being loaded into cars. There are six partitions, or baffles, and wooden gates which tend to hold back the dirt and aid in settling it. Each division of the basin is 161 ft. long and 26 ft. wide, the water flowing over a weir at the end. The capacity is 500 tons of wet dirt. Each side of the settling basin holds 197,920 gal. of water when clear of dirt. The dirty water was allowed to flow through for a day; then, from a platform erected over the first compartment, samples were taken at various depths. A brass tube, fitted with a rubber stopper and a rod, was used to obtain the samples. The closed tube was lowered to the desired depth, the bottom opened and the water allowed to flow in. The stopper was then drawn into place and the sample put in a jar for weighing and measuring. The results are shown in Table III.

Time Settling Tests With Flue Dust and Sludge from Gas-Cleaning Plant.—The sludge in the settling basin is of the consistency of a thick jelly and filters with almost the same rapidity. In this condition it is extremely difficult to handle. We allow the basin to drain as much as possible, then remove the sludge with a grab bucket, and load into cars. These cars must be calked water tight to hold the material. If the material would drain easily most of the trouble would be obviated, and tests were made to locate the difficulty. A flume was first constructed to carry the water from one scrubber and with sufficient fall to create a

TABLE III.—*Amount of Dust at Various Levels and for Different Periods of Settling*

Unsettled Solution				Series A (After settling 18 min.)			
Level Feet	Time	Volume	Grains Dust per Gal. Water	Level Feet	Time	Volume	Grains Dust per Gal. Water
1	A. M.	200 c.c.	37.00	1	3:14	193 c.c.	5.78
2	2:39	229 "	32.00	2	3:17	201 "	4.15
3	2:42	209 "	33.00	3	3:20	185 "	3.97
4	2:45	245 "	37.50	4	3:23	142 "	4.10
5	2:48	184 "	36.50	5	3:26	172 "	4.15
6	2:51	191 "	73.60				

Series B (After settling 36 min.)				Series C (After settling 51 min.)			
Level Feet	Time	Volume	Grains Dust per Gal. Water	Level Feet	Time	Volume	Grains Dust per Gal. Water
1	3:32	188 c.c.	9.46	1	3:47	206 c.c.	3.77
2	3:35	185 "	5.38	2	3:50	181 "	3.20
3	3:38	195 "	3.62	3	3:53	203 "	3.38
4	3:41	233 "	2.26	4	3:56	128 "	2.80
5	3:44	175 "	3.90	5	3:59	171 "	3.22

Series D (After settling 66 min.)				Series E (After settling 76 min.)			
Level Feet	Time	Volume	Grains Dust per Gal. Water	Level Feet	Time	Volume	Grains Dust per Gal. Water
1	4:02	212 c.c.	4.90	1	4:17	221 c.c.	3.45
2	4:05	197 "	3.62	2	4:20	220 "	4.22
3	4:08	193 "	3.85	3	4:23	202 "	4.80
4	4:11	134 "	5.84	4	4:26	213 "	5.15
5	4:14	160 "	3.68	5	4:29	165 "	3.32

fairly rapid flow. Constrictions were built in the flume so as to speed up at different points to carry the lighter material forward. By this scheme we succeeded in making a rough separation, and obtained a fine but granular product from which the water drained easily. The trough was widened out somewhat at the end and the speed of flow decreased. This gave us the same product we had to deal with in the basins, as a lot of fine material was deposited. In order to determine a settling standard for comparison, we took some Group-3 ore, a standard Mesabi, and wet sieved it through the various 60-80-100-150-200-300 mesh screens. The 300-mesh screen was the finest we could get. These different screenings were put into similar bottles filled with water and shaken up. Following is the time of settling of the different sizings, fixed by numerous observations:

Screen	Time of Settling, Seconds
60.....	1½
80.....	3
100.....	4
150.....	7½
200.....	14

The material which passed through 300-mesh required several hours to settle clear. This was also devided by time intervals; that is, the material was shaken and allowed to settle for 2 min., and the water and suspended material poured off. This was repeated until a standard, clearing itself in 2 min., was prepared. Thus different time separations were made. Of course we could not determine the fineness of this material, but could only recognize it as that which settled in a certain interval of time. The sludge obeyed the same rules and that which came down in less than 5 min. filtered with comparative ease. A scum of extremely fine, light material, principally coke dust, extremely gelatinous, forms on top of the basins and afterward sinks. A small quantity of this material, if mixed with the coarse dirt which previously filtered freely, at once prevents all filtration. It is reasonable to suppose that a lot of the dirt could be easily handled if this slime was eliminated. We are at present experimenting with various means to make this separation and expect to do away with many of the difficulties in handling this material and also recover a good product for sintering or charging back. An analysis of the material floating on the top is exhibited:

	Dry	Wet
Silica.....	22.54	
Iron.....	24.40	
Manganese.....	0.61	
Phosphorus.....	0.068	
Moisture.....		57.00
Alumina.....	7.43	
Lime.....	1.52	
Magnesia.....	0.25	
Ignition loss.....	40.84	

Quantity of Dust Carried by Furnace Gas.—An attempt was made to arrive at a correct figure for dirt in the gas entering the scrubbers, and it was decided that 500-c.c. samples of water should be taken at intervals from the overflow water of the scrubber and collected in a carboy. The water was afterward filtered. In this manner and over a period of 19 days, samples were taken, consisting of 2,000 lb. of water, or 240 gal. From our weir readings on the settling-basin overflow, we determined the quantity of water for the period, and by means of recording meters and Pitot-tube readings we determined the gas volume passing through the scrubbers, per gallon of water, and therefore determined the actual dirt represented in 6,240 cu. ft. of gas, which was 2.6 grains per cubic foot. The sampling was done under normal operating conditions. The results shown in the following table are as nearly accurate as it is possible to reach under the conditions at this plant.

Computation of Dust Carried by Furnace Gas for Month of June, 1913

Gas cleaned, cu. ft.....	3,853,044,000
Water consumed, gal.....	152,762,000
Average dust content of scrubber water per U. S. gal., grains.....	60.345
Gas cleaned per hour, $\frac{3,853,044,000}{720}$ cu. ft.....	5,351,45
Water consumed per hour, $\frac{152,762,000}{720}$ gal.....	212,169
Grains dust removed per cu. ft. gas, $\frac{212,169 \times 60.345}{5,351,450}$ grains.....	2.393
Average dust content of primary gas per cu. ft., grains.....	0.20
Grains dust per cu. ft. raw gas (2.392 plus 0.20).....	2.592

Note.—Gas consumed at standard conditions of 62° and 30 in. Hg.

Blast-furnace gas at the scrubbers, from the foregoing determinations, contains 2.592, say 2.6 grains of dust per cubic foot. This dust consists of ore, coke dust, and limestone dust, beside organic acids and compounds which seem to condense and are composed principally of silica and alumina. The gas may be of a dense white nature, or of a reddish hue, due to the predominance of fume in the former case, and ore dust in the latter case. The former is the harder to remove, and almost invariably if the dirt on the filter capsule in the testing apparatus is light in color, the gas is dirtier than if red. Gas after final cleaning still retains some of the whitish fume. This material is deposited in the bottom of the stoves, in the stacks, and on the boiler settings. In places where the gas has only been rough cleaned the color is a pinkish white, without grit and with a feeling similar to talcum powder. The stacks at all times show this white fume, and a determination was made to see how much of the dust left in the gas at the scrubbers passed through the entire system without depositing. Below is that determination and an analysis of the material that was deposited in the stoves and boilers. It is seen that 27.5 per cent. of the dirt shown in the scrubber analysis passes through the entire system.

Wet Sieve Test on Deposit Found in No. 2 Stove, No. 6 Furnace

Remaining On	Weight, Grams	Percentage
10	1.10	1.77
20	0.54	0.87
40	0.54	0.87
60	0.29	0.46
80	0.51	0.82
100	0.62	1.00
150	0.80	1.29
200	0.75	1.24
Through 200	56.85	91.69
Total,		62.00

Nature and Quantity of Dust Passing through System and Exhausted to Atmosphere.—Material that passed through the washing plant with the gas and found in the flue gas of No. 15 boiler, No. 3 boiler house extension—clean gas:

Silica.....	22.80
Iron.....(=Fe ₂ O ₃ 8.57 per cent.)	6.00
Manganese.....(=Mn ₂ O ₄ 2.09 per cent.)	1.50
Alumina.....	15.73
Lime.....	21.42
Magnesia.....	2.50
Potassium oxide.....	20.90
Ignition loss.....	6.08

Deposit Found on the Vanes of the Theisen Washers

Silica.....	23.84
Iron.....(=Fe ₂ O ₃ 9.43 per cent.)	6.50
Manganese.....(=Mn ₂ O ₄ 3.52 per cent.)	2.58
Alumina.....	12.65
Lime.....	15.52
Magnesia.....	1.62
Potassium oxide.....	6.96
Ignition loss.....	23.90

The dust in the gas at the top of No. 3 fan separator was 0.09171 grains per cubic foot; the dust in exit gases and boiler (referred to fuel gas), 0.0252 grains per cubic foot. The amount passing through the system equaled 27.50 per cent. This test was taken when the gas volume was low, which accounts for the low dust content.

The material passing through the final cleaners and deposited on the piston heads and at the several outlet ports of the engine, analyzed as follows:

	Per cent.
SiO ₂	18.40
Mn ₂ O ₄	2.40
Al ₂ O ₃	15.55
CaO.....	12.20
MgO.....	1.68
Fe ₂ O ₃	4.71
Alkalies.....	42.89
Ignition loss.....	2.17

DESCRIPTION OF PLANT

The system of cleaning gas at the Duquesne blast furnaces consists at present of scrubbers, fans, and Theisens. This installation was first

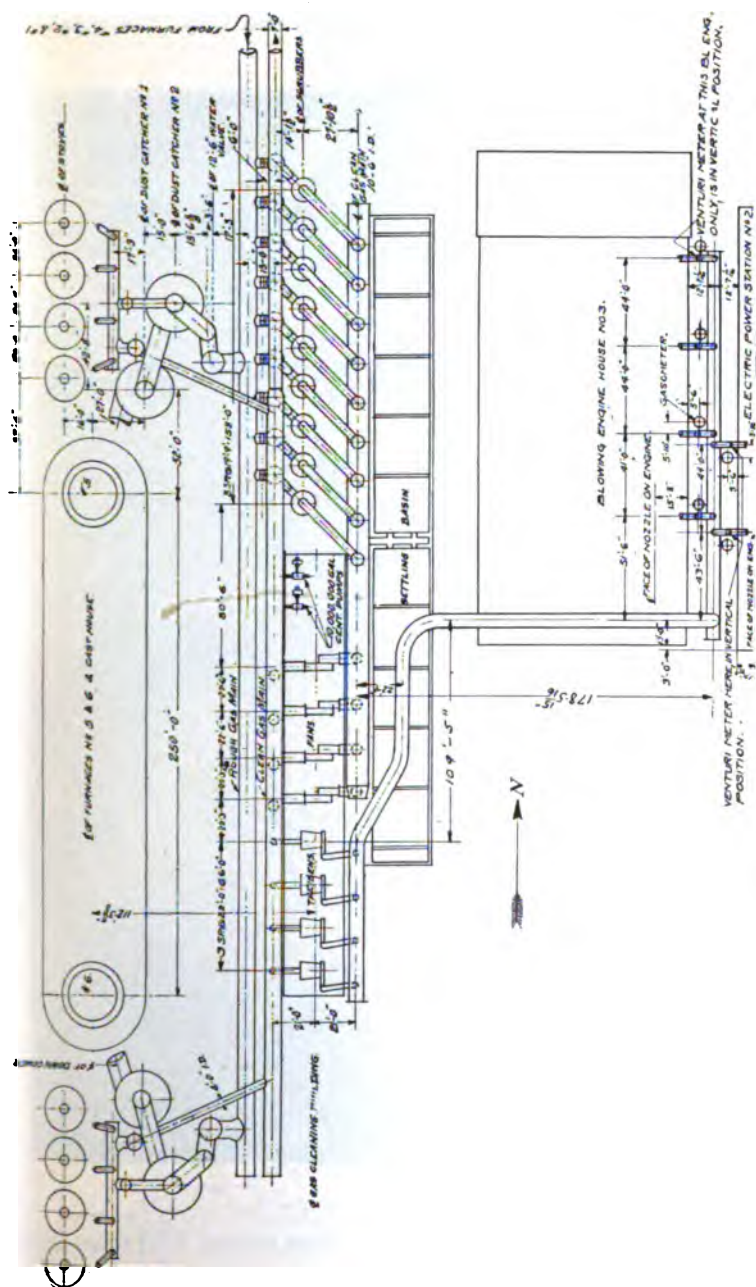


FIG. 1.—GENERAL ARRANGEMENT OF THE PLANT.

designed for the purpose of cleaning gas for six 3,600-h.p. gas engines, four sets of stoves, and 1,500 boiler horse power. At the time of its installation it was the largest in the country and very little reliable data

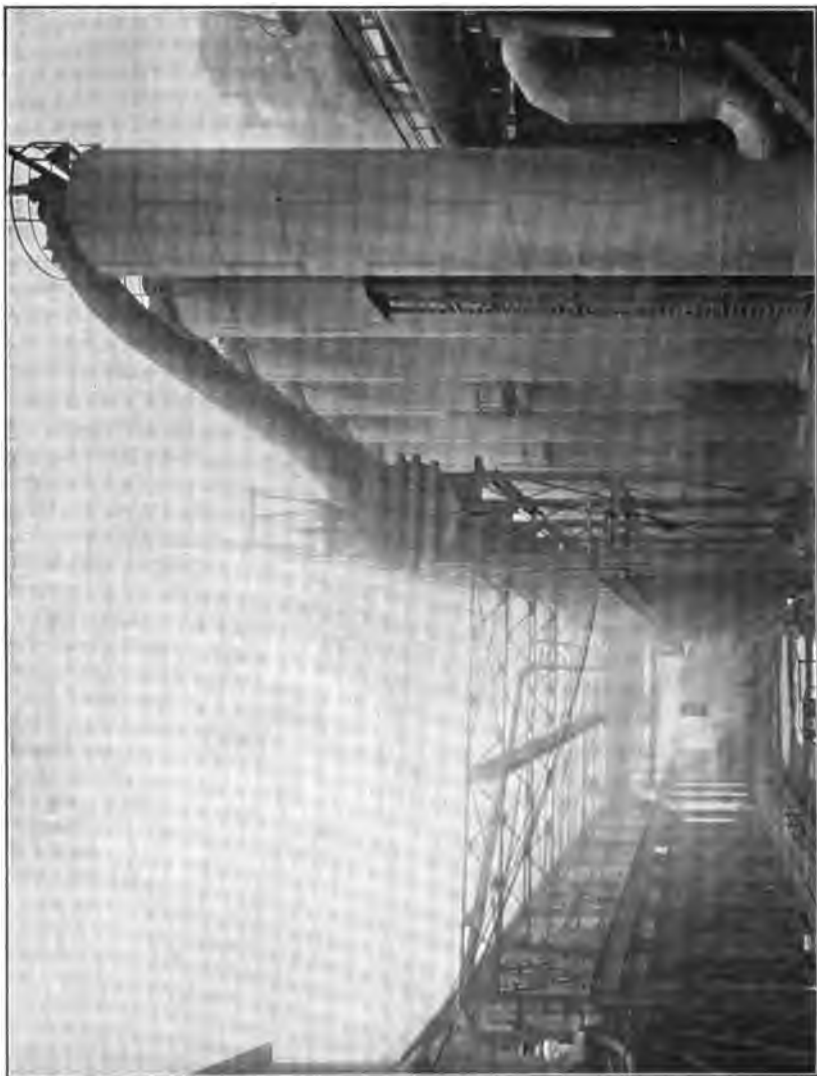


FIG. 2.—DUQUESNE GAS-CLEANING PLANT, SHOWING SCRUBBERS, WITH INTAKE FROM DIRTY-GAS MAIN AND OFFTAKE TO COLLECTING MAIN; ALSO SETTLING BASINS.

could then be collected on the subject. Nine towers, four fans, and four Theisens were erected. After operating a while, numerous changes became necessary. We found that four scrubbers, two fans, and two Theisens were sufficient for the demands. The plant is shown in the

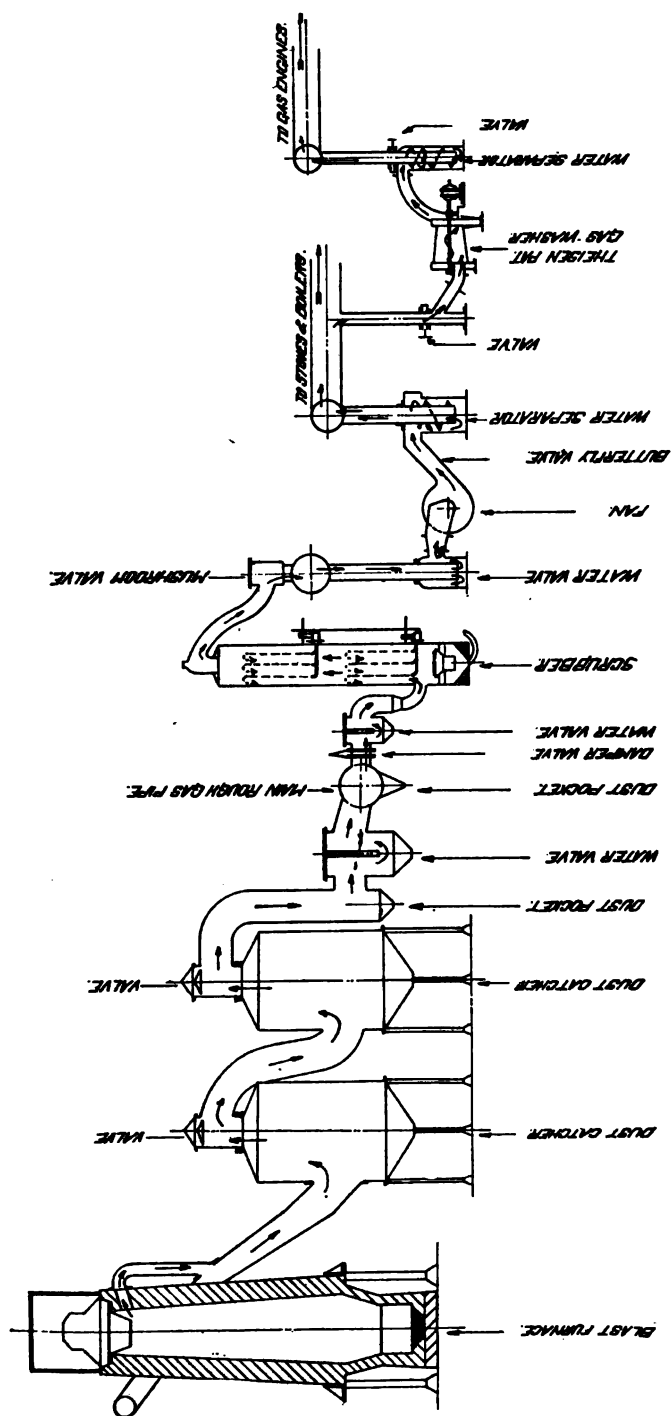


FIG. 3.—THE COURSE OF THE GAS THROUGH THE CLEANING PROCESSES TO STOVES, BOILERS, AND ENGINES.

illustrations as at present. Fig. 1 is a general plan and Fig. 2 a view of the plant. Fig. 3 shows the course of the gas through the cleaning processes.

Scrubbers

We first started with scrubbers having a rotating nozzle on the top, the water to form a rain and thus come in contact with the gas and clean, as well as cool it in this manner. The gas channeled under these conditions, the pressure over the scrubber area being constant; but the ascending gas column, due to velocity head, deflected the water. This was shown clearly by repeated tests, using a pipe with a funnel on one end and a standard pail outside, the time of the filling of the bucket being taken. After the bucket was full, the pipe was withdrawn 4 in. and the filling timed once more. In this way we tested across the scrubber. Distinct dry areas were found, irrespective of all possible regulations of water and deflecting gas schemes. It was then decided to reverse the scheme of having gas the dominating element and put the water in in such manner as to change the direction of the gas in its upward course and at the same time spray it repeatedly. This was done by injecting the water through a valve having a number of openings and a core moving therein. This core successively blocks off the openings to the different nozzles, and thereby temporarily stops the flow of water, causing an area of low pressure directly above the nozzle. When the core has passed, the water resumes and sprays through the gas which has flowed to this point, driving it to the next nozzle, where the same process is continued. The core is revolved electrically, a 5-h.p. motor being ample for four valves—i.e., two scrubbers, or 60,000 ft. of gas. The gas rises through the scrubber at the rate of 4 ft. per second and the water at 60 ft. per second with a head of 35 lb. main pressure. A second valve is placed in the scrubber just above the range of the lower one and operates in the same manner. Screens are placed above the nozzles to break up the water into fine drops so as to bring the gas and water into intimate contact. It is evident that the gas will be repeatedly sprayed as it travels in an upward spiral course until it has passed the range of the water. Fifteen revolutions per minute is found to be about the right speed for the core. Fig. 4 is a section through the scrubbers.

The gas is cooled down very effectually, numerous readings showing only 5° to 6° above the temperature of the ingoing water. An average moisture test shows 0.5 grain above the saturation point at the temperature of the outgoing gas. These tests were taken on the outlet pipe of the scrubbers and were made with the Brady apparatus and also with the Brown apparatus. The difference in temperature between the gas and water can be increased or decreased, depending on the quantity of gas

passing through the scrubbers. We have found that between 25,000 and 30,000 cu. ft. can be passed through our 12-ft. scrubbers, and although the average difference in temperature is higher than that above mentioned, for the year 1912, yet it is the most convenient working point from a financial standpoint, if not so efficient thermally. In other words, the

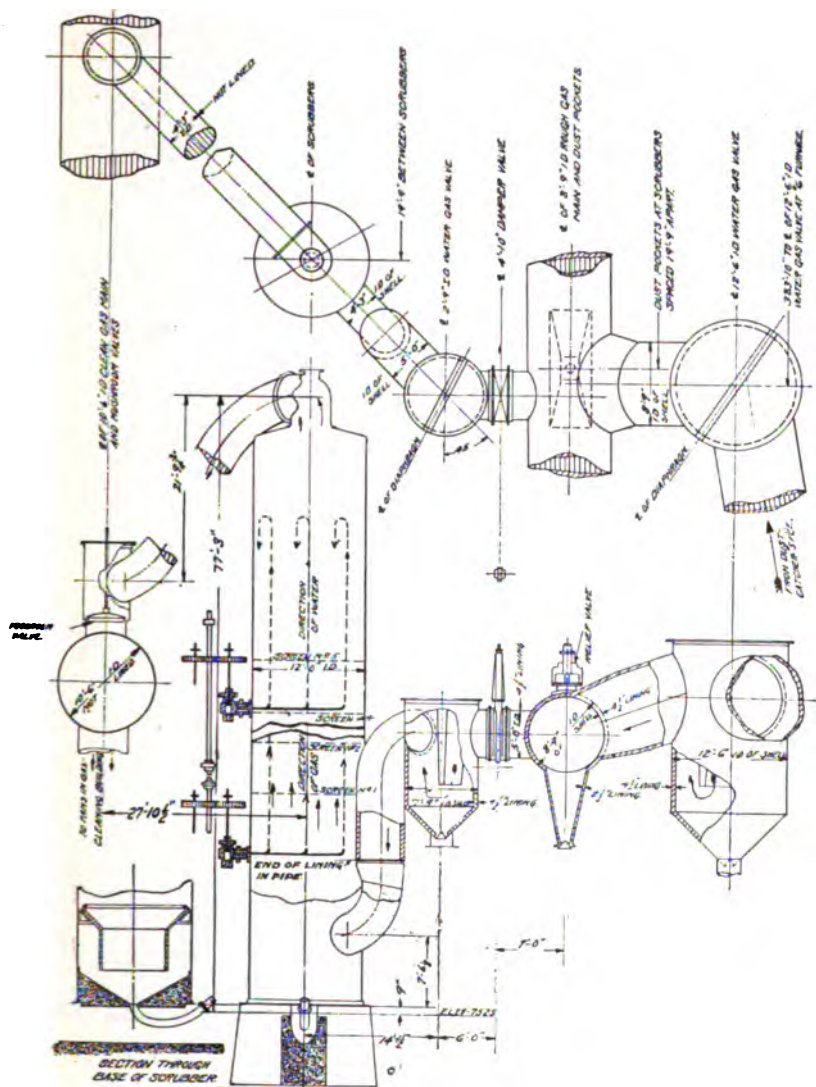


FIG. 4.—SECTION THROUGH THE SCRUBBERS SHOWING THE METHOD AND APPARATUS FOR PRIMARY CLEANING. THE CUT-OFF VALVES ARE ON THE SIDE AND DRIVEN FROM A VERTICAL SHAFT.

moisture content of the gas increases very rapidly above 90° F. and we aim to keep under this as much as possible. The moisture, if not controlled within limits, will lower the stove efficiency to such an extent as to make cleaning prohibitive.

The water is taken from the service mains and is charged at the regular water rate. This condition is different in plants where waste water is used. The Duquesne plant construction is such as to deliver our bosh cooling water to mill condensers and water-purification plants. About 815 gal. of water are used per minute per scrubber. All water flows over a weir, so that the quantities are accurately determined, readings

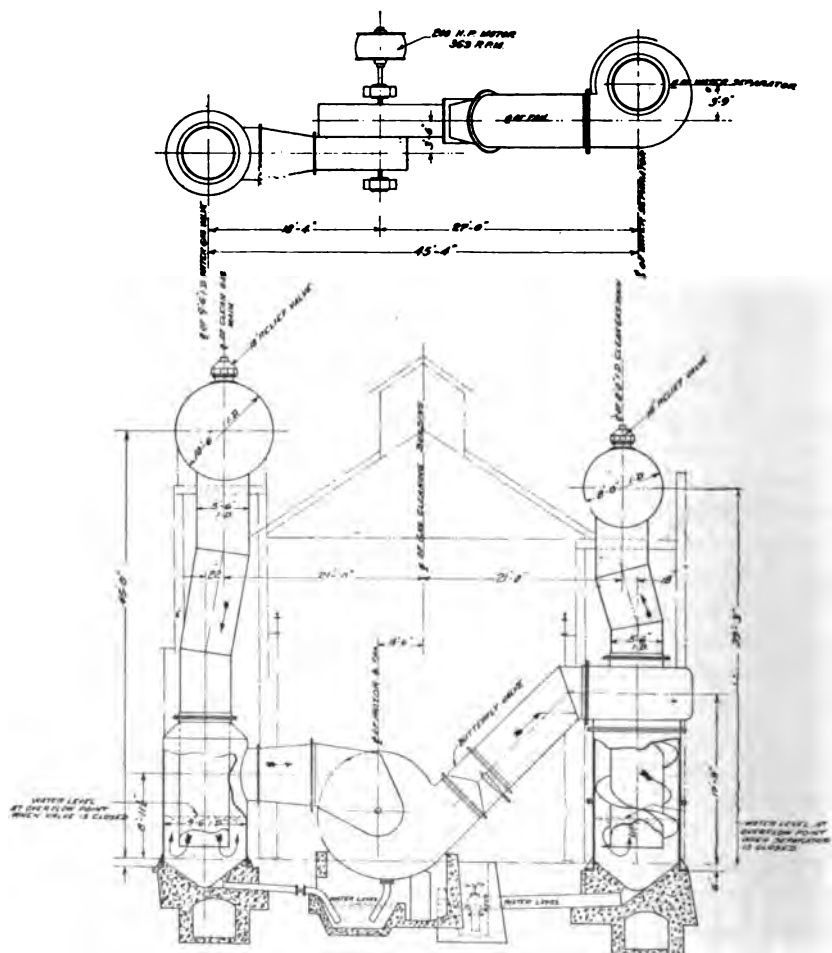


FIG. 5.—SECTION THROUGH FANS, AND METHOD OF PASSING GAS THROUGH SEALS AND SEPARATORS.

being taken four times a day. The electric power is metered, and the quantity of gas to the engines is determined by Venturi recording instruments. The stove gas is determined by Pitot-tube readings, hydro determinator, and Ellison gauge, all results checking closely.

The water used in gas valves and seals is that which is used to keep the

gas from entering the scrubbers not in use and a small stream flows continuously to preserve these seals. All water flows to the settling basins and is therefore recorded in the weir readings.

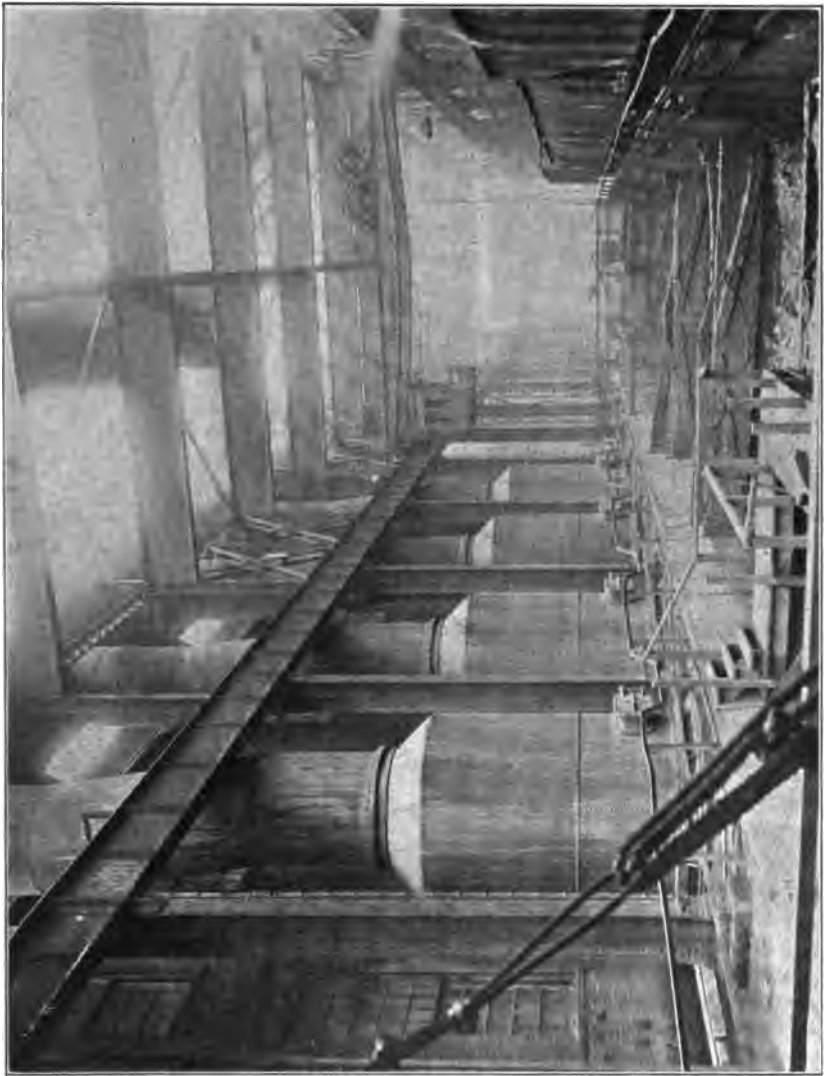


FIG. 6.—WATER SEAL BETWEEN COLLECTING GAS MAIN FROM SCRUBBER AND THE FOUR FANS.

Fans

The fans, Fig. 5, are of Sturtevant type, electrically driven by direct-current and alternating-current motors, and each has a capacity of 84,000 cu. ft. of gas per minute. A water seal, Fig. 6, is placed on either side to provide means for repairing. The seal after the fan is also

a spiral separator for water which may be picked up in the fan. A calked main serves to connect the scrubbers with the fans and is 10 ft.

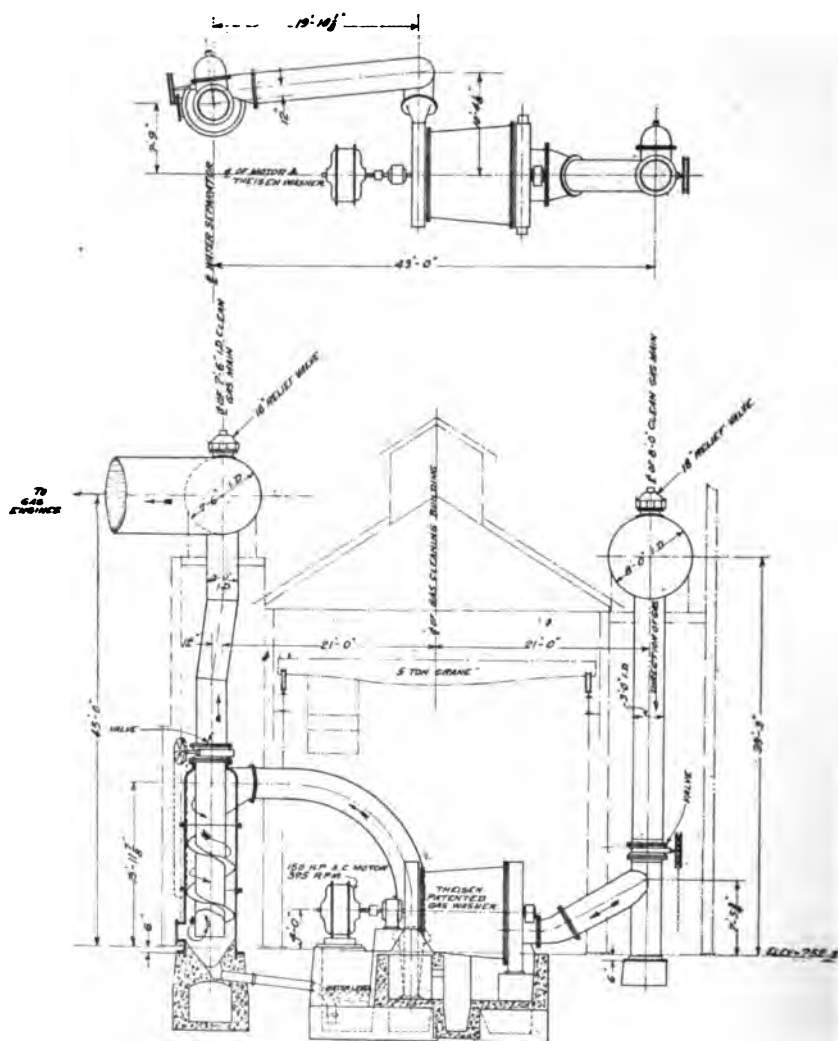


FIG. 7.—SECTION THROUGH THEISENS, SEALS, AND SEPARATORS.

6 in. in diameter. After leaving the fans the gas is carried into an 8-ft. main, or primary gas main, and distributed.

Theisens

The Theisens, Fig. 7, are four in number, three driven by induced and one by direct-current motors. The motors are each 150 h.p. and drive

the rotor 375 rev. per minute. Each Theisen has a capacity of 15,000 cu. ft. of gas per minute. There are seals and separators on the Theisens the same as on the fans.

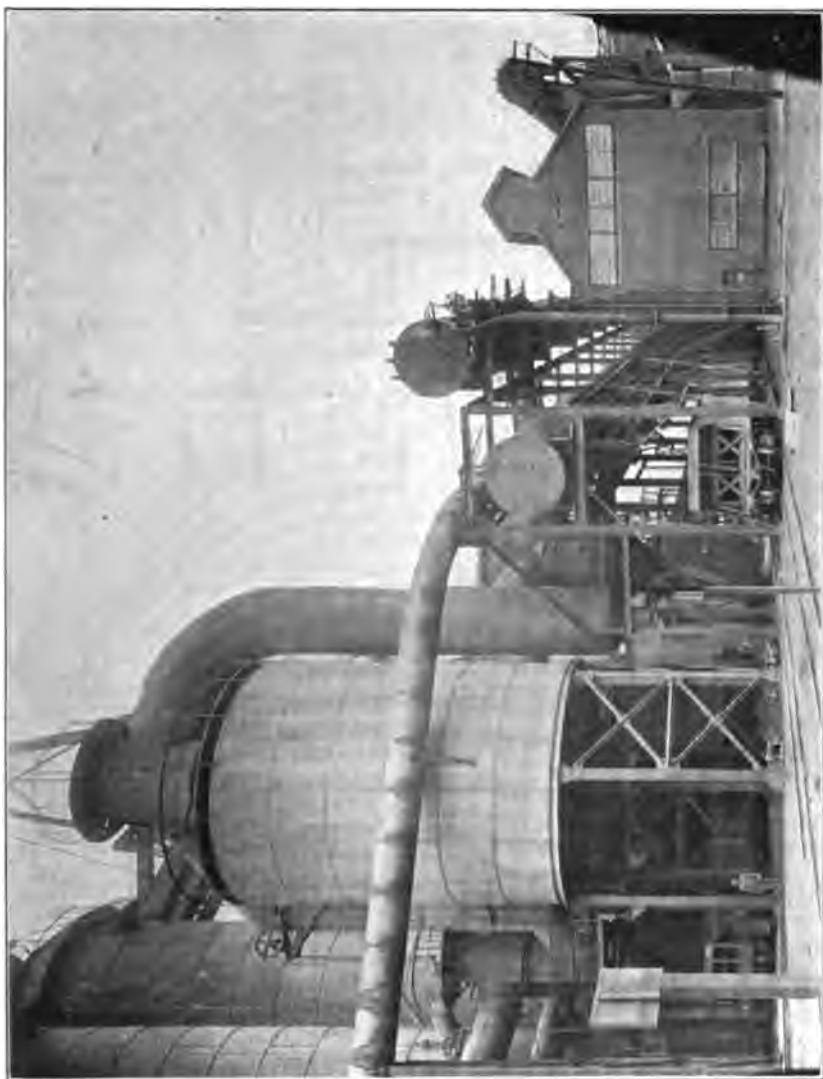


FIG. 8.—DIRTY-GAS MAIN WITH CLEAN-GAS MAIN AT HIGHER LEVEL, ALSO THE END OF THE REFINED-GAS MAIN LEADING FROM THE THEISENS TO THE ENGINES.

Settling Basins

The settling basins are 161 ft. in length, 26 ft. wide, and 11 ft. 6 in. total depth. The settling depth is 6 ft. There are two sections and these are used alternately. The sewer from the scrubbers into which the dirt and water is collected is so situated as to make possible the diversion

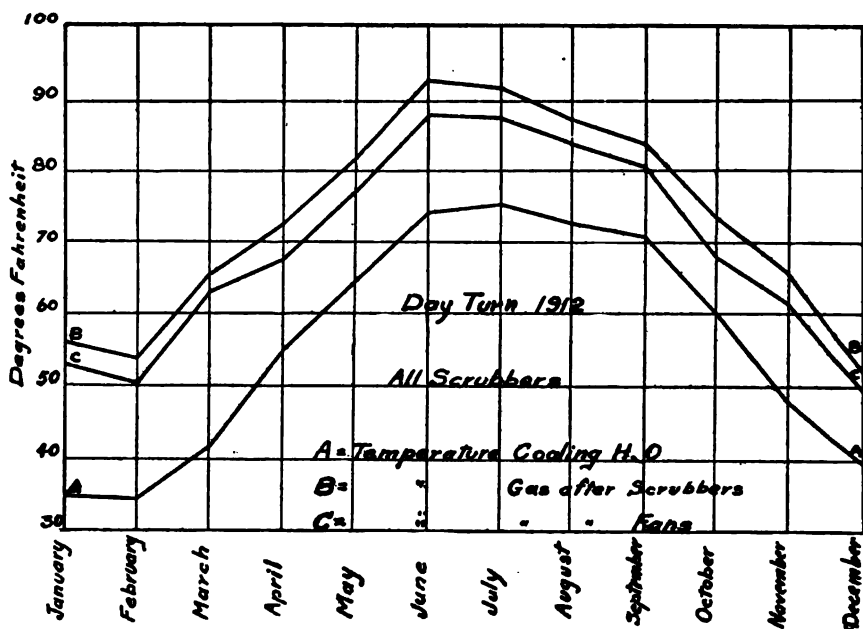


FIG. 9.

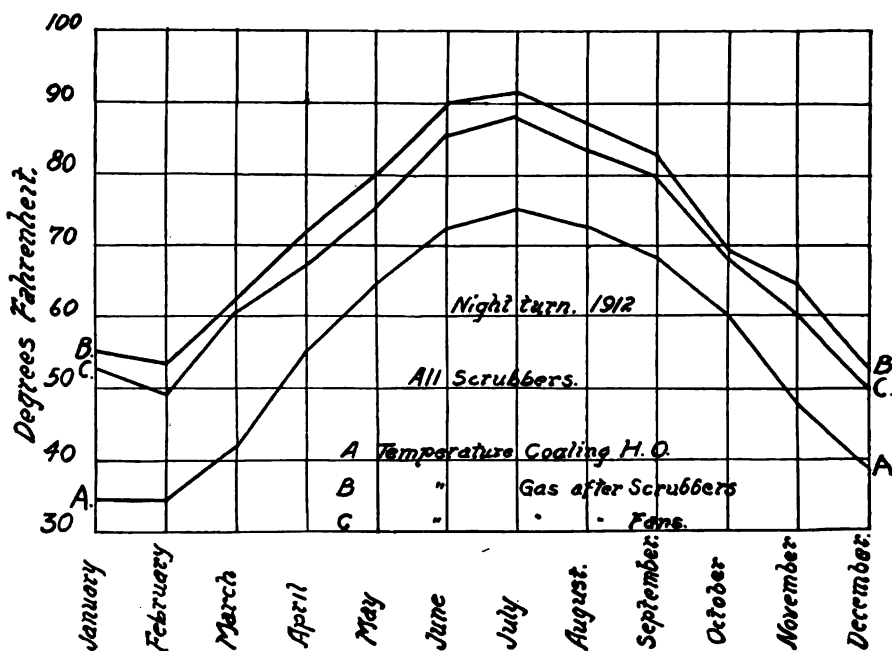


FIG. 10.

of the water to either compartment. When the basins are filled with dirt the water is drained and the material loaded into cars by a grab bucket. Partitions are raised in the basins to aid in the retardation of the dirt.

OPERATION OF PLANT

The gas from six blast furnaces is allowed to go into a general dirty-gas main, 8 ft. 6 in. in diameter, shown in Fig. 8, and from here is distributed

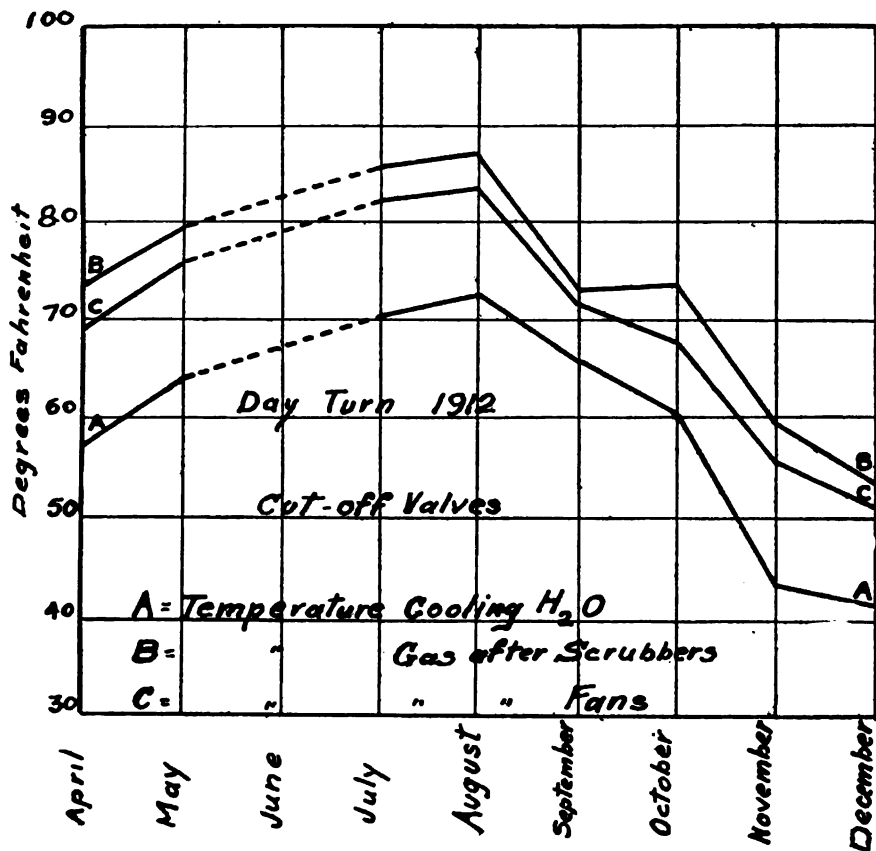


FIG. 11.

to various destinations, as follows: No. 1, 2, 3, and 4 boiler houses, No. 1 and 2 sets of stoves, and gas-cleaning plant. A pressure of 12 in. of water is maintained in this main. The gas to the gas-cleaning plant goes through water valves to scrubbers, in parallel, going only through one and thence to the 10 ft. 6 in. collecting main, from which it goes through the fans to the primary or scrubbed-gas main and is distributed

to No. 3 boiler house extension, stoves on No. 3, 4, 5, and 6 furnaces, and to the Theisens.

The pressure in the receiving main is $10\frac{1}{2}$ in. and in the primary main 8 in. The Theisens boost the pressure to 12 in. and deliver to the engines. These pressures relate to operating without fans, the gas being propelled by the pressure of the furnace. We operated for a period of a

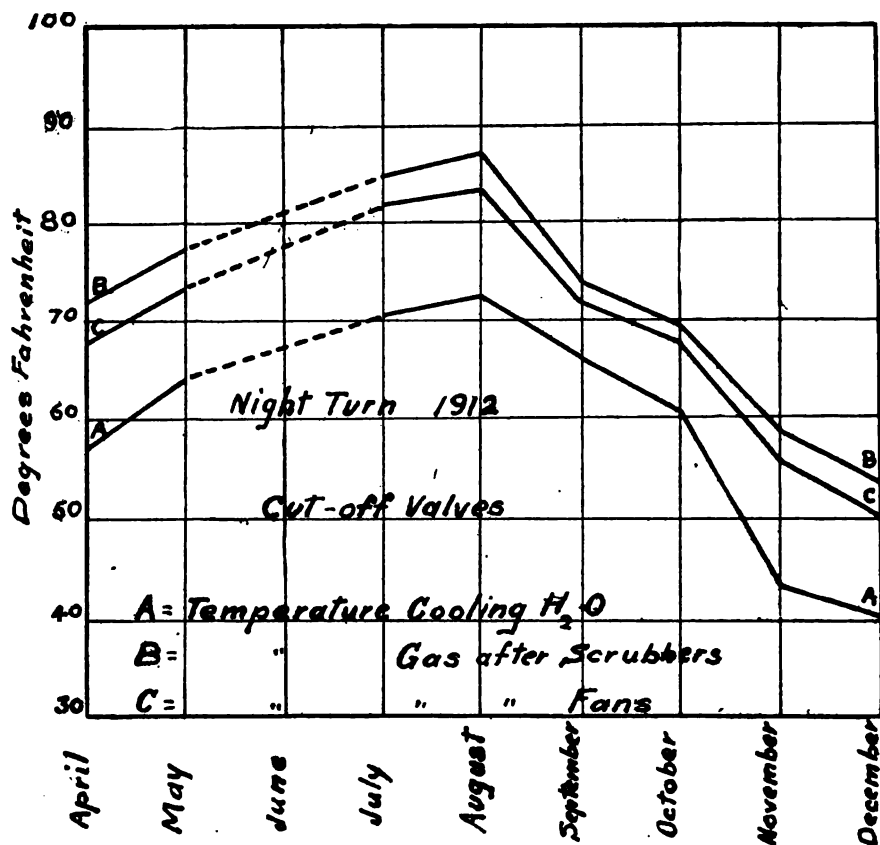


FIG. 12.

year and a half this way, but are using fans now, due to a desire to lower the top pressure, saving gas losses, and lessening the danger in operation during tuyère changing, etc., when a high gas-main pressure causes trouble. We desire a pressure of 5 to 6 in. at the stove burners, for the reason that our distributing mains are too small and we have poor stove draft. The friction through the various fan seals, and through the fans also, made it necessary to back up the gas. In a new installation many of these difficulties would be obviated. Formerly we removed no dirt

in the fans, but we have so altered them lately as to reduce the dirt content from 0.2 to 0.05 and 0.06 grain per cubic foot.

The Theisens, as shown in the gas test for 1912, lower the dirt content to 0.009 grain per cubic foot.

Data of our operating conditions for 1912 are shown in Tables IV to VII and Figs. 9 to 12.

TABLE IV.—*Gas Cleaned and Water Consumed, Primary Plant, Year 1912*

Month	Gas Cleaned	Water Consumed			Cu. ft. Gas Cleaned per Gal. Water Consumed		
		Scrubbers	Water Gas Valves	Total	Scrubbers	Water Gas Valves	Total
Jan.....	4,180,005,000	128,464,000	13,392,000	141,856,000	32.5	304.7	29.5
Feb.....	3,718,755,000	123,424,000	16,096,000	139,520,000	30.1	231.0	26.7
Mar.....	4,044,490,000	128,161,000	13,392,000	141,533,000	31.6	302.0	28.6
Apr.....	3,947,919,000	123,122,000	13,140,000	136,262,000	32.1	300.5	29.0
May.....	3,906,599,000	124,947,000	13,392,000	138,339,000	31.3	291.7	28.2
June.....	3,747,409,000	123,941,000	12,960,000	136,901,000	30.2	289.2	27.3
July.....	3,736,670,000	127,090,000	13,392,000	140,482,000	29.4	279.0	26.6
Aug.....	3,790,763,000	127,760,000	13,392,000	141,152,000	29.7	283.0	26.9
Sept.....	3,885,908,000	146,491,000	12,960,000	159,451,000	26.5	299.8	24.4
Oct.....	4,488,127,000	155,525,000	13,492,000	169,017,000	28.9	332.7	26.6
Nov.....	3,922,370,000	149,861,000	12,960,000	162,821,000	26.2	302.7	24.1
Dec.....	4,141,773,000	151,625,000	17,856,000	169,481,000	27.3	231.9	24.4
				Average	29.7	287.4	26.9

The water shown as used in water-gas valves is only used as a seal and does no washing, so that the figure for scrubbers is the proper one to consider.

TABLE V.—*Recapitulation of Operators' Daily Report, Year 1912*
No. 3, 4, 5, and 6 Scrubbers, Cut-off Type

DAY TURN.	Temperature Cooling Water.			Temperature Gas after Scrubbers.			Temperature Gas after Fans.			Temperature Gas after Thaisens.			Temperature Atmosphere.		
	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.
Month.															
Apr.....	61.0	55.0	57.4	81.0	69.0	73.7	75.0	66.5	69.2	68.0	60.5	64.2	71.5	45.0	59.9
May.....	68.0	59.5	64.1	84.5	73.5	79.3	81.5	70.0	75.8	75.0	65.5	70.7	82.5	64.5	75.2
June.....	73.5	69.0	70.6	85.5	83.0	85.6	85.0	79.5	82.2	78.0	75.0	75.4	77.0	66.0	72.5
July.....	76.0	67.5	73.0	93.0	83.5	87.1	89.0	78.5	83.4	81.5	73.0	78.1	81.5	64.5	73.7
Aug.....	70.0	60.0	66.2	77.0	67.5	73.3	76.0	65.0	71.7	74.5	66.0	70.8	73.0	52.0	63.5
Sept.....	67.0	52.0	60.6	77.5	62.5	73.6	76.0	60.0	68.0	74.0	57.5	56.9	75.5	47.0	60.3
Oct.....	46.0	40.0	43.2	70.5	53.0	59.8	65.5	48.0	55.9	54.0	46.0	50.3	54.0	35.0	43.0
Nov.....	47.0	37.0	41.1	67.0	43.0	53.7	64.0	41.0	51.2	55.0	41.0	47.3	62.0	19.0	41.3
Dec.....															
Average.....			59.5			73.3			69.7			65.3			61.2
NIGHT TURN.															
Apr.....	61.5	55.0	57.0	77.5	69.0	71.8	73.0	65.5	67.6	67.0	61.0	63.6	59.5	44.0	52.7
May.....	68.0	60.0	64.1	82.0	70.5	77.1	77.5	68.0	73.4	74.5	65.0	70.3	71.0	60.5	65.3
June.....	72.0	69.0	70.3	88.5	83.5	84.4	84.0	80.5	81.4	78.0	75.5	76.6	73.0	64.5	68.2
July.....	76.0	67.0	72.3	93.5	80.0	86.9	86.5	78.0	83.1	82.5	73.0	77.9	76.0	57.0	68.3
Aug.....	70.0	59.5	66.0	78.5	66.5	73.8	77.5	65.0	71.4	75.0	64.0	69.8	67.0	48.0	58.2
Sept.....	69.5	52.0	60.4	76.0	60.0	69.2	74.5	59.5	67.2	69.0	57.5	65.6	72.0	44.0	55.2
Oct.....	56.0	39.5	43.0	64.0	54.0	58.7	60.0	50.0	55.5	52.5	45.5	50.0	54.0	30.0	40.7
Nov.....	45.0	36.0	40.2	62.0	47.0	53.5	59.0	42.0	50.0	54.0	42.0	47.7	62.0	18.0	38.7
Dec.....															
Average.....			59.2			71.9			68.7			65.2			55.9

TABLE VI. —Recapitulation of Operators' Daily Report, Year 1912
All Scrubbers, Primary

DAY TURN.	Month.	Temperature Cooling Water.			Temperature Gas after Scrubbers.			Temperature Gas after Fans.			Temperature Gas after Tholsens.			Temperature Atmosphere.		
		Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.
	Jan.....	40.0	33.5	34.8	70.0	46.5	55.7	63.0	45.0	52.8	50.0	38.5	42.6	48.0	55.0	25.0
	Feb.....	38.5	33.5	34.6	63.5	45.5	53.3	56.5	43.0	50.0	48.0	45.0	43.0	51.5	45.0	29.0
	Mar.....	49.0	35.0	41.2	94.5	54.5	65.7	86.5	53.0	62.9	63.0	44.0	51.0	56.5	20.0	40.0
	Apr.....	61.0	48.5	54.9	85.0	63.0	72.4	75.0	59.0	67.6	68.0	59.0	62.0	76.0	38.0	58.2
	May.....	71.0	58.0	64.0	93.0	73.0	81.2	93.0	72.0	77.0	85.0	65.5	71.1	83.0	83.5	68.9
	June.....	75.5	69.0	73.9	93.0	83.5	92.6	90.0	81.0	87.8	82.0	75.0	81.1	90.5	82.5	75.3
	July.....	81.0	70.0	75.1	100.0	83.0	91.3	96.0	81.0	87.5	87.0	75.0	81.4	87.0	70.0	77.6
	Aug.....	76.0	67.5	72.6	93.0	83.5	87.0	89.0	78.5	83.4	81.5	73.0	78.1	81.5	64.5	74.0
	Sept.....	75.0	60.0	70.7	92.0	67.5	83.5	87.5	65.0	80.2	80.0	65.0	76.2	84.0	52.0	72.7
	Oct.....	67.0	52.0	60.2	78.0	62.5	73.6	76.0	60.0	67.9	71.5	57.5	65.7	75.5	47.0	60.2
	Nov.....	55.0	40.0	47.5	79.0	53.0	65.4	71.5	48.0	61.4	62.0	46.0	54.7	63.0	35.0	47.6
	Dec.....	47.0	36.0	39.1	67.0	43.0	51.9	64.0	41.0	49.6	55.0	41.0	46.1	62.0	19.0	39.8
	Average.....			55.8			72.8			69.0				62.8		
NIGHT TURN.	Jan.....	40.0	33.0	34.6	67.5	45.5	54.9	63.0	44.5	52.2	48.5	41.0	43.5	44.0	3.0	23.6
	Feb.....	39.0	33.0	34.6	62.5	45.5	52.8	61.0	42.5	48.9	49.0	40.5	43.1	44.5	7.0	26.8
	Mar.....	49.5	35.0	41.2	75.0	52.5	61.7	71.0	51.0	60.6	57.5	43.5	53.6	56.5	19.0	37.5
	Apr.....	61.5	49.0	55.0	84.0	60.0	71.6	76.0	57.0	66.9	67.0	55.0	61.8	65.0	33.0	52.5
	May.....	72.0	59.0	64.0	86.0	70.5	79.6	82.0	70.0	74.9	78.0	65.0	70.4	77.0	52.0	62.5
	June.....	77.0	68.5	72.2	97.0	84.0	89.8	90.0	80.0	85.0	83.0	75.0	78.3	74.5	52.0	62.6
	July.....	80.5	70.0	74.9	98.5	83.5	91.2	99.0	80.5	87.5	87.5	75.5	81.4	79.5	64.5	71.7
	Aug.....	76.0	67.0	72.3	93.5	80.0	86.8	88.0	78.0	83.1	82.5	73.0	77.9	76.0	57.0	68.5
	Sept.....	74.0	59.5	68.1	90.0	66.5	82.4	86.5	65.0	79.2	80.0	64.0	75.3	75.5	48.0	66.5
	Oct.....	69.5	51.0	60.0	76.0	61.0	69.3	74.5	59.5	67.3	69.0	57.0	65.3	72.0	44.0	55.0
	Nov.....	55.0	39.5	47.1	72.5	54.0	64.4	71.0	50.0	60.9	62.0	45.0	54.1	63.5	30.0	44.7
	Dec.....	45.0	36.0	48.3	66.0	45.0	52.8	66.0	42.0	49.5	54.0	42.0	46.0	62.0	18.0	37.9
	Average.....			55.2			71.4			68.0				62.6		

TABLE VII.—*Difference in Temperature Between Cooling Water and Off-Coming Gas-Scrubbers, Year 1912*

Month	Day Turn		Night Turn	
	All Scrubbers.	Cut-off Type.	All Scrubbers.	Cut-off Type.
Jan.....	20.9	—	20.3	—
Feb.....	18.7	—	18.2	—
Mar.....	24.5	—	20.5	—
Apr.....	17.5	16.3	16.6	14.8
May.....	16.8	15.2	15.2	13.0
June.....	18.7	—	17.6	—
July.....	16.2	15.6	16.3	14.4
Aug.....	13.4	14.1	14.5	14.6
Sept.....	16.8	8.1	14.3	6.8
Oct.....	13.4	13.0	9.3	8.8
Nov.....	17.7	16.6	17.3	15.7
Dec.....	12.8	12.6	14.5	13.3
Average.....	17.3	13.8	17.0	12.6

During part of the year 1912 we used some of our original scrubbers and part of our remodeled water cut-off type. The difference in temperature is notable. The cut-off type is included in the first column, "All Scrubbers," and tends to lower the average temperature, while the second column shows the temperature using the cut-off type alone.

Daily Observations

A daily log sheet of all important temperatures, pressures, and general performance of the plant is maintained. The daily observations may be summarized as follows:

Temperatures of the gas in mains throughout the plant.

Temperature of the atmosphere. (Recording.)

Temperature of cooling water.

Temperature of the waste water from scrubbers, fans, and Theisens.

Weir readings on settling-basin and fan-house discharges.

Pressures of gas in mains. (Recording.)

Psychrometric readings. (Atmosphere, sling-sychrometer.)

In addition to the above, notation is made on the log sheet as to general operating conditions, such as the number of scrubbers, fans, and Theisens in operation; number of stoves and boilers on clean gas; and number of gas power and blowing engines operating. At the end of each month, the data are compiled and averages struck. Reports are submitted to the various department heads interested.

Sampling Gas for Analysis

An average sample of the gas delivered to the gas engines is drawn over a period of 7 hr. each day. This is transmitted to the laboratory for analysis, which is reported back along with the calculated heat value in



FIG. 13.—APPARATUS FOR TAKING GAS SAMPLES.

terms of net B.t.u. per cubic foot at 62° F. and 30 in. Hg. The apparatus used in drawing such sample, which we have found very convenient and entirely satisfactory for the purpose, is shown in Fig. 13.

Conducting and Reporting Tests

In conducting and reporting the various tests, we have endeavored to follow a standard method of procedure and form. A complete scrubber

CARNEGIE STEEL COMPANY—DUQUESNE WORKS

REPORT OF TESTS AT GAS CLEANING PLANT

Test No. 128

10/4, 1910

DESCRIPTION OF TEST Dust determination on clean gas delivered to the gas engines.

MOISTURE AND DUST DETERMINATION—DATA

Gas Sampled at #1 Theisen Separator.
 Cubic Feet Gas Metered 313.991
 Temperature " " °F 78.0
 " " In Main " 76.6
 " " Inlet Water " 71.0
 " " Outlet " " "
 Pressure Gas in Meter (Ins. Water) -5.55 Hg.
 " " Main " " 10.0
 Barometric Reading (Ins. Mercury) 29.40
 Temperature Atmosphere °F "
 Weight of Dust as Sampled (Grammes) .1791
 " " Moist " "
 Duration of Test 9:05 A.M. to 2:45 P.M.

2 Gas Power and 3 Gas Blowing engines operating.

RESULTS

Dust per Cubic Foot Dry Gas at 78.0	°F and 29.40	Ins. Mercury .00909	Grains
" " " " " 76.6	" " "	" " .00911	"
" " " " " 62.0	" " "	" " .00936	"
Moist " " " " "	" " "	" " "	"
" " " " " "	" " "	" " "	"
" " " " " "	" " "	" " "	"

RESULTS { 62 °F and 30.00" Mercury }
 Grains per Cubic Foot Dry Gas Moisture Dust .00955

VOLUME OF GAS—DATA

Reading taken at	Velocity Pressure, Pitot Tube, (Ins. Water)
Temperature of Gas in Main °F	Barometric Reading (Ins. Mercury)
Pressure of Gas in Main (Ins. Water)	Dry Gas Factor at 78.0 °F .968

RESULTS

Wet Gas per Min. at	°F and	Ins. Mercury	Cubic Feet
Dry " " " "	" " "	" " "	" "
" " " " "	" " "	" " "	" "
" " " " "	" " "	" " "	" "

ANALYSIS OF GAS—

Gas Sampled at Outlet #1 Theisen.
 Carbon Dioxide, CO₂ (% by Volume) 11.8
 Ethylene, C₂H₄ " " 0.0
 Oxygen, O₂ " " 0.0
 Carbon Monoxide, CO " " 20.4
 Methane, CH₄ " " 1.4
 Hydrogen, H₂ " " 4.4
 Nitrogen, N₂ " " 62.0
 Heat Value, Calculated, B.T.U. 95.3

CALORIMETER DATA—

Temperature Room °F	
" Gas "	
" Exhaust "	
" Inlet Water °F	
" Outlet " "	
Pressure Gas (Ins. Water)	
Barometric Reading (Ins. Mercury)	
Lbs. Water Heated	
Cubic Feet Gas Metered	
Heat Value, B. T. U. 62 °F and 30 Ins. Mercury	

test consisting of 71 items is shown in Table VIII. This manner of reporting such tests and the data shown, seem to cover pretty thoroughly the performance of the unit, including those items of observation which are of paramount importance, and we have found it most valuable.

I believe some such method should be pursued in making all washer tests at the various plants and would suggest a form similar to the one here shown. It would be of considerable advantage to have all tests reported in such a manner that they become comparative, which, of course, was our primal object in this case.

There is shown, also, a standard form for reporting dust and moisture determinations on engine gas. We find that this supplies just the data, as to performance, that we desire.

TABLE VIII.—*Standard Form for Gas-Washer Test. Carnegie Steel Co., Duquesne Furnaces. Gas-Cleaning Plant.*

Item	DESCRIPTION	Tower Washer Number 4
<i>General.</i>		
1	Date of test.....	7/19/1913
2	Duration of test.....	Hrs. Min. : 46
3	Type of washer.....	Tower.
4	Inside Diameter.....	Ft. 12
5	Height.....	Ft. 76
6	Area.....	Sq. Ft. 113.1
<i>Temperatures.</i>		
7	Temperature of gas entering washer.....	F. 475.0
8	Temperature of gas leaving washer.....	F. 81.4
9	Temperature of water entering washer.....	F. 74.0
10	Temperature of water leaving washer.....	F. 112.4
<i>Pressures.</i>		
11	Pressure in system before washer.....	Ins. Water 9.0
12	Pressure in system after washer.....	Ins. Water 5.0
<i>Quantities.</i>		
13	Water consumed per minute in washer.....	Gallons 875.0
14	Volume of gas passing through washer per minute at 62° F. and 30 in. Hg.....	Cu. Ft. 22,800.0
15	Weight of dust per cubic foot of dry gas at 62° F. and 30 in. Hg., entering washer.....	Grains 2.6517
16	Weight of dust per cubic foot of dry gas at 62° F. and 30 in. Hg., leaving washer.....	Grains 0.2257
17	Moisture per cubic foot dry gas at 62° F. and 30 in. Hg., entering washer.....	Grains 38.4042
18	Moisture per cubic foot dry gas at 62° F. and 30 in. Hg., leaving washer.....	Grains 11.7004
19	Moisture per cubic foot of gas at observed conditions, leaving washer.....	Grains 10.2862
20	Moisture per cubic foot of gas, saturated at observed conditions, leaving washer.....	Grains 11.4154
<i>Economic Performance.</i>		
21	Volume of gas passing through washer per sq. ft. of area, at 62° F. and 30 in. Hg.....	Cu. Ft. 20.16
22	Volume of gas passing through washer per gallon of water consumed, at 62° F. and 30 in. Hg.....	Cu. Ft. 26.06
23	Difference in temperature between gas entering and leaving washer.....	F. 393.6
24	Difference in temperature between water entering and leaving washer.....	F. 38.4
25	Possible depression in gas temperature.....	F. 401.0
26	Actual depression in gas temperature.....	F. 393.6
<i>Heat Interchange.</i>		
27	Heat in dry gas entering washer per minute (Above 32° F.)....	B.T.U. 194,418
28	Heat in water vapor entering washer per minute (Total above 32° F.)....	B.T.U. 160,090
29	Heat in dust entering washer per minute (Above 32° F.)....	B.T.U. 579
30	Total heat above 32° F. entering washer per minute.....	B.T.U. 355,087
31	Heat in dry gas leaving washer per minute (Above 32° F.)....	B.T.U. 20,690
32	Heat in water vapor leaving washer per minute (Total above 32° F.)....	B.T.U. 41,697
33	Heat in dust leaving washer per minute (Above 32° F.)....	B.T.U. Nil.
34	Total heat above 32° F. leaving washer per minute.....	B.T.U. 62,387
35	Heat transferred to water and radiated.....	B.T.U. 292,700

TABLE VIII.—*Standard Form for Gas-Washer Test. Carnegie Steel Co., Duquesne Furnaces. Gas-Cleaning Plant. (Continued.)*

Item	DESCRIPTION	Tower Washer Number 4
36	Heat transferred to water as shown by observed rise in temperature.	B.T.U. 280,560
37	<i>Efficiencies.</i> Efficiency of washer as dust remover.....	% 91.49
38	Efficiency of washer in lowering temperature of gas.....	% 98.15
	<i>Gas Analysis, Gravity, etc.</i>	
39	Carbon Dioxide, CO ₂	% Vol. 12.8
40	Ethylene, C ₂ H ₄	% Vol. 0.0
41	Oxygen, O ₂	% Vol. 0.0
42	Carbon Monoxide, CO.....	% Vol. 25.8
43	Methane, CH ₄	% Vol. 0.0
44	Hydrogen, H ₂	% Vol. 4.0
45	Nitrogen, N ₂	% Vol. 57.4
46	Weight per cubic foot dry gas at 62° F. and 30 in. Hg.....	Pounds 0.07651
47	Specific gravity referred to air.....	1.0033
48	Calorific value of gas per cubic foot, gross, at 62° F. and 30 in. Hg.	B.T.U. 96.7
49	Calorific value of gas per cubic foot, net, at 62° F. and 30 in. Hg.	B.T.U. 94.7
	<i>Moisture and Dust Determination.</i>	
50	Gas sampled at.....	After Washer
51	Volume of gas metered.....	Cu. Ft. 12.923
52	Temperature of gas at meter.....	F. 82.7
53	Pressure of gas in meter.....	Ins. Water]
54	Barometer reading.....	Ins. Hg.] 29.18
55	Weight of dust as sampled.....	Grams 0.1708
56	Weight of moisture as sampled.....	Grams] 8.2044
57	Dry gas factor at meter temperature.....	% 0.963
	<i>Volume of Gas.</i>	
58	Reading taken at.....	
59	Temperature of gas in main.....	 81.4
60	Velocity pressure, Pitot Tube.....	Ins. Water
61	Pressure gas in main.....	Ins. Water 5.0
62	Barometer reading.....	Ins. Hg. 29.18
63	Dry gas factor at main conditions of temperature.....	F. 0.963
64	Area of main at point where Pitometer is used.....	Sq. Ft.
65	Velocity of gas per minute at point of measurement.....	Ft.
66	Volume of gas per minute at main conditions.....	Cu. Ft.
	<i>Miscellany.</i>	
67	Pumps, power requirements, average for test.....	K.W.H.
68	Washer, power requirements, average for test.....	K.W.H.
69	Cost of power per kw-hr.....	Cents
70	Cost of water per 1,000 gallons.....	Cents
71	Water pressure required for washers.....	Lb. per sq. in. 45 to 50

Moisture and Dust Determinations

Until within the last six or seven months, all moisture and dust determinations were made with the Brady apparatus. We found that the results on dirty gas could not be depended upon and were very inconsistent at times, and therefore adopted the method of dust estimation in dirty gas mentioned elsewhere in this paper. On clean gas, however, the results obtained by the use of the Brady apparatus are sufficiently reliable for all practical purposes. We are using the apparatus as originally designed, but we understand that it has since been considerably improved and that more reliable and accurate determinations are possible. Some few months ago a number of determinations of dust and moisture, Table IX, were made with the Brown apparatus, the results of which indicated accuracy and reliability. On engine gas we seem to get very good results with the Brady apparatus, encountering very little difficulty

and inconvenience in manipulation, at the same time taking every precaution to minimize error.

In making dust and moisture determinations it has been our aim to sample the gas at the same velocity as the gas flowing in the mains. With our present Brady apparatus, we find this a little tedious on clean gas to stoves and boilers, requiring constant attention and adjustment on the part of the observer. However, on engine gas we encounter very little inconvenience from this source and have been able to maintain the proper velocity with comparative ease.

The first dust and moisture determinations were made using volumetric tanks in determining gas volumes, which we found to be very inconvenient and decidedly limited. These we have replaced by wet test meters and as a result have sampled as much as 500 cu. ft. during a single test.

We have not made regular or daily determinations of dust and moisture, since we have not been inconvenienced by excessive dust deposition, either in our clean mains or the engine cylinders. Tests indicate a very uniform dust content in clean gas, with the moisture

TABLE IX.—*Résumé of Dust Tests Run with Brown Apparatus*

Date of Test	Cu. ft. Gas Metered	Tempt. Gas at Meter ° F.	Tempt. Gas at Main ° F.	Barometer In. Hg.	Press. Gas in Main In. Water	Duration of Test Minutes	Dust as Weighed Grams	Dust per cu. ft. Dry Gas at 62° F. and 30 in. Hg.	Point at Which Sample was Taken
7/16/13	9.962	76.50	82.20	29.40	8.00	30.0	0.0995	0.166	After Scrubber
7/17/13	22.843	83.80	80.30	29.17	9.00	105.0	0.3440	0.259	
7/18/13	8.590	84.70	86.10	29.18	8.00	30.5	0.0910	0.183	
7/18/13	9.537	87.00	87.00	29.18	7.00	35.5	0.1122	0.204	
7/19/13	9.135	80.60	80.40	29.18	6.00	33.0	0.0845	0.209	
7/19/13	12.923	82.70	81.40	29.18	6.00	43.0	0.1545	0.205	
7/21/13	3.867	80.00	79.30	29.30	7.00	15.0	0.0725	0.317	
7/22/13	5.055	88.70	81.40	29.40	6.00	21.0	0.0780	0.289	
7/22/13	6.620	86.50	81.00	29.37	6.00	23.5	0.0903	0.248	
7/31/13	7.205	93.00	86.00	29.30	5.00	25.0	0.0678	0.160	
Average								0.224	Before Fan
7/23/13	4.524	75.40	85.50	29.28	6.00	21.00	0.0635	0.235	
7/24/13	6.776	78.00	97.40	29.20	7.00	26.0	0.0980	0.244	
7/24/13	5.098	77.00	102.00	29.20	6.00	18.5	0.0555	0.183	
7/26/13	2.180	77.00	83.60	29.37	5.50	11.25	0.0305	0.234	
Average								0.224	After Fan Separator
7/28/13	6.114	90.00	86.25	29.39	10.00	18.0	0.0240	0.066	
7/29/13	9.049	87.10	89.70	29.39	12.00	29.0	0.0305	0.058	
7/29/13	4.370	93.00	89.00	29.38	12.00	11.5	0.0215	0.087	
7/30/13	10.958	88.40	89.10	29.34	10.00	39.0	0.0425	0.067	
Average								0.0695	

practically corresponding to saturation at the off-coming gas temperatures, and at no time have we encountered mechanically entrained water in amounts to be detrimental.

RAW VS. CLEAN GAS

The Use of Raw Gas

The use of raw gas in stoves is conducive to low efficiencies. This is due primarily to the heavy coating of dust on the brickwork fusing in masses, thus preventing the absorption of heat, destroying the brickwork, and requiring constant attention to the removal of this material. During the periods of scraping out the combustion chamber the stoves are burning gas very inefficiently, and at such times as the stove is off for complete cleaning the furnace is deprived of one-fourth of its blast-heating capacity. The intermittent pressures also make it impossible to obtain a constant flue-gas analysis and allow either excess air or gas in the stack.

The use of raw gas in the boilers shows a similar low efficiency, but not so marked. It is customary to blow off the tubes every 24 hr., and while the blowing is going on much excess air enters the combustion chamber. From the time the tubes are cleaned they start collecting dirt again and the steaming power suffers accordingly. Intermittent pressure also affects the boiler with regard to flue-gas analyses similarly to the stoves.

The Use of Clean Gas

By the use of clean gas in the stoves we completely do away with cleaning stoves, have a constant pressure in the gas supply, and use an orifice plate over the air-inlet door, which gives us an extremely constant and economical flue-gas. The saving on clean gas over dirty gas on stoves with regard to iron was shown in detail by the author¹ in a discussion of Mr. Forbes's paper on gas cleaning,² read before the Institute in October, 1913, and amounts to 15.91 c. per ton of iron.

The use of clean gas in the boilers gives the same benefits in operation

The Use of Refined Gas

The thoroughly cleaned gas is used only for engine purposes. A yearly average of efficiencies of our power engines, with the analysis of the gas is given in Tables XI and XII. The engines are 42 by 60 twin tandem Snow engines, driving a 2,000-kw. generator at $83\frac{1}{2}$ rev. The quantity of gas is approximately 5,000 cu. ft. per minute, each engine of this size taking about the same amount as a stove.

¹ *Bulletin No. 84*, December, 1913, p. 2954.

² *Bulletin No. 82*, October, 1913, p. 2477.

TABLE X.—*Extracts from Boiler Tests*

Observations		Fuel—Un- washed Blast- Furnace Gas	Fuel— Washed Blast- Furnace Gas
1	Number of tests.....	16	15
2	Average duration of tests.....	Hrs..Min. 6:53	7:12
3	Water-heating surface.....	# Sq. Ft. 2,790	2,790
4	Steam pressure, gauge.....	Sq. In. 129.1	121.8
5	Draft.....	Ins. W. 0.6956	0.8226
6	Temperature of stack gases.....	°F. 649.3	652.8
7	Temperature of fuel gas.....	°F. 376.3	101.7
8	Calorific value of gas per cu. ft. at 32° F. and 30 in. Hg	B.t.u. 113.07	106.86
9	Calorific value of gas per cu. ft. at 32° F. and 30 in. Hg	B.t.u. 104.34	96.44
10	Quality of steam, corrected.....	Per cent. 97.01	97.32
11	Equivalent evaporation per 1,000 cu. ft. gas.....	Lb. 59.14	61.93
12	Equivalent evaporation per sq. ft. heating surface...	Lb. 3.890	3.254
13	Equivalent evaporation per 1,000 B.t.u.....	Lb. 0.5297	0.6190
14	Fuel gas at 32° F. and 30 in. Hg. per developed H.P. ...	Cu. Ft. 613.7	572.3
15	Builders' rating.....	B.H.P. 250	250
16	Horse-power developed.....	B.H.P. 313.83	264.16
17	Per cent. of rated capacity developed.....	Per cent. 125.53	105.60
18	Heat lost due to incomplete combustion.....	Per cent. 7.28	7.20
19	Heat lost due in waste gases.....	Per cent. 27.92	27.44
20	Heat lost due radiation and unaccounted for.....	Per cent. 14.32	6.31
21	Efficiency (heat absorbed by water).....	Per cent. 50.48	59.05
tem No. 8 calculated from analysis; Item No. 9 as per calorimeter.			

Notation.—The above tests were run on boilers of the same make, type, and rating. It will be observed that the boilers working on clean gas developed but very little more than the builder's rating, although working at a higher efficiency than those on unwashed gas. This is due to an insufficient gas supply, which is directly traceable to undersized mains and connections, the mains having been so designed as to take care of only 1,500 boiler horse power of boilers on clean gas. In this case, however, the operation of approximately 3,750 boiler horse power was attempted, resulting in a gas shortage for each of the respective units.

The above tests were also run under normal conditions and are therefore representative of working performance. It has been our experience recently that steam can be raised very quickly on washed gas, having gotten such boilers on the line at 140 to 150 lb. gauge from feed water at 70° F., in 30 to 45 min. Due to very uniform gas pressures, as mentioned elsewhere, the boilers on washed gas steam very regularly.

There are four blowers and two power engines, the gas cylinders being the same in both cases. Gas containing 0.009 grain of dust per cubic foot is thus shown to be sufficiently clean for operating purposes. The data given relate to electric power engines. No. 1 gas power engine started in January, 1910, operated to Dec. 1, 1913, a total of 28,988 hr. No. 2 gas power engine started in July, 1910, operated to Dec. 1, 1913, a total of 27,095 hr. Total hours operated equaled 56,083.

Average load per engine, 1557.5 kw.

Average cu. ft. dry gas at 62° F. and 30 in. Hg consumed per kw-hr., 175.32.

Average thermal efficiency at switchboard, 19.12 per cent.

Average thermal efficiency per indicated horse power, 27.14 per cent.

Average dust in gas = 0.00925 grain per cu. ft. of dry gas at 62° F. and 30 in. Hg.

Average dust in air = 0.0016 grain per cu. ft. of dry air at 62° F. and 30 in. Hg.

Engines operate on constant mixture at all loads. Mixture is approximately one volume of gas to one volume of air.

Dust carried into cylinders per cu. ft. gas consumed = $0.00925 + 0.0016 = 0.01041$ grain.

Gas consumed per engine per hour = $175.32 \times 1557.5 = 273,061$ cu. ft. at 62° F. and 30 in. Hg.

Gas consumed by both engines since starting = $56083 \times 273061 = 15,314,080,063$ cu. ft. at 62° F. and 30 in. Hg.

Dust carried into cylinders since engines were started = $15,314,080,063 \times 0.01041 = 159,419,573$ grains = 22,775 lb. = 11.387 tons.

TABLE XI.—Performance of Gas Power Engines, Year 1912

	Legend		No. 1 Engine	No. 2 Engine
1	Total time operating.....	Hi.....	8,313.90	8,542.38
2	Total power output.....	Kw-hr....	14,020,290	14,158,340
3	Average load.....	Kw.....	1,704	1,627
4	Load factor.....	Per cent..	85.20	81.35
5	Total gas consumed.....	Cu. ft....	2,461,872,000	2,601,288,000
6	Gas consumed per kw-hr.....	Cu. ft....	175.59	183.73
7	Gas consumed per i.h.p-hr.....	Cu. ft....	120.51	126.10
8	Gas consumed per b.h.p-hr.....	Cu. ft....	91.59	95.84
9	Calorific value of gas, average for year.....	B.t.u.....	95.80	95.80
10	Heat consumed per kw-hr.....	B.t.u.....	16,822	17,601
11	Heat consumed per b.h.p-hr.....	B.t.u.....	11,545	12,080
12	Heat consumed per i.h.p-hr.....	B.t.u.....	8,774	9,181
13	Thermal efficiency at switchboard.....	Per cent..	20.28	19.38
14	Thermal efficiency at b.h.p.....	Per cent..	22.04	21.07
15	Thermal efficiency at i.h.p.....	Per cent..	29.01	27.72
16	Gas consumed per minute, average for year.....	Cu. ft....	4,935	5,075
17	Gas consumed per hour, average for year.....	Cu. ft....	296,100	304,500
18	Time operating.....	Per cent..	94.40	97.50
19	Time delay due to engine.....	Per cent..	2.79	1.52
20	Time delay due to engine, not required.....	Per cent..	0.91	0.64
21	Time delay due to other causes.....	Per cent..	1.80	0.34

Gas volumes and calorific values all at 62° F. and 30 in. Hg.

Within a few hours after starting, all exposed parts inside of the cylinders except the wearing surface become coated with a light brown deposit. It has been our experience that if oil can be kept away from these surfaces this deposit is only very slightly increased by further operation of the engines. In the combustion chambers, where there is never any oil present, the deposit is so even that we have operated cylinders for nearly four years without cleaning these surfaces, and consider that the presence of such an even coating is an advantage, having an insulating effect, and protecting the metal from the direct action of the hot gases. However,

TABLE XII.—Résumé of Dust Tests on Blast-Furnace Gas Delivered to Gas Engines

Test No.	Cu. ft. Gas Metered	Temp. of Cooling Water ° F.	Temp. of Gas at Meter ° F.	Temp. of Gas at Main ° F.	Press. of Gas at Meter Ina. Water	Press. of Gas in Main Ina. Water	Barometer Ina. Hg.	Weight of Dust as Sampled Gram	Duration of Test Hrs. Min.	Dry Gas Factor at Meter Temperature	Grains Dust per cu. ft. Dry Gas at 62° F. and 30 in. Hg.
67	8,500	34.0	44.0	46.0	13.5	13.5	28.24	0.0025	2	0.991	0.00435
68	110,000	38.0	48.5	44.5	2.0	14.0	28.20	0.0100	15	0.989	0.00189
69	139,173	36.0	60.3	46.6	2.0	14.0	28.48	0.0430	3	0.988	0.00477
72	146,073	36.5	39.5	44.6	2.0	14.5	28.23	0.0230	4	0.982	0.00238
76	142,312	34.0	45.0	55.2	2.0	15.0	28.38	0.0605	3	0.990	0.00543
78	86,108	34.0	41.8	42.1	2.0	14.0	28.54	0.0265	2	0.992	0.00449
79	83,561	33.0	44.0	40.2		15.0	28.43	0.0380	2	0.991	0.00659
80	246,436	34.0	46.0	41.4		13.5	28.31	0.1380	5	0.990	0.00865
81	116,561	34.0	50.2	44.5		12.0	28.27	0.0405	3	0.988	0.00544
83	111,413	33.0	44.6	41.0		14.0	28.35	0.0280	2	0.988	0.00406
85	146,597	33.5	44.6	41.0		15.5	28.02	0.0435	2	0.990	0.00578
88	139,421	36.0	56.6	46.0		14.5	28.07	0.0318	3	0.984	0.00343
89	139,421	36.0	38.0	45.5		15.0	28.21	0.0144	3	0.992	0.00157
91	132,570	44.0	57.1	53.7		14.5	28.29	0.1185	4	0.984	0.01079
92	105,821	45.0	53.3	50.0		11.5	28.93	0.0465	3	0.987	0.00558
93	83,698	44.0	57.2	51.6		13.0	28.15	0.0318	3	0.984	0.00485
94	146,083	40.0	53.2	52.0		13.5	28.29	0.0070	2	0.984	0.00133
95	141,358	41.5	56.5	49.2		14.5	28.27	0.0795	3	0.987	0.00856
96	144,349	46.0	58.4	52.7		14.0	28.35	0.0410	3	0.978	0.00471
98	161,684	48.0	74.0	60.0		14.0	28.41	0.0630	2	0.984	0.00573
99	160,857	45.0	72.0	66.0		13.0	28.36	0.0153	3	0.971	0.00311
101	159,356	55.0	78.0	62.0		15.5	28.30	0.0675	3	0.972	0.00690
102	134,354	57.0	78.9	66.0		16.0	28.23	0.0865	3	0.977	0.01286
103	145,355	54.0	76.9	69.1		13.0	28.61	0.0510	3	0.966	0.00648
106	159,865	62.0	78.3	69.1		12.0	28.29	0.1233	3	0.988	0.01183
107	178,731	59.5	83.1	67.4		10.5	28.45	0.0975	3	0.987	0.01386
108	163,454	50.0	63.4	65.2		11.0	28.45	0.1545	4	0.980	0.00991
109	179,450	49.0	69.6	68.5		12.0	28.43	0.0810	3	0.980	0.01422
110	173,454	50.0	68.6	68.5		13.0	28.02	0.0372	3	0.975	0.00783
111	145,352	60.0	72.3	67.0		12.0	28.35	0.0890	3	0.975	0.00355
112	145,352	66.0	72.3	67.0		12.0	28.45	0.0745	3	0.983	0.00804
113	163,455	66.0	73.2	71.2		12.0	28.02	0.1785	3	0.973	0.00848
114	163,450	84.0	72.1	71.2		11.0	28.35	0.1500	3	0.970	0.00921
115	124,380	75.5	86.5	82.0		10.0	28.32	0.1075	3	0.973	0.01469
116	139,890	77.5	87.0	85.0		12.0	28.30	0.0960	2	0.961	0.01490
117	189,017	78.0	87.9	86.2		13.0	28.30	0.1765	3	0.966	0.00737
118	145,312	80.5	88.9	87.2		10.5	28.15	0.1665	3	0.955	0.01088
119	173,090	81.5	88.5	88.5		10.0	28.15	0.1635	3	0.955	0.02196
120	198,462	74.0	86.6	79.0		10.0	28.37	0.3597	2	0.967	0.02453
126	263,834	71.0	78.0	78.6		11.0	28.37	0.2767	4	0.980	0.01472
127	313,981	71.0	78.0	77.6		10.0	28.40	0.1761	3	0.974	0.02602
128	238,930	73.0	73.0	77.6		9.0	28.50	0.1301	4	0.968	0.00955
129	273,780	68.5	74.1	75.8		12.0	28.10	0.3143	4	0.980	0.00812
130	220,231	41.0	43.5	47.8		14.5	28.30	0.2305	0	0.972	0.01407
131	250,980	46.9	50.9	43.8		14.5	28.74	0.2411	0	0.960	0.01613
132	236,041	35.7	44.6	44.6	2.76	14.0	28.94	0.2034	0	0.985	0.01571
133									4	0.990	0.01355
Average											0.00925

where an excess of oil is present, as on the inside of cylinder head directly below the piston rod (the oil being carried in by the rod), the dirt accumulates more rapidly and often very unevenly. If an engine is shut down and the head allowed to cool, then started and brought up to working temperature again, the contraction and expansion of the head breaks the accumulated oil and dirt loose in the form of scale or blisters, the edges of which are easily heated and cause premature explosions. Usually this scale will partly break off, due to these premature explosions, and trouble at this point usually ceases until the next time the engine is stopped and started. When this trouble occurs too frequently, the engine is shut down, the cylinder examined through the opening for air-starting check valve, and the deposit scraped off if possible. If too hard or considered too extensive an accumulation the head is pulled out and the surfaces thoroughly cleaned. Following this procedure, we have some cylinder heads on our blowing engines that have not been out for over 18 months. Previous to August, 1913, when new piston rods were installed in the two power engines, practically every cylinder on the two engines had been closed up for more than a year, although examined through the starting-valve opening at shorter intervals.

We have an advantage over engines with exhaust valves directly in the bottom of the cylinder in that the oil and dirt that accumulate here do not interfere in any way with the operation of the valve, but can be blown out through the drain valve, with which each end of each cylinder is equipped. These valves are held open for two full cycles every 3 hr.

Our experience indicates that with our type of engine we can operate from 12 to 18 months without removing heads for cleaning, the smaller particles of dirt being removed at lesser intervals, and also that the cleaning is necessary principally from the fact that oil is present. The portions free from oil are covered so evenly that the presence of the deposit is an advantage.

Comparison of Systems

For the purpose of comparing general systems of gas cleaning, it might be interesting to note, first, a proposed installation of Duquesne tower washers as suggested for two furnaces, washing only for stoves. This installation depends on the simple scrubbing of gas without fans and will only reduce the top pressure about 2 in. of water, delivering gas to stoves at 0.5 grain of water above saturation point and cleaned to 0.2 grain of dust per cu. ft. The water used is standpipe water at its cost, delivered to point of consumption and boosted 15 ft. by a centrifugal pump. Second, an installation of two 40-ft. scrubbers in series, taking advantage of the bosh water head if the plant construction justifies the same, or with just sufficient "boosting" to obtain a 30-ft. head. These

installations will compare in cost per 100,000 cu. ft. of gas washed and per ton of iron as follows:

	Cost per 100,000 cu. ft.	Cost per Ton Iron
First case.....	0.04941	0.02386
Second case.....	0.00709	0.0034

It is evident that where sufficient head is available to use bosh water direct, two of these towers in series will make the cheapest installation possible, both as to first cost and operation cost, as the cost of pumping water is the major expense. The cost of operation between the towers and mechanical washers depends on the relation of water cost to power cost and machinery repairs.

It is also very desirable to have the temperature of the gas as low as possible to remove moisture, and this means either a series of machines or a counter current effect such as is obtained in a tower construction. It is also evident that in a machine where the water is in continuous contact with the gas, the gas will leave at the temperature of the outgoing water. The plant shown in Fig. 14 will be sufficient for two furnaces and will allow ample spare capacity, as each scrubber will be capable of handling 40,000 cu. ft. of gas per minute when pushed. This means that one scrubber can be used as a spare if so desired. An estimate of the cost of construction for the plant shown in the first proposition is as follows:

Estimated Cost of Duquesne Gas Scrubber for a Two-Furnace Plant

Excavating and foundations.....	\$1,200
Scrubbers, incl. water valves and flues, main gas flues not included	13,450
Brick lining for flues, main gas flues not included.....	500
Supports for water valves.....	750
Pipe line, 4 in. in diameter.....	400
Rotary water valves.....	2,750
Water strainers.....	500
Centrifugal pump and motor, capacity 5,000,000 gal. per 24 hr.....	1,550
	\$21,100
General expense and contingencies, 5 per cent.....	1,055
Total.....	\$22,155

COST OF CLEANING GAS

The cost of cleaning gas for the year 1912 is given in Tables XIII to XV, and may be useful in showing an actual condition and what can be expected from an equipment working under the local conditions as outlined.

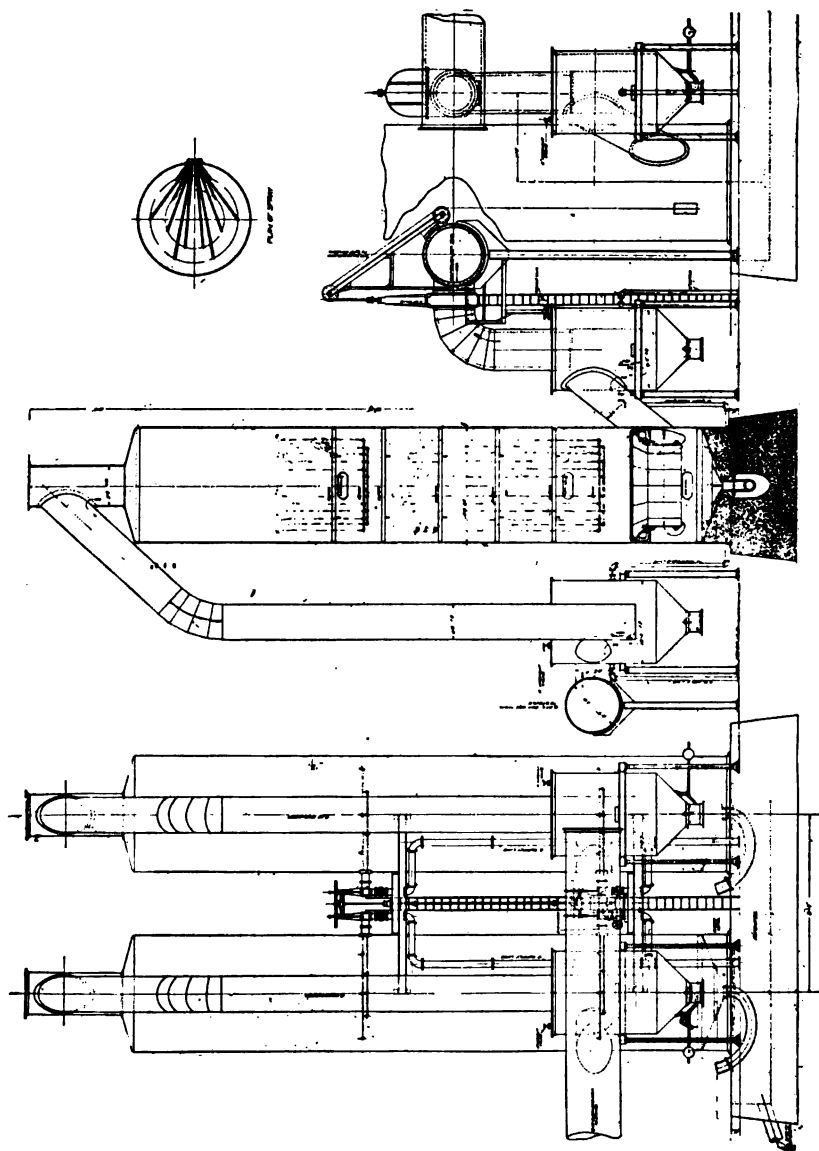


FIG. 14.—PROPOSED INSTALLATION OF SCRUBBERS FOR STOVES AND BOILERS FOR TWO FURNACES. CAPACITY OF EACH SCRUBBER, 40,000 CU. FT. PER MINUTE.

TABLE XIII.—*Cost of Cleaning Blast-Furnace Gas—Primary, Year 1912*
General Statement

		Year's Average
1	Gas cleaned:	
	No. 3, 4, 5, and 6 furnace stoves.....	3,117,660,000
	Gas engines.....	804,322,000
	Total.....	3,921,982,000
2	Water consumed (gallons):	
	Scrubbers.....	134,209,000
	Fans.....	22,157,000
	Water valves.....	13,869,000
	Total.....	170,235,000
3	Electric current consumed:	
	Lighting, kw-hr.....	2,203
	Pumps.....	51,650
	Fans.....	225,515
	Total.....	279,368
4	Producing labor cost.....	\$373.42
5	Labor in repairs and maintenance.....	248.65
6	Materials in repairs and maintenance.....	190.65
7	Cost of electric current consumed.....	1,053.44
8	Cost of water consumed.....	595.88
9	Total cost.....	\$2,462.04

Primary Cleaning—Economic Performance

		Year's Average
	Cu. ft. gas cleaned:	
1	Per gallon of water, scrubbers.....	29.65
2	Per gallon of water, fans.....	180.65
3	Per cu. ft. of water, scrubbers.....	222.26
4	Per cu. ft. of water, fans.....	1,354.11
5	Per gallon of water, primary.....	23.00
6	Per cu. ft. of water, primary.....	172.20
7	Per kw-hr., primary.....	14,615.70
	Cost per 100,000 cu. ft.	
8	Producing labor.....	\$0.01052
9	Labor in repairs and maintenance.....	0.00630
10	Materials in repairs and maintenance.....	0.00481
11	Electric current.....	0.02660
12	Water.....	0.01506
13	Total cost per 100,000 cu. ft., primary.....	0.06329
	Percentage of total cost:	
14	Producing labor.....	16.62
15	Labor in repairs and maintenance.....	9.95
16	Materials in repairs and maintenance.....	7.60
17	Electric current.....	42.03
18	Water.....	23.80
19	Cost per 150,000 cu. ft. gas, primary.....	\$0.09496
20	Cost per 1,000,000 B.t.u., primary.....	0.00657
21	Cost per 1,000,000 B.t.u., natural gas.....	0.04155
22	Cost per 1,000,000 B.t.u., coal.....	0.04464

TABLE XIV.—*Cost of Cleaning Blast-Furnace Gas—Final Cleaning, Year 1912. General Statement*

	Year's Average
1 Gas cleaned, engines.....	804,322,000
2 Water consumed, theisens (gallons).....	15,043,000
3 Electric current consumed:	
Lighting, kw-hr	513
Power.....	137,653
Total.....	138,166
4 Producing labor cost.....	\$157.50
5 Labor in repairs and maintenance.....	66.06
6 Materials in repairs and maintenance.....	140.80
7 Cost of electric current consumed.....	538.54
8 Cost of water consumed.....	52.65
9 Total cost.....	\$955.55

Final Cleaning—Economic Performance

	Year's Average
Cu. ft. gas cleaned:	
1 Per gallon water consumed.....	53.48
2 Per cu. ft. water consumed.....	400.77
3 Per kw-hr. water consumed.....	5,822.10
Cost per 100,000 cu. ft.:	
4 Producing labor.....	\$0.01965
5 Labor in repairs and maintenance.....	0.00822
6 Materials in repairs and maintenance.....	0.01701
7 Electric current.....	0.06710
8 Water.....	0.00658
9 Total cost per 100,000 cu. ft., final cleaning.....	0.11856
Percentage of total cost:	
10 Producing labor.....	16.57
11 Labor in repairs and maintenance.....	6.94
12 Materials in repairs and maintenance.....	14.35
13 Electric current.....	56.59
14 Water.....	5.55
Cost per 1,000,000 B.t.u.:	
15 Final cleaning.....	\$0.01237
16 Natural gas.....	0.04155
17 Coal.....	0.04464
18 Cost per 150,000 cu. ft.....	0.17784

TABLE XV.—*Cost of Cleaning Blast-Furnace Gas, 1912. General Exhibit*

A.	Cost per ton of iron—Gas cleaned and delivered to stoves	Average for Year
1	Total gas delivered to stoves for month, cu. ft.....	32,406.93
2	Cost per 100,000 cu. ft. at stoves, primary.....	\$0.06329
3	Total cost gas cleaned and delivered to stoves.....	\$2,051.03
4	Tonnage, furnace working on clean gas in stoves.....	63,061
5	Cost per ton of iron—gas cleaned and delivered to stoves.....	\$0.03252

B.	Cost per ton of iron—Gas cleaned and delivered to blowing engines	Year's Aver.
1	Total gas delivered to engines for month, cu. ft.....	3,828.01
2	Cost per 100,000 cu. ft., gas delivered to engines.....	\$0.18185
3	Total cost of gas cleaned and delivered to engines.....	\$696.124
4	Tonnage credited to gas-blowing engines.....	31,524
5	Cost per ton of iron—gas cleaned and delivered to engines.....	\$0.022082

C.	Cost per 1,000 kw-hr., gas cleaned and delivered to gas power engines	Year's Aver.
1	Total gas delivered to gas power engines for month, cu. ft.....	4,215.21
2	Total cost 100,000 cu. ft.—gas delivered to engines.....	\$0.18185
3	Total cost gas cleaned and delivered to engines in month.....	\$766.536
4	Total power output, gas power engines—kw-hr.....	2,367,386
5	Cost per 1,000 kw-hr.—gas cleaned and delivered to engines.....	\$0.32379

Note: Items No. 1—100,000's cu. ft.

Cost of Fans and Scrubbers

The fan operation is expensive, due to the high power consumption. This can be noted in the yearly cost for 1912. If we had ample mains and good draft, as can be obtained in a simple plant for only scrubbing for stoves and boilers, a fan would be unnecessary and costs would be lessened accordingly. If bosh water were used at no charge, the cost would be lowered further. The cost of water and power in 1912 was 66 per cent. of the entire cost, and with a simple tower plant, without fans and using bosh water, the cost would be reduced in that proportion. The operating force due to the secondary cleaning, handling of motors, etc., gives a proportion chargeable to primary cleaning which would be unnecessary in a simple primary tower plant for stoves and boilers, as the attention so required is very small. From the costs shown and the deductions which may be drawn, it is very reasonable to expect to wash gas by efficient towers sufficiently clean for stoves and boilers for 1.5 c. per ton of iron.

Cost of Theisens

The Theisen cost is high, due also to power and service water. The power cost cannot be changed very much with this type of machine. The water expense would be cut down if bosh water were used. The repair cost is high, as is natural to expect in such a rapidly revolving piece of apparatus.

CONCLUSION

There are numerous ways and means of cleaning gas, as presented in Mr. Forbes's paper before the Institute in October, 1913. The dry cleaning, as at present operated, is cumbersome, occupies large space, is expensive to install and is open to numerous operating difficulties. In all wet systems the handling of the fine sludge becomes the same problem, whether it is removed by mechanical devices or by towers, and from the standpoint of an operator, the simplest device is generally the most inexpensive and requires the least attention. From the experience derived from our installation it seems that a simple tower with proper means of preventing channeling is the easiest solution for primary gas, as 0.20 to 0.25 is generally conceded to be sufficiently clean for stove and boiler purposes.

The author wishes to gratefully acknowledge the aid of S. E. Eldridge, in charge of gas cleaning at the Duquesne Works, under whose direction many of these data were collected and tests prepared. Other data, tests, and deductions, as well as some of the details only outlined here, were presented to the Institute by the author in a discussion of Mr. Forbes's paper on gas cleaning in October, 1913, and might be interesting in this connection. It has been the desire in the present paper to present actual and experimental data, as well as general and interesting information on the subject of blast-furnace gas cleaning. The data are confined to one plant, but as that plant has operated for five years, it is representative of general practice, and the information presented may be useful as an aid to others in furthering the knowledge on this present live subject.

DISCUSSION OF THIS PAPER IS INVITED. It should be preferably presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

Is It Feasible to Make Common Carriers of Natural Gas Transmission Lines?

BY SAMUEL S. WYER, COLUMBUS, OHIO

(New York Meeting, February, 1914)

OVER 8,000,000 people in the United States depend on natural gas for their cooking, heating and lighting service. This service has been made possible only by the investment of large amounts of capital in transportation lines connecting the gas wells to the various communities, thus combining a *hazardous* mining operation with a public utility service. The effect that making these lines "common carriers" will have on the present and future service to these 8,000,000 or more people is of far reaching ethical and economic importance, and deserves careful consideration. As an introduction, it must be conceded that:

1. Natural gas, although a mineral and obtained by a mining operation—with more uncontrollable and uncertain features to cope with than exist in the mining of coal or other minerals—when served to the public becomes a public utility service.

2. The governmental power to regulate all public utilities in the interests of the public cannot be successfully disputed.

3. Mere suggestion of inconvenience to the utility is wholly irrelevant, as it cannot be considered or allowed to influence in making a defense against governmental regulation. Unfortunately it is not generally appreciated that:

1. Legislative enactments are not necessarily laws, and cannot abrogate economic laws or alter engineering facts.

2. There is a clear distinction between "common carrier" and "public utility," although the terms are frequently confused. "A common carrier is one who undertakes for hire to transport persons or goods, or both, from place to place for all persons indifferently, the term implying indifference as to whom shall be served, and an equal readiness to serve all who apply, in the order of their application."

On the other hand, property becomes a "public utility" only "when for a particular purpose the public have a right to resort to the premises and make use of them."

It is, therefore, important to note that not all public utilities can be regarded as common carriers. The U. S. Supreme Court has already

made this distinction in the case of telegraph lines, noting the peculiar service characteristics that prevail there. A proper analysis of natural

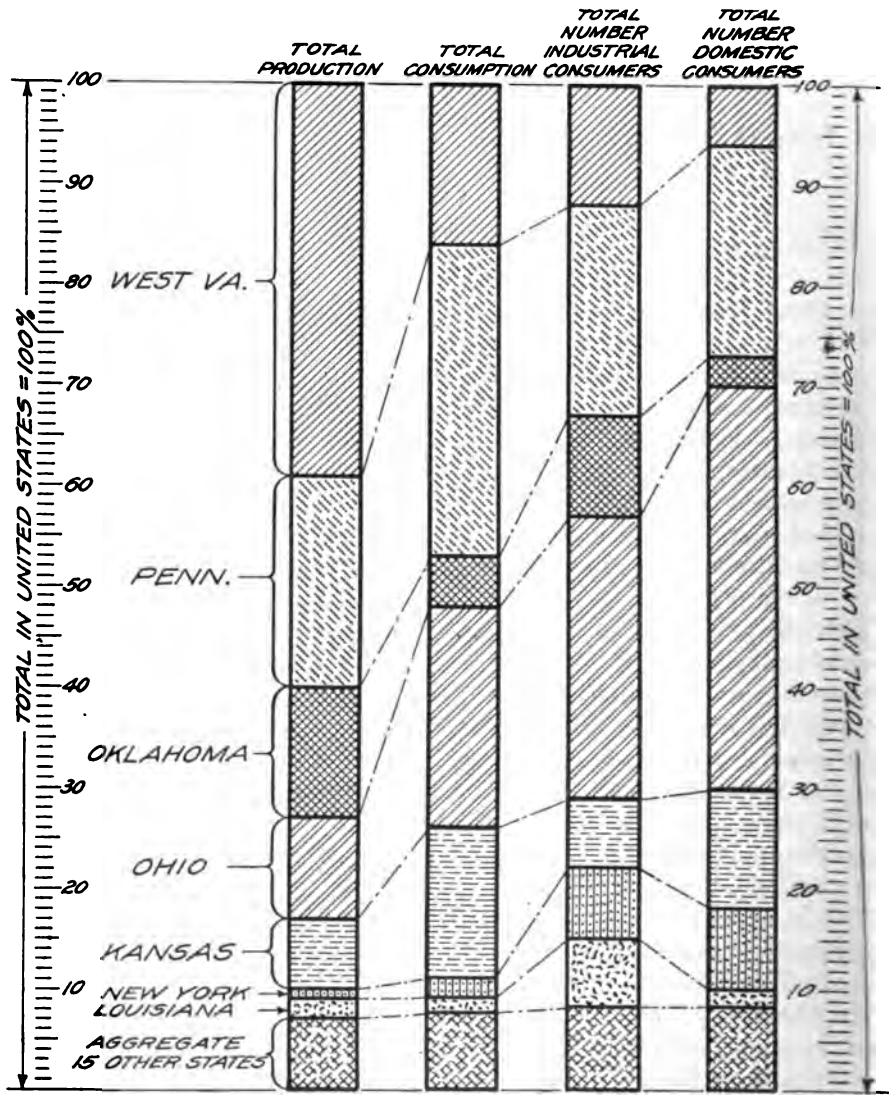


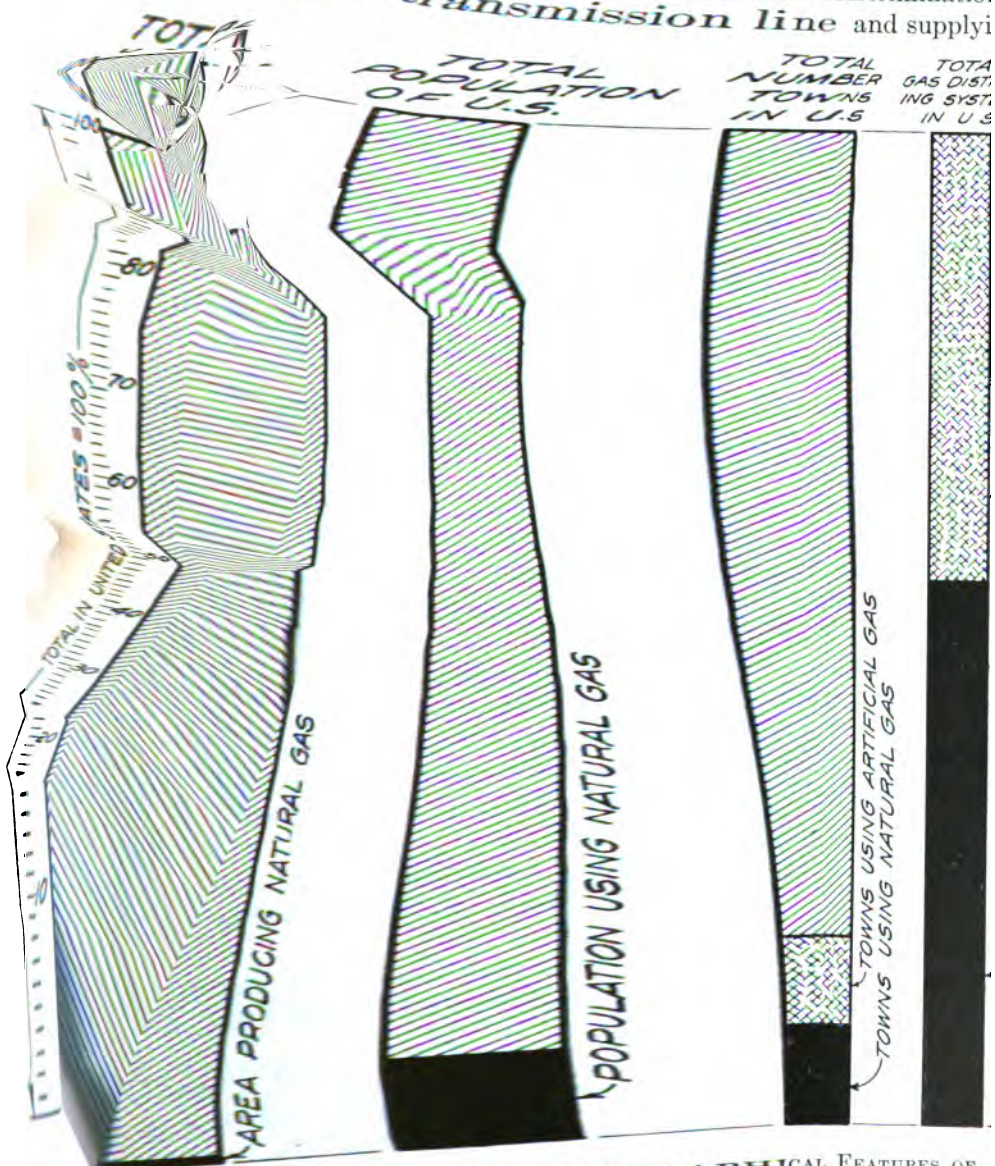
FIG. 1.—INTERSTATE RELATION OF PRODUCTION AND CONSUMPTION OF NATURAL GAS IN THE UNITED STATES.

gas operating conditions will show that a similar exception ought to be made in the case of natural gas service.

3. Natural gas companies are not chartered to act and do not offer to act as common carriers.

COMMON CARRIERS AND NATURAL GAS

4. The legal definition of "common carrier" is a business, for the reason that a common carrier is non-discrimination in gas transmission line and supply



2.—RELATIVE GEOGRAPHICAL FEATURES OF UNITED STATES.

nature of things, must give its own nature of public policy and the contractual nature of the gas company. consumers and the gas company.

5. The consumer's interest and rights extend clear back to and depend on the gas wells and reserve acreage that the producing company maintains to insure an adequate present and continuous future service.

6. Large interstate natural gas business—which cannot be burdened or interfered with by any State—has been developed as shown in Fig. 1.¹

7. The relative geographical features of the natural gas business in the United States are startling, as shown by Fig. 2.

8. The hazard in the natural gas business is greater than that in most mining operations or in any other utility service. It is more difficult to determine the outline and volume of a natural gas formation than it is to make a similar determination for coal or other minerals. In fact, it is impossible from surface indications to determine whether gas can be found, and even a large well is no sure indication of a paying field, as it frequently happens that prolific wells are surrounded by dry holes or unprofitable territory, and the wells are short lived.

9. Gas companies discharging their legal duty to their domestic consumers cannot depend upon the initiative of the occasional producer for a supply of gas, but must depend upon their own initiative in order to maintain proper field operating conditions and an adequate reserve acreage for future development to insure a good service to their patrons. Experience has many times shown that satisfactory continuous service to the consumer can be rendered only when the *production, transportation, and distributing* features are properly co-ordinated. To subordinate the transportation side of the business to either the producer's or the large industrial consumer's interest is indefensible.

10. Even though natural gas is a mineral it requires constant attention from the time it is reduced to possession at the bottom of the well, and embodies an unbroken chain of service features until it is burned at the consumer's fixtures.

11. Less than 1 per cent. of the total number of natural gas consumers of the United States are industrial users, yet these users take over 65 per cent. of the total gas used, as shown in Fig. 3.

12. The 1,621,557 domestic consumers in the United States have at least \$150,000,000 invested in service connections, house piping, and gas-using appliances.

13. To make common carriers of natural gas lines will destroy the policy of conservation and greatly increase the waste of natural gas in industrial work, thus tending to soon exhaust the available supply and leave the householders with large investments of appliances and pipes which will be useless, owing to the permanent failure of the gas.

14. Gas may be produced from a small territory and a few wells at a

¹ Figs. 1, 2, and 3 are based on the latest U. S. Geological Survey and U. S. Census statistics.

much lower cost than is possible where the producer so develops his property as to hold a large acreage in reserve in order that a continuous gas supply may be guaranteed for a period of years.

The important ethical features of this common carrier problem are:

1. The domestic consumer's interests are superior to the industrial consumer's interests. This is because it is an easy matter to install

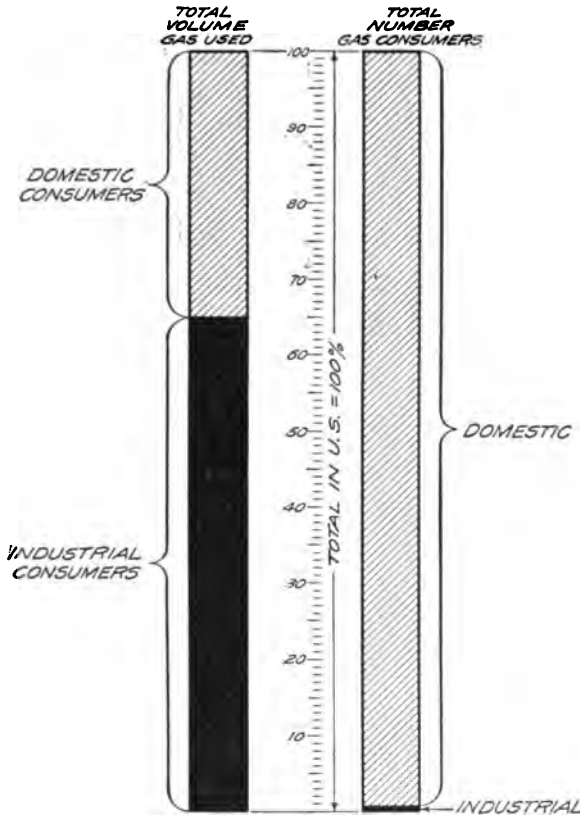


FIG. 3.—RELATION OF DOMESTIC AND INDUSTRIAL NATURAL GAS CONSUMERS.

industrial appliances for burning solid fuel satisfactorily, but practically impossible to do this for the domestic consumer. Natural gas, being superior to any man-made product, is ideally adapted for domestic service, and ought to be conserved for this, rather than used in ordinary industrial operations.

2. Natural gas service to the domestic consumer is one which in no sense bears any relation or sustains any connection to the duties of a common carrier.

3. Natural gas, although worth 50 per cent. more on a heating-value basis than artificial gas, when used for industrial work must be sold at

an absurdly low price to compete with producer gas for industrial service. For instance, it may be shown that in a producer gas plant gasifying 3 tons of (12,000 B.t.u.) coal per hour, allowing 6 per cent. for interest on investment, 7 per cent. for sinking fund, and 3 per cent. for repairs, plus operating labor, 24-hour operation, 300 days per annum, with coal at \$2 per ton, natural gas must be sold at 15 c. per "M" cubic feet; and with coal at \$2.50 per ton, natural gas must be sold at 18 c. per "M" cubic feet in order to compete on a price basis. In view of the fact that the average price of artificial gas throughout the United States is over 85 c. per "M," the economic waste in permitting the use of natural gas for industrial service is self-evident.

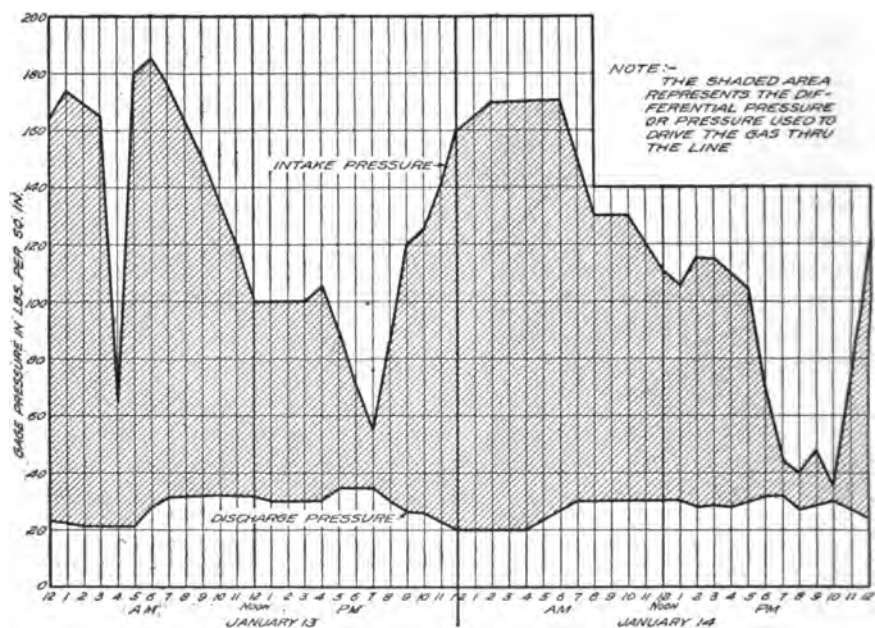


FIG. 4.—TYPICAL INTAKE AND DISCHARGE PRESSURES OF A NATURAL GAS TRANSMISSION LINE NECESSARY TO MEET THE VARYING DEMANDS OF THE DOMESTIC CONSUMERS.

4. There can be no conservation for future generations until there is an appreciation of present and future worth of natural gas service.

The leading legal features of this common carrier problem are:

1. The retail price of the gas service to the consumer is fixed by ordinances for relatively long periods.
2. That there can be no legal deprivation of property, either directly by confiscation, or indirectly by onerous burdens.
3. The contractual relations existing with present gas consumers.

The present agitation to make natural gas lines common carriers comes from three sources:

1. Small companies with no reserve acreage to take care of their normal growth or depleted condition of old and operated natural gas supplies. Many small companies are operating properties with an acreage of not over 50 acres per well. The average acreage per gas well in the United States is about 400 acres per well, and the companies that can be depended upon to render continuous service in most cases hold from 700 to 800 acres per well.

2. Manufacturers who own gas property in gas field, but who do not wish to incur the expense of carrying such gas to a point where factories are located. In several cases, after a field has been developed manufacturers have stepped in and bought a small amount of production without having any means of taking such production to the point where their factories are located. These manufacturers, therefore, expect that adjacent pipe lines serving large numbers of domestic consumers should be placed at their service in handling their industrial gas.

3. Small producers in gas field with no reserve acreage, and unable to find a local market for their gas, or unable to make the investment necessary to carry such gas to the market, are clamoring for the common carrier power in order to compel the adjacent pipe lines to act not only as common carriers in carrying their gas to a market, but also to compel such pipe-line owners to buy such gas by virtue of the economic conditions surrounding the transmission and marketing of natural gas.

It will be observed that all of these are distinctly local and selfish, and ignore the conservation principles entirely.

The engineering and economic operating conditions that must be considered and coped with in making common carriers of natural gas lines are:

1. Gas can go in *one* direction only. This makes a common carrier service for a pipe line entirely different from a common carrier service for a railroad, where transmission may be carried on in two directions.

2. Capacity of transmission lines is rigidly fixed, and will not stand any overload. This has a marked effect in taking care of peak loads, in contradistinction to railroads, which may run extra trains to carry extra traffic.

3. Natural gas service is instantaneous. There can be no delays in rendering service, as is possible (and universally practiced) in other transmission agencies, such as railroads and traction lines. For instance, a railroad can very easily start one hour late in case of congested traffic, but a natural gas service that delivers gas for cooking breakfast one hour after the consumer needed it, would not only be valueless to the consumer, but would not be tolerated in any community. This instantaneous

feature differentiates natural gas service from all transportation utilities and common carrier agencies.

It may be argued that as between a domestic consumer and an industrial consumer, the domestic consumer could be given preference in cold weather, but sudden changes in temperature would not permit the transfer from the manufacturing to the domestic consumer in time for the protection of the latter.

4. While it has been urged that gas transmission companies be obliged to build additional lines so as to care for all common carrier gas offered them, it is patent that should this be done the company would be burdened with a large indebtedness, with no assurance of continuous common carrier business, because of the uncertain life of the wells from which the common carrier gas would be offered. The well-known common law doctrine that "a utility cannot be forced to make an expenditure unless there exists a reasonable expectation that the income from such expenditure will justify the investment made" refutes this contention.

5. Since the size and length of the transmission line remain stationary, the only variability possible in taking care of a large fluctuating load is to vary the pressure on the line within the strength limits of the pipe. Fig. 4 shows the practical difficulties in taking common carrier gas into transmission lines on account of the great variation in pressure conditions necessary to insure satisfactory service to the consumers at the delivery end of the line. The pressure on the line that is to receive the gas must, obviously, be lower than the well pressure, or else artificial means must be provided to increase the pressure of the gas that is to be injected into the transmission line.

6. The market for natural gas is not fixed, but varies largely with the season of the year, time of day, temperature of atmosphere, locality, and consumer's caprice in using or not using gas on his premises.

7. Gas consumers will not contract for or agree to use a fixed amount of gas each day, but are entitled by law to take gas as they need it. With industrial consumers the consumption is controlled by business conditions, and plants may be shut down for days at a time, or work over time, without any definite arrangement with the gas company for such fluctuation in service demands. From this, it is evident, from the very nature of the case, that a consumer cannot and will not contract for a definite supply of gas from some shipper, as he cannot tell the shipper how much to forward through the common carrier lines, or when to make deliveries.

8. Leakage in transit must be taken care of, either in the rate paid for such transmission or by allowance for shrinkage at the delivery end.

9. Storage facilities are not commercially feasible for storing natural gas at the intake end, in transit, or at the delivery end of the transmission line. For instance, an 18-in. pipe, 1 mile long, at 300 lb. pressure, will

store but 200,000 cu. ft. of gas as delivered to the consumer at 4 oz pressure.

10. Since the gas in transmission will be thoroughly intermixed, if the quality of the common carrier gas received by the common carrier is different from that delivered, a correction will have to be made for this.

11. The volume of gas taken into and discharged from the common carrier line must be accurately determined. Gas being volatile and invisible cannot be transported or identified or measured in bulk as coal or oil or other commodities; and apparatus for measuring gas in large quantities is expensive to install and operate. Furthermore, the errors in gas measurement, due to defects in the apparatus or the calculations based on the apparatus readings, are not apparent as in other commodities, and after the gas has passed the meter the volume cannot be checked to detect errors in measurement or settle disputes, as is possible with ordinary commodities.

12. Where common carrier gas is delivered to and removed from a common carrier line, it would be necessary to know:

- (a) Specific gravity of gas;
- (b) Pressure of gas;
- (c) Temperature of gas;
- (d) Quality of gas.

The first three are necessary in order to determine the volume. The quality of the gas from different places may vary considerably; thus, there may be a variation of several hundred heat units per cubic foot in the heating value. Occasionally freak wells produce large amounts of sulphur which, if introduced as common carrier gas into the main line, may vitiate the supply in many communities. It is also a well-known fact that gas as it comes from the wells contains an uncertain quantity of moisture and frequently salt water, which, if not properly handled in the field, will get into the main lines, causing damage by lines freezing up and water interfering with the action of regulators and meters.

13. A common carrier natural gas line would be limited to handle *one* commodity; whereas railroads can handle every known commodity.

14. Common carrier natural gas lines can have no extensive inter-connecting service with other lines; whereas, every railroad can handle commodities from every other railroad.

15. The transportation of natural gas is a *local* business, naturally centralized in the fields of production, the deliveries being made near those fields, and not throughout the United States, as are other commodities. The limited area producing natural gas, and limited population using natural gas in the United States, are emphasized in Fig. 2.

16. Features Nos. 1, 2, 3, 13, 14, and 15, just mentioned, emphasize a common error made in comparing natural gas transmission lines with railroad transportation systems, and the fallacious reasoning therefrom

that because railroads are common carriers, natural gas lines can also be made common carriers.

In conclusion, the converting of natural gas transmission lines into common carriers:

1. Presents many features that are impossible, and none that are feasible or expedient.
2. Could not require a company operating a pipe line to pick up gas anywhere along its line and take it to a market except under the *legal*, *engineering*, and *economic* conditions herein defined.
3. Could not require the pipe-line company to find a market for the gas.
4. Would so disorganize the existing business as to make it impossible to render satisfactory continuous service to either domestic or industrial consumers. This would be true regardless of what might be charged for carrying this common carrier gas.
5. Would make the consumers—especially the domestic—subordinate to *occasional producers*; that is, to men who have no intention of following the business of hunting for gas for future service, but would be interested only in finding a good market, at the expense of others, for such gas as might be found as a result of an occasional venture.
6. In all cases, where tried, will impair and usually destroy the cooking, heating, and lighting service of the domestic consumer.
7. Would greatly increase the amount of gas used for manufacturing purposes, thus hastening the day when natural gas will be merely the memory of a wasted and unappreciated resource.
8. Could be based only on distinctly local and selfish interests, and would have to ignore entirely the broad public interest in an effective and continuous service and a future generation's equity in a conserved fuel supply.

The Progress of the Metallurgy of Iron and Steel

BY SIR ROBERT HADFIELD, F. R. S.,* LONDON, ENGLAND

(An address prepared for presentation at the New York Meeting, February, 1914)

Introduction.—I esteem it a great honor to be asked by this Institute to give them an address chiefly devoted to metallurgy. While it is with great regret that I find myself unable to be present to deliver this in person, I take this opportunity of wishing your society every success in its continued career of usefulness to the metallurgists of the great Republic.

To be elected, eight years ago, along with Dr. J. E. Stead, F. R. S., as one of your Honorary Members, was a privilege which I greatly esteemed, for I consider that no society in America has done so much as ours for the advancement of metallurgy.

Those who have Served our Society.—Where many have served so well, it is difficult, without appearing to make unavoidable distinctions, to select names from those of your members who have done so much for the cause we have at heart—the advance and progress of metallurgy. As typical, however, and not in any way meant as a complete list, there are among others the following names which in the past and to-day stand out prominently in this science of ferrous metallurgy. With most of those now mentioned I have personally come in contact, or by correspondence in some few cases, and have exchanged views, since I first came over to America 30 years ago. Holley; Jones; Weeks; Andrew Carnegie; Thomas Carnegie; Fritz; Sterry Hunt; Egleston; Maunsel White and Taylor, who did so much for high-speed tool steel, and helped to revolutionize machine-shop practice; W. J. Taylor; Robert Hunt; Alfred Hunt of Pittsburg, who did so much in the early history of aluminum, and, alas! did not live to see the wonderful progress which has been made in that particular branch of metallurgical industry; James Douglas; C. B. Dudley; Birkinbine; Howe; Metcalf; Barus; Langley; Raymond; P. H. Dudley; H. H. Campbell; W. R. Webster; E. D. Campbell; Talbot; Spilsbury; Gayley; Dinkey, under whose supervision and responsibility modern armor has been so successfully produced in this country, which the writer can testify from personal knowledge of test results possesses very high Figure of Merit; Schwab; Kennedy; Wellman; Garrison; Wood; Moldenke; Webster; Swank; Unger; Richards; Sauveur; Strat-

* Honorary Member.

ton; Burgess; Kirchhoff; Waldo; Stoughton; W. R. Walker; Colby; Cushman; Keep; Sahlin; Clamer; J. E. Johnson, Jr., and others.

As I have said, it is quite impossible to do justice in the above short list, or to prevent omissions. The foregoing list seems, however, to fairly represent the names of those who, by their valuable contributions to metallurgical research work, and quite apart from the business side of life, have freely added to the common stock of knowledge, which is the highest service a man can render to the State.

I would conclude these personal references by saying that our Institute is to be congratulated upon being presided over by Mr. Charles F. Rand, who has devoted his personal time and attention with such conspicuous success to the important and ever-increasing work of the Institute.

Faraday and Metallurgy.—Some portions of my remarks have been prepared rather from the British point of view, for I was recently induced to accept the Presidency of the Faraday Society in England and asked to give them an address, so have been preparing some notes for this purpose. I therefore beg your indulgence should any portion of the address appear to be directed too much from the British standpoint.

Speaking of Faraday and his work, it is not so well known as it should be that apart from his many other scientific activities he bestowed much attention on the fascinating subject and particular corner of scientific research to which I have personally devoted a large part of my life, the study of the numerous alloys of iron with other elements. The extraordinary advances which have been made throughout the world during the last generation, one might say within the last 20 years, have been largely owing to the study and introduction of these new combinations of iron with other elements.

Although Faraday was not the first in this field, yet he was certainly one of the pioneers. This is indicated and set forth in his admirable papers on Alloys of Steel, read jointly with Stodart on March 21, 1822, and appearing in the *Philosophical Transactions of the Royal Society* for that year (p. 253); also in the *Philosophical Magazine* (vol. ix, p. 363). An interesting report of his various experiments is given in his work entitled *Experimental Researches in Chemistry and Physics*, published in 1859.

Among his numerous discoveries were: induction currents, which eventually gave us electric light and the electric motor; the law of chemical decomposition in electrolysis, styled "Faraday's Law"; diamagnetism; the action of magnets on polarized light; the liquefaction of carbonic acid gas and of protoxide of nitrogen, and many others of importance.

The name of the Faraday Society, therefore, is one to conjure with, and one that the Society in England may well feel proud to own. I believe this Society largely owes its inception to the example set by your

American Electrochemical Society, whose valuable work has been made to prosper, and whose progress has been so rapid, owing to the indefatigable labors of Prof. J. W. Richards.

Therefore there does not seem any need for apologizing for the several references made in this paper to Faraday, seeing how great was the interest shown by him in the science which this society so wisely helps to foster, the metallurgy of iron and steel.

As my address deals more particularly with the scientific side of metallurgy, no attempt will be made to deal with the more practical side, or the wonderful part played in practice by the American metallurgical engineer, who has astounded the world by making possible the production of those phenomenal outputs of iron and steel, and by which your country has been able to advance with such leaps and bounds. Such men as Bessemer, Siemens, and Bell, to say nothing of your own great pioneers Holley and Jones, would indeed be astounded if they could have seen such outputs as 30,000,000 tons of steel per annum!

Early Metallurgy and Alloys of Iron and Steel.—The problem of alloys of carbon and of many different elements with the world's leading metal, iron, is an extraordinarily fascinating one. I use the qualitative "leading" advisedly, since without this metal the world would be plunged into uncivilized darkness, and when this metal does fail, for fail it must at some future time, it is difficult to see how the world can avoid returning to the Stone Age at a quicker pace than that at which it emerged from it. Iron is in truth the leading metal: it is constantly used and in everybody's hand and service in some shape or other and for an immense variety of purposes; yet we all recognize that it still embodies a vast amount of secrets, it still affords a wonderful field for research.

One can picture to one's self the joy of the first metallurgist, apparently Tubal Cain, when, in probably the year 252462 B. C., he discovered that a lump of red-looking earthy material tossed by accident in his fire gave quite another material, a "metallic" product which could be made to cut. This first member of the prehistoric Iron and Steel Institute was not long in recognizing the far-reaching consequences of his accidental discovery. He, no doubt, first made weapons—a crude type of javelin presumably—from this new product, but appalled at the rate at which the men of his period had become able to destroy one another by reason of this discovery of his, he probably endeavored to counteract the action of the new offensive weapon he had been able to put in their hands by providing them with defensive ones, and he gave them "metallic" shields. But by that time others had entered the field and started producing both classes of weapons, making each alternately the stronger, and so matters progressed. The idea caught on, and Tubal Cain, besides being the first metallurgist, was probably also the first man to initiate the "projectile versus armor" conflict, a conflict which has continued ever since, is being

continued to this day, and one in which my city of Sheffield, I venture to think, has acquired a world-wide renown. As regards this early history of metallurgy, I have very fully dealt with comparatively early metallurgical products, though only going back some 15 or 16 centuries, in my paper to the Iron and Steel Institute, Sinhalese Iron and Steel of Ancient Origin, read in May, 1912.

Ancient Iron and the Delhi Pillar.—It may be interesting to add here that thanks to the kindness of J. H. Marshall, C.M.G., C.I.E., the Director-General of Archaeology in India, since my paper was written I have been able to obtain a specimen of the famous Delhi Pillar, which is about 24 ft. in length and weighs $6\frac{1}{2}$ tons. It was found to have the following composition:

C.	Si.	S.	P.	Mn.	Fe.
0.080	0.046	0.006	0.114	nil	99.720

Specific gravity, 7.81 per cent. Brinell ball hardness number, 180.

This unique specimen is probably one of the finest specimens of iron work ever produced until modern times. In other words, until less than 100 years ago a forging of such length and weight could not have been obtained! Therefore, great credit is indeed due to the Eastern metallurgist who worked up such a specimen of the metallurgical art. As to the manner in which this was accomplished, I have dealt with the problem in my paper before referred to.

Faraday's Experiments in 1822.—From Tubal Cain to Faraday is a far cry indeed. The latter was one of Britain's noblest sons. In the earlier years of his activity he attacked the problem for the sake of the problem itself. His whole life was one of untiring energy, and a detailed record of his researches would fill volumes of any Institute's proceedings. He devoted a certain amount of attention to the study of metallurgy. As regards his metallurgical research work, fully described in the source to which I have previously referred, it was in 1822 he mentioned in one instance that he himself could not go to Sheffield, where he intended to have his alloys made in continuance of the experiments he and Stodart had started at the Royal Institute, but that he was carefully preparing in London the mixtures to be melted in Sheffield, placing them in separate parcels. The work was to be given to a trusty assistant, who was to go down and see the experiments put in hand and completed at Sanderson's works, this firm being still in existence to-day. In those early days there was only one way of sending the specimens from London to Sheffield, that is, by coach, a long journey, and one can picture the great scientist carefully sorting out his materials, various alloys and iron, for mixing together in order to produce the different classes of material he had in view, and carefully packing them in different parcels.

Faraday foresaw that metallurgy was destined to become one of the

leading branches of science, and that in a general way it is upon progressive metallurgy most of the advances in modern effort and progress are dependent. I say this advisedly, for what would life be to-day without the aid of the metal iron and its alloys and combinations. This I pointed out very fully in my Presidential Address to the Iron and Steel Institute in 1905. So much was I then imbued with this spirit that in that address I gave the portraits of some 21 early scientists, largely Swedish, to whom the world owes such a great debt of gratitude with regard to the work they did in first separating many of the various elements now used in the alloys of iron and steel. Probably in no country was more good work done than by these early scientists in Sweden.

Early Workers in Metallurgy.—It may be interesting to give a list of these early workers in scientific metallurgy; of those against whose names an asterisk is placed I have been able to obtain portraits.

Early Workers in Scientific Metallurgy

Aristotle (Greece), 384–322 B. C., philosopher, described the manufacture of Indian steel.

Theophrastus (Greece), 374–287 B. C., philosopher, wrote on iron ore.

Diodorus Siculus (Sicily), 60 B. C., geographer, described the smelting of Elba ore.

Plinius (Italy), 79–23 B. C., naturalist, showed that the properties of iron depend upon its treatment.

Rugerus Theophilus (Germany), monk, dealt with iron in his *Schedula diversarum Artium* (1050).

Leonardo Da Vinci (Italy), 1452–1519, physicist, artist, etc., introduced blast for smelting iron.

Vanuccio Biringuccio (Italy), 1480–1539, dealt with iron in his *Pyrotechnia* (1540).

Georgius Agricola (Germany), 1490–1555, dealt with metallurgy of iron in his *De Re Metallica* (published 1556).

Lazarus Ercker (Germany), wrote on assaying (1574).

Louis De Geer (Sweden), 1587–1652, introduced the Walloon furnace.

Dud Dudley (England), 1599–1684, first smelted iron with pit coal.

Johann Rudolf Glauber (Germany), 1604–1668, chemist, discovered several metallic chlorides.

Simon Sturtevant (England), dealt in his *Metallica* with the use of coal for smelting (1612).

John Rovenzon (England), dealt in his *Metallica* (1613) with the use of coal for smelting.

*Sir John Pettus, Knight (England), 1613–1690, author of metallurgical works.

Andrew Yarranton (England), 1616–1684, introduced tin plate manufacture.

The Hon. Robert Boyle (England), 1627–1691, chemist and physicist.

*Sir Isaac Newton, Knight (England), 1643–1727, scientist.

*René Antoine Ferchault Réaumur (France), 1683–1757, physicist, wrote on cementation and decarburization.

*Emanuel von Swedenborg (Sweden), 1688–1772, engineer and theologian, wrote on iron.

G. Brandt (Sweden), 1694–1768, discovered cobalt (1733).

*Benjamin Huntsman (England), 1704–1776, manufacturer, first to melt steel.

- *Benjamin Franklin (United States), 1706–1790, statesman and scientist, inventor of the Franklin stove.
- Johann Andreas Cramer (Germany), 1710–1777, author, wrote on metallurgy.
- Abraham Darby (England), 1711–1763, used coke for smelting.
- *Sven Rinman (Sweden), 1720–1792, wrote on metallurgy of iron.
- *Axel Frederic Cronstedt (Sweden), 1722–1765, mineralogist, discovered nickel (1751).
- *Joseph Priestley (England), 1733–1804, chemist, discovered oxygen (1774).
- *Richard Kirwan (Ireland), 1733–1812, author, first English writer on mineralogy.
- *Torbern Olaf Bergman (Sweden), 1735–1784, chemist, founder of analytical chemistry, drew distinction and explained differences between pig iron, wrought iron and steel.
- *Henry Cort (England), 1740–1800, engineer, introduced puddling and the use of grooved rolls.
- *Karl Wilhelm Scheele (Sweden), 1742–1786, chemist, anticipated Priestley's discovery of oxygen, discovered manganese (1774), chlorine, barium, and molybdenum.
- *Martin Heinrich Klaproth (Germany), 1743–1817, mineralogist, discovered uranium (1789) and titanium (1794).
- *Antoine Laurent Lavoisier (France), 1743–1794, chemist, creator of modern chemistry.
- Johann Gottlieb Gahn (Sweden), 1745–1818, chemist, isolated phosphoric acid.
- *Claud Louis De Berthollet (France), 1748–1822, chemist, discovered composition of ammonia.
- Fausto De Elhuyar (Spain), 1755–1832, chemist, first prepared metallic tungsten (1792).
- William Reynolds (England), 1758–1803, patented (1799) the use of manganese oxide for steel manufacture.
- John Dalton (England), 1766–1844, chemist, published table of atomic weights.
- Jeremias Benjamin Richter (Germany), 1762–1807, chemist, invented alcoholometer.
- *Louis Nicholas Vauquelin (France), 1763–1829, chemist, discovered chromium (1797).
- David Mushet (England), 1772–1847, metallurgist, author, discovered black-band ore.
- Sir Humphry Davy (England), 1778–1829, chemist, first isolated potassium and sodium.
- *Jöns Jakob Berzelius (Sweden), 1779–1848, prepared pure iron, discovered selenium, thorium, and cerium, isolated silicon, and introduced the theory of allotropy.
- Pierre Berthier (France), 1782–1861, chemist, wrote on metallurgical analysis.
- Robert Sterling (Scotland), 1790–1878, patented regenerative principle (1817).
- *Michael Faraday (England), 1791–1867, chemist and physicist, investigated the properties of iron and steel alloys.
- James Beaumont Neilson (Scotland), 1792–1865, invented hot blast (1828).
- *Friedrich Wöhler (Germany), 1800–1882, chemist, discovered aluminum (1827).

Conservation.—I may add, although it may seem a truism, that the investigation of alloys of iron is exceedingly important and of great practical benefit and utility not only from the point of view of obtaining the required combination for a given purpose, but also from that of obtaining for a given quantity of metal the maximum degree of utilization. As above alluded to, the world's resources in the matter of iron ores are not inexhaustible, and it may be incumbent upon us before long to more carefully husband them. The United States is now annually producing

no less than 33,000,000 tons of pig iron, Germany some 20,000,000 tons, Great Britain, France, and Belgium nearly as much more, all involving enormous consumption of iron ore. Quite apart from these two points of supreme economic interest, the study of alloys, as I have already stated, is a most fascinating study and indeed a very wide one. The rapid introduction of alloys of iron with other elements shows that they have met the necessity of the times. An alloy steel, although it is more expensive, is really an attempt at conservation; it is possible to make a pound of alloy steel go as far as many pounds of ordinary steel.

It must be remembered, too, that, as in the past, the production of iron by different countries must vary from time to time. The country leading at one period may not necessarily do so for all time. The great and leading point is to provide for a source of supply of the ore.

The well-known French writer, Henri Gaston, in his interesting brochure *Où va l'Allemagne* points out that Germany will in the near future exhaust her own supply of iron ore; then what is she going to do? She controls no outside sources of any great value. Without introducing politics, perhaps there was more than met the eye in Germany's interest in Morocco with its great stores of iron ore. Gaston claims that France has a much greater and more certain metallurgical future. He goes so far as to say "Nous sommes destinés, si rien ne vient contre carrer nos desseins, à devenir la nation métallurgique la plus puissants du monde." I have written these few remarks to show that any temporary heading of the list of nations producing pig iron cannot by itself alone be considered. There are many other factors.

French Metallurgy in the 18th and 19th Centuries.—Two remarkable books were published in 1846, one *L'Art de Tremper les Fers et les Aciers*, by M. Camus, and the other *Mémoires sur la Fabrication et le Commerce des Fers à Acier dans le Nord de l'Europe*, by M. F. le Play, Ingénieur en chef des Mines, Professeur de Métallurgie à l'École Royale des Mines. That by M. Camus, who is described as "Ingénieur-Manufacturier, membre de la Société d'encouragement pour l'Industrie nationale, ex-Ouvrier Contre-Maitre et Directeur-Fondatent de plusieurs Usines et Manufactures métallurgiques en France," deals in a most practical way with the methods required to obtain the best qualities from steel which has to be hardened and tempered. Much of the information he gives is quite as applicable to-day as when it was first written. He furnishes particulars of various temperatures as they were then estimated by the Wedgwood pyrometer, heat colors, and other information. I reproduce one of his tables in full (Table I), as it shows the care and attention then bestowed by this French expert. This list was published in 1846, showing that excellent work had even then been done with reference to the determination of high temperatures.

TABLE I.

Centifex	Détail, Couleur des Chaudes	Millifex	Correspondance Thermomètre Centigrade. Fraction Négligée	Pyromètre de Wedgwood
1	2	3	4a	5
Degré		Degrés	Degrés	Degrés
0	EAU bouillante, ZINC, état de ductilité pour son laminage.....	0	100	—
0	REDRESSAGE à la presse des objets trempés			
	Maximum de chaleur.....	3	153	—
0	ETAIN, son premier point d'entrée en fusion	6	206	—
0	Jaune-PAILLE, première recuite de trempe (sèche).....	7	224	—
0	Id. d'OR, deuxième id. (moins).....	8	242	—
0	GORGE-DE-PIGEON, troisième id. (moins)			
	Entrée en fusion du Bismuth.....	9	260	—
1	VIOLET, quatrième id. (moins).....	10	278	—
1	Bleu-FONCÉ, cinquième id. la plus douce à donner.....	11	295	—
1	PLOMB, il entre en fusion à forcée.....	13	330	—
1	MERCURE, id., à forcée.....	15	366	—
1	ZINC, id., à forcée.....	16	384	—
1	Expiration forcée de la trempe, à cette chaleur il y a recuit.....	17	402	—
1	ANTIMOINE, il entre en fusion à forcée à.....	19	437	—
1	Rouge-BRUN, rien pour la trempe, recuit de petits objets à la forge.....	27	579	O _{ass}
1	ARGENT, il entre en fusion à.....	108	2,018	20
11	LAITON, id., à.....	112	2,089	21
14	CUIVRE, id., à.....	137	2,533	27
16	OR, id., à.....	157	2,888	32
18	Rouge-CERISE, à.....	185	3,196	39
22	Rouge-VIF, à.....	226	4,114	49
26	Rouge-CLAIR, à.....	262	4,753	58
30	Rouge-ROSE, à.....	299	5,410	67
33	Rouge-BLANC. Au feu de forge de faibles étincelles partent des angles du fer, à.....	335	6,049	76
35	BLANC PUR, maximum d'éprouvette, fin de cémentation (fabrication d'acier) à.....	351	6,334	80
40	Premier BLANC-SUANT, soudure du fer tendre, carburé, à.....	400	7,204	92
41	Deuxième BLANC-SUANT, à goutelettes métalliques, soudure du bon fer ordinaire, à.....	412	7,417	95
43	Troisième BLANC-SUANT, id., soudure du fer très-pur, à.....	430	7,736	99
51	FORTE GRISE, au creuset, première fusion agglutination (réunion) à.....	513	9,211	120
55	ACIER de cémentation au creuset; il entre en fusion à.....	554	9,939	130
57	FORTE GRISE, id., en état de bain, à.....	570	10,222	134
58	FER, il entre en fusion pleureuse, à.....	574	10,294	135
59	ACIER de cémentation, maximum du bain, à.....	586	10,507	138
60	MINÉRAI, état de bain (Haut-Fourneau), à.....	595	10,667	140
63	FER, il y a agglutination (réunion), à.....	635	11,378	150
65	Id. état de bain, à.....	655	11,733	155
67	Id. No. 1, maximum du bain, à.....	676	12,106	160
70	Id. très-pur, id., à.....	700	12,532	166

*As some of the figures in column 4 are so high, probably M. Camus had the Fahrenheit scale in mind. Even then the figures are of course entirely incorrect in the light of our present knowledge.

Turning to the work by Professor le Play, this writer shows how for nearly a hundred years or more French metallurgy as regards the production of high-class steel had been entirely misdirected by the idea to which they obstinately clung for so long that it was possible to produce in France bar iron for making steel of the same quality as produced by Sweden; this too in face of the wonderful progress then made by the English steel makers, who had wisely adhered to the importation of high-class Swedish irons.

This book has for the first time made me aware that Réaumur¹ in 1715, by his elaborate researches published from 1720 to 1722, although a great scientist in many ways, made a terrible mistake which hampered French industry for a long period, when he proved to his own satisfaction, as he thought, that it would be possible to treat irons made in France so that their quality would be equal to the pure products of Sweden. Owing to his misapprehension of the true conditions attaching to the production of high-class steels, very large sums of money were lost in France, both under the Monarchy and the Republic. This shows how exceedingly careful a scientific man should be in advancing theories not fully tested and confirmed in practice.

The position which had already then been reached by Sweden is but another proof of the debt of gratitude the metallurgical world owes to that country. As Professor le Play said in 1846 (p. 91).

"For two centuries the Swedish metallurgists have exploited with great cleverness the steel-making quality of their minerals. This branch of the metallurgy of iron is due to their efforts."

Professor le Play points out (p. 99) how it was that Réaumur, clever as he was, had gone wrong.

"The disastrous results from which the French steel industry suffered, by following the advice of Réaumur, did not change the opinion of the practicability of the process, although based merely on laboratory experiments. The most distinguished scientific men and the most eloquent writers persisted in maintaining that French iron was very suitable for steel making, and the government, much influenced by their arguments, continued to insist on the use of this iron as a condition in making any grant to assist any infant industry."

Works after works in France came to grief owing to the obstinacy with which the French clung to their methods of trying to utilize their own Empire's products at the expense of quality. Even when works were started, as for example those of *Acéries de Nerouville* in 1770, where Swedish iron was exclusively used, special pressure was brought to bear to compel them to discontinue using the imported product, researches being again renewed in order to try and utilize French iron, again only with the same disastrous consequences.

¹One of Réaumur's well-known books was *L'art de convertir le fer forgé en acier*, Paris, 1722.

Professor le Play further points out (p. 132):

"As I have already said, if Réaumur, in 1722, had not misled those engaged in the industry, it would have developed, as in England, by using Swedish iron. It would have done it with probable success, since, during the whole eighteenth century, England believed it should lay taxes on the right of importation of foreign iron.

I have given the foregoing at some length because it is interesting to see that British common-sense and practical way of regarding matters of this kind kept them on the right road. It was good judgment of no little value, for it resulted in Sheffield becoming the center of the special steel trade of the world, a position which it still holds. The French idea was that the quality of the material, then known as *body*, was to be obtained by purifying the products of their more imperfect ores. But, as we all know, the idea was not practical and Sweden still holds its position in the world for the production of the purest quality of raw material. There is more Swedish pig and bar iron now being used than at any previous time.

Le Play also mentions, and I think this is specially interesting in view of Faraday and Stodart's experiments in 1820, that these commenced owing to Faraday's having analyzed certain specimens of material from India known as Wootz steel. A similar series of experiments were then undertaken in France. Assistance was given to the project by the well-known Société d'Encouragement pour l'Industrie Nationale, a society still existing and doing most excellent work. This society found the necessary funds for carrying out some 300 experiments upon alloys of iron with different metals. A report on this subject was presented to the society in 1821. The experiments were continued to the year 1824, but did not appear to result in any manufacturing benefit. The Société d'Encouragement pour l'Industrie Nationale then granted gold medals to several investigators in the art of steel making.

Value of Research.—Let us remember that it is to the valuable properties of the many alloys of iron now made, from carbon steel to the complex one known as high-speed tool steel, which contains no less than five different elements apart from the iron itself, is due the remarkable progress we have made, whether in the arts of peace or war. I will give one simple concrete instance: the modern motor vehicle, whether for private or trade use, with which such an enormous traffic is now carried on with so great convenience and comfort to the public of all lands. Take away the alloy steel used in its construction, and it could no longer be produced. The combination of lightness and strength necessary in such modern products is only possible by the use of special alloys steels.

To study the qualities of alloys of iron with other elements involves much research work, which is a noble occupation for mankind. While

this work may not appear so attractive as the discovery, for example, of a new element, it is not less important.

The value of research work has early recognized in Great Britain, for in the very interesting *History of the Royal Society of London for the Improving of Natural Knowledge*, published in 1702 and written by "Tho. Sprat, D. D., Lord Bishop of Rochester," it is said in the Preface or Dedication portion to the King (Charles II):

"Of all Kings of Europe, Your Majesty was the first who confirmed this noble design of Experiments, both by your own example and by the public establishment of the Royal Society. An Enterprise equal to the most renowned actions of the best Princes. For to increase the Powers of all Mankind, and to free them from the bondage of errors, is greater glory than to enlarge Empire, or to put chains on the necks of Conquered Nations. A higher degree of Reputation is due to Discoverers than to the Teachers of Speculative Doctrines, nay even to Conquerors themselves. Nor has the True God Himself omitted to show His value of vulgar Acts. In the whole history of the first Monarchs of the World, from Adam to Noah, there is no mention of their Wars or their Victories. All that is recorded is this. They lived so many years, and taught their posterity to keep Sheep, to till the Ground, to plant Vineyards, to dwell in Tents, to build Cities, to play on the Harp and Organs, and to WORK IN BRASS AND IRON. And if they deserved a Sacred Remembrance for one Natural or Mechanical Invention, your Majesty will certainly obtain Immortal Fame, for having established a perpetual Succession of Inventors."

Quaint as may seem these words to us to-day, there is indeed still now as great truth in them as when they were uttered. The marvelous recent progress of the world has been largely due to the scientist, to the inventor, and not least in these branches of human thought are those who work in "Natural or Mechanical Inventions," or, to use again the Bishop's words, "In Brass and Iron," though nowadays the latter metal plays the more prominent part.

Since the date when these wise words were uttered, the Royal Society has seen one long-continued career of success, and owing to the devotion of its Fellows to the cause of science and scientific progress, mostly without reward of any kind, there is no country where science and scientific work is more highly esteemed than in Great Britain. I know it is often customary to say that we lack appreciation of scientific merit, but I do not believe it. The fight for the cause of advance may be severe, but in the end we English recognize to the full true merit. We need not be ashamed when from this little island have come—not to mention those still with us—such men in the Anglo-Saxon race as Bacon, Newton, Priestley, Dalton, Boyle, Cavendish, Faraday, Davy, Huxley, Tyndall, Spencer, Darwin, Kelvin, Thompson, Dewar, Rayleigh, Oliver Lodge, Percy, Roberts-Austen, and many others. On the more technical side, Dudley, Sturtevant, Pettus, Huntsman, Darby, Kirwan, Cort, Heath, Heaton, Mushet, Bessemer, Siemens, Bell, Whitwell, also many others in the Old World.

In the New World, such men, as regards both classes of research, as Benjamin Franklin, Fulton, Agassiz, William Gibbs, Rowland, Barus, Edison, Bell, Stienmetz, Sterry Hunt, Howe, Holley, Fritz, and others who are referred to in the earlier part of this address.

The foregoing list is necessarily a very incomplete one and refers only to the workers in the Anglo-Saxon race. It, however, represents at any rate some of those who have done the greatest work in the domains of the sciences of chemistry, physics, electricity, engineering, and metallurgy.

The Author's Research Work.—At the time I first started my own research work, in 1882, little was known on the subject of alloys of iron. In fact, in the light of our present knowledge, one might safely say that the subject was practically an unknown one.

Mars, the eminent German writer, in his recent book *Die Spezialstahle*, was good enough to say:

"The most extensive experimental researches, which may be said to have laid the foundation for entire knowledge of steel alloys, were carried out by Hadfield in the eighties of the last century. . . . His two works on Manganese Steels and Silicon Steels have become of decisive importance as regards the development of the technics of steel alloys. They not only supplied a wealth of important facts from the purely scientific point of view, but they subsequently led up to the evolution of many new qualities of steel.

"The first thorough researches concerning silicon and iron alloys which could claim to have investigated the properties of silicon steels from the standpoint of the steel manufacturer, with a view to the uses of the alloys examined, were published by Hadfield. This work was the second of the series of publications forming the foundation of the entire modern knowledge of alloys of iron."

Pourcel, the eminent French metallurgist, said that he "considered the production of manganese steel the most important event in practical metallurgy during the last ten years, and which might take its place beside the result of the labors of Gilchrist, Bessemer, Siemens, Martin, and Mushet."

Dumas, in his *Recherches sur les Aciers au Nickel à Hautes Teneurs*, said that my alloy studies were "the first publications which had been made upon alloys containing high percentages of elements other than iron."

Bradley Stoughton, in his excellent work *The Metallurgy of Iron and Steel*, was good enough to make similar expressions.

Finally, Osmond, the great French scientific metallurgist, said that my discovery of manganese steel was of the same order and equaled in importance the discovery in past centuries of the effect of quenching upon carbon steel.

I have referred to these quotations in order to show the truth of my earlier remarks that when first starting my labors in the eighties of the last century, the general knowledge available on this subject was very small indeed.

Definition of the term "Alloy."—In this connection, I should like to add that the conception which formerly ruled in regard to the term "alloy" prevails to this day in the mind of the general public. An alloy to many implies nothing more than the addition of a baser metal to a finer one, the object of the mixing of the two together being to obtain a cheaper commercial article and one that will wear better. This, we all know, is not at all the case in regard to special steels and special steel alloys or combinations of the present day. The definition of the word alloy might better be made to read "the combining by fusion of two or more metals together, or of a metal with one or more metalloids, for various specific purposes."

Metallurgy in the Middle of Last Century.—A most valuable work was issued in 1867, Kohn's *Iron and Steel Manufacture*, the best of its kind for almost a generation, and for the origin of which we have largely to thank those who have given us the modern and most valuable technical journal known as *Engineering*. This served as a text-book for over 20 years and now remains of much historical interest. This book gave among its 270 pages not more than about half a dozen references to metallurgical papers. How little this seems when compared with the special literature now available! Take, for example, the well-filled *Journals* of the Iron and Steel Institute, containing as they do annually about 1,000 pages of original matter, being the reproduction of about 30 papers with the discussions thereon, and about 500 pages more of notes showing the progress made in the home and foreign iron and steel industries; they afford an illustration of the huge step forward which has been made in this direction since the "sixties." The *Journals* of this Institute, devoted as they are to ferrous metallurgy, have now entered upon their eighty-eighth volume, besides which there have been issued from time to time a number of extra volumes devoted entirely to the reproduction of the Carnegie Scholarship memoirs. The Iron and Steel Institute had small beginnings, but it has developed into a most powerful body, and one whose influence is felt the world over. To its able Secretaries, J. Stephen Jeans in the past, now gone from us, and G. C. Lloyd to-day, the metallurgical world is greatly indebted.

Carnegie Scholarships.—I would specially like to remind the members of this society that a number of Carnegie scholarships are yearly granted by the Council of the Iron and Steel Institute. While the applications for these scholarships are numerous, it is to be hoped that our younger members will not fail to come forward in still larger numbers to try and avail themselves of the privileges to be obtained under these scholarships. Mr. Carnegie's munificence has proved of the highest benefit. It must be a great satisfaction to him to know that his generosity has proved of advantage to so many of our younger members in the pro-

fession of metallurgy, both in this country, Great Britain, and on the Continent.

The Past Work of our Institute.—Our own society, the American Institute of Mining Engineers, has done splendid work in the past. I understand that no less than 47 volumes of the *Transactions* of our Institute have been issued from the year 1871 to 1913 inclusive. The membership roll now amounts to no less than 4,200.

Who can forget the great meetings in New York, Philadelphia, Pittsburg, Chicago and elsewhere, during the year 1890, when you welcomed the Iron and Steel Institute, presided over as it so ably was by Sir James Kitson, afterward Lord Airedale; also the results of the wonderful International Congress in Chicago in 1893, pioneered and arranged by this Institute, covering as it so fully did our branch of science, metallurgy.

The fraternal feeling which has so long existed between metallurgists of the two countries was also shown in the historic gathering of the two metallurgical societies of America and England at the memorable Guildhall Banquet during my Presidency of the Iron and Steel Institute in 1906. The many expressions of good will which were then exchanged added to this feeling.

This also brings back the echoes of a similar and equally interesting gathering of the two societies in the year 1890 so ably addressed as it was, as those present well remember, by your great General Sherman, whose words of encouragement to the Anglo-Saxon race to press on in the field of progress, and his offer of help to us on the other side, if necessity demanded, I shall never forget.

The Good Feeling between Great Britain and the United States.—During this and next year, as you are probably aware, there is to be the celebration of the One Hundred Years of Peace between the great English-speaking nations of Great Britain and the United States. May the good feeling ever continue to grow as the years roll on. Each country is working for freedom of thought and for the gradual uplifting of the masses to a position of greater comfort. While the sea may divide us, and while from time to time inevitable little differences of opinion, possibly competition between each other, may arise, these are but passing clouds. Among other things, you may send and flood us with motor cars; we may occasionally send you products which appear to compete with those of your own makers, but competition is good for both of us. Some day perhaps we may be united in one great and common bond of Anglo-Saxon brotherhood.

Steel Alloy Research.—As regards steel alloy research apart from carbon steel—though even the latter might also be included, as so comparatively little was known concerning it but 50 years ago—after carefully looking through the pages of Kohn's book above referred to, I can

find in it but very little evidence of work having been done in this direction. I have specially selected this book, as a starting point as it were, because it represented at the time and for many years after its publication, in 1867, a very complete illustration of the practical side of thought which then prevailed. At that period, no one appeared to have given very much thought to the effect which numerous elements can have upon iron and to the possibilities in store for an alloy obtained by combining them with iron. In its 270 pages, forming a very complete treatise on siderurgy and mechanics, I cannot find a glimmering of that enormously important development which was to be due to alloy steels.

I use these last words advisedly to depict the situation then existing compared with what obtains at the present day; and as I have already pointed out, as further illustrating the point, taking one branch of progress alone, the manufacture of a motor car, if you take away special alloy steels that manufacture becomes an impossibility. Further, without special steels, our battleships would still be provided with a protective covering of a quality only equal in resistance to that of the first armored ship, "La Gloire," and would be, comparatively also, quite as ineffective as the latter as far as offensive power is concerned; but even this great advance of iron plates only occurred about 50 years ago.

Ferro-Manganese.—Kohn in his excellent article devoted to the subject of ferro-manganese, a product of such great importance to the world's steel makers, does say that "steel makers are now entirely dependent upon spiegeleisen, which gives good results, but they recognize the want of a richer manganese alloy, and this will probably make itself felt all the more when the attention of Bessemer steel makers is more largely turned to the manufacture of armor-plates." But, in thus speaking of a "richer manganese alloy," Kohn had evidently in mind the action of manganese in the carrying out of the Bessemer process. He was thinking of the solubility of ferrous oxide in the bath and its attendant disadvantage, namely, that of yielding a metal saturated with the oxide, a disadvantage which is done away with by using manganous iron, the manganese reducing the oxide when it remains unreduced by the silicon, the carbon, or the phosphorus and protecting the metal against a saturation of ferrous oxide. Kohn's remark on this point was written in 1867. That is hardly 50 years ago, and it was not until some 20 years later, by the combination of scientific research with every-day work practice, that the "richer manganese alloy" he referred to was developed in the shape of ferro-manganese. I well remember that in the year 1886 down in Sheffield we paid as much as £120 per ton for ferro-manganese of quite indifferent quality, judged on the standard of to-day, containing about 60 per cent. of manganese, and were glad to get even that. The following represents the analyses of two lots of ferro-manganese with which some of my early experiments were carried out in the year 1883:

Made by	C.	Si.	S.	P.	Mn.
Terre Noir Co. (France).....	6.0	0.65		0.123	81.60
Pyle & Blaina Co. (South Wales).....	5.8	0.67	0.02	0.23	81.50

Since the above date the quality of ferro-manganese has been much improved, and the modern product would contain less phosphorus and less silicon.

Science and Practice Working Together.—To-day ferro-manganese of very high quality, containing from 80 to 84 per cent. of manganese, can be obtained at under £10 per ton. This speaks volumes in favor of what science and the mutual working together of science and practice can achieve. And in this connection, I cannot help referring again to the admirable way in which the leading metallurgical society, the Iron and Steel Institute, has forwarded this combination of the two, not in Great Britain alone, but universally also. As was pointed out at the Spring Meeting in 1912, by A. Greiner, the able President-elect of that Institute, "twenty-four nationalities were represented within it, from which he thought it would be agreed that science recognized no boundaries of nationality or of race. They worked together not only for the good of the particular country to which they belonged, but for the advancement of scientific knowledge in the world at large."

Incidentally, as showing the wonderful advances which the studies of the metallurgical engineer have made possible, the following interesting fact may be referred to. Although an astonishing statement, it can be accepted as correct, as the figures were given to me by one of your experts: namely, that assuming the cost of pig iron in America is about \$14 per ton at the furnace, and excluding certain charges of comparatively small nature, then on this figure of \$14 per ton, the cost of labor is only about 36c. This result is obtained largely by the ingenuity and skill of the metallurgical engineer.

To put it another way, the approximate cost per ton of pig iron has been estimated as follows:

Coke	\$2.00
Ore	5.00
Limestone	1.00
Assembling coke, ore, limestone	5.00
Supplies	0.60
	<u>\$13.60</u>

This leaves only 36c. to 40c. per ton to pay the bill for the human labor employed in the production of one of the first necessities of life, "pig iron," including in this term the product of the blast furnace.

Development of Research During the Last Fifty Years.—In regard to the

alloys of iron with other elements than carbon, this field of research, the development of which I claim without fear of contradiction to be one of the most marvelous of the many extraordinary developments of the last 50 years, was indeed a narrow one even so near our time as the sixties and seventies of the last century. As showing how scanty was the metallurgical literature of those days, even as late as 1870, Sir William Crookes, F. R. S.,^{*} who was joint author with Dr. Ernst Röhrig of an adaptation of Professor Kerl's *Practical Treatise on Metallurgy*, says (vol. iii, p. 7): "There are very few published works on the manufacture of steel; the following treatises are those most worthy of mention:

- DAMEMME, *Practisches Handbuch der Stahlbereitung*; deutsch von Hartmann, 1839.
 OVERMANN, *Die Bereitung und Verarbeitung des Stahls*; deutsch von Hartmann, 2 Auflage, 1856.
 GRUNER et LAN, "Etat présent de la Métallurgie du Fer en Angleterre; Paris, 1862.
 JULLIEN, *Handbuch der Eisenhüttenkunde*; deutsch von Hartmann, 1861.
 TUNNER's *Abhandlungen in Leoben, Jahrbuch*.
 SCHAFFHAUTL's Artikel über Stahl, in Bd. XV von Prechtl's technologischer Encyclopädie.
 LANDRIN, *Traité de l'acier*; Paris, 1859.
 DESBOYE, *Etudes théorétiques, etc.*; Paris, 1859."

It is curious to note that there is no reference here to Dr. Percy's *Metalurgy*; on the other hand, perhaps it was not then very well known. Contrast this with the admirable and very full metallurgical literature available to-day!

Effect of Carbon upon Iron.—An extract from the *Practical Treatise on Metallurgy* in question reads as follows. It deals, as will be seen, with iron and carbon alloys exclusively. The extract is from vol. iii ("Steel, Fuel, Supplement"), pp. 21, 22:

"Ever since the Swedish chemist, Bergmann, in the second half of last century called attention to the part which carbon plays at the formation of different kinds of iron, steel has been considered as a combination of iron with, in certain limits, a variable proportion of carbon, which proportion, and its more or less uniform distribution, influences the quality of the steel; this view is still very frequently entertained.

"Accepting this view, Gurlt [*B. u. h. Ztg.*, 1860, p. 502; *Berggeist*, 1860, No. 19] considers steel as a combination of $\text{Fe}_3\text{C} + \text{Fe}$; Tunner [*Leob. Jahrb.*, 1861, x, 480] as a combination of $\text{Fe}_4\text{C} + \text{Fe}$; Lohage [*Zeitschrift des Vereines deutscher Ingenieure*, 1857, p. 268; *B. u. h. Ztg.*, p. 48; *Berggeist*, 1858, p. 14] as certain combinations of carbon (Fe_{11}C , Fe_{10}C , Fe_{11}C) consisting of tetrahedral molecules which admit of hardening, in opposition to compounds forming octahedral molecules which cannot be hardened.

"Mayrhofer [*Leob. Jahrb.*, 1861, x, 437] has suggested the following chemical formulæ for different kinds of steel:

^{*} Sir William Crookes, one of the most marvelous men of the age, now occupies the position of President of the Royal Society.

	Form- ula	Fe	C.	Fe *	C. *
Hard natural and cast-steel	Fe_8C	98.16	1.84	96.55	3.45
Cast- and shear-steel, not fit for springs and wire owing to its brittleness	Fe_7C	98.42	1.58	97.03	2.97
	Fe_8C	98.61	1.39	97.39	2.61
	Fe_9C	98.77	1.23	97.67	2.33
	Fe_{10}C	98.89	1.11	97.90	2.10
Cast- and shear-steel, elastic and fit for springs and wire	Fe_{11}C	98.99	1.01	98.09	1.91
	Fe_{12}C	99.06	0.94	98.25	1.75
	Fe_{13}C	99.14	0.86	98.38	1.62
	Fe_{14}C	99.20	0.80	98.49	1.51
	Fe_{15}C	99.25	0.75	98.59	1.41
	Fe_{16}C	99.30	0.70	98.68	1.32
	Fe_{17}C	99.34	0.66	98.76	1.24
Soft puddled steel, fine-grained iron	Fe_{18}C	99.38	0.62	98.82	1.18

Compounds below Fe_{18}C can no longer be hardened, and pass into wrought-iron. According to Mayrhofer, welded Brescia steel of the composition Fe_{14}C is more elastic than English cast-steel of the composition Fe_{12}C ; some English steel for clock springs has the composition of Fe_{11}C or Fe_{12}C , and some of this steel that from Fe_{11}C to Fe_{17}C .

" . . . Fuchs [Dingl., Bd. 158, p. 209; 1860, pp. 43, 124] seemed to be of the same opinion, and he considered the different crystallisation as the essential difference between steel, wrought-iron, and cast-iron. He considered wrought-iron as an aggregation of cubic crystals, cast-iron as an aggregation of rhombohedral crystals, and steel as standing between the two; on hardening the steel, the cubes were transformed into rhombohedrons, and on slowly cooling the reverse took place."

I have quoted this somewhat fully because the opinions expressed are not a little remarkable. It will be seen that the chemist of that day had formulated theories which at any rate to some extent were running in the direction in which modern research seems to be tending with regard to combinations of iron and carbon. It seems to me, too, that we have not followed up this particular direction of research as much as might have been the case. There is a rich and profitable field for the younger chemist if he will devote his attention to this particular branch of chemical research.

Prize for Research Work.—In fact, to show the importance I attach to this subject, I have suggested to our Secretary, Bradley Stoughton, that I should be glad to offer a prize of \$1,000 for research. While it is naturally somewhat difficult to at once define the scope of such research, what has been running in my mind is as follows: The best methods of determining the forms of carbon in steel or iron, including those in iron alloys. In fact, a continuation of such work as has in the past been

* The formulæ for the different percentages of iron and carbon, as now worked out do not seem to agree with the figures given by Mayrhofer. The columns headed by an asterisk (*) show the relative percentages as we work them out to-day.

carried out by Jullien, Abel, Muller, Ledebur, Arnold, Campbell, Sterry Hunt, Hogg, Parry, and others. In a generic way, we now talk of carbides, sub-carbides, double carbides, special carbides, and other combinations. Cannot these be defined more accurately? Are there other new forms; if so, can these be separated and their characteristics obtained? It has been stated that the carbon compounds now definitely known number over 80,000. It would appear, therefore, that there is much room for valuable and useful research with the object of increasing our knowledge of the combinations of carbon with iron.

Special Alloy Steels.—Turning now to the question of research work carried out in connection with alloys of iron with other elements, in order to show the small amount of attention devoted to alloy steels by Crookes and Röhrig, this now so vastly important side of metallurgy was disposed of in only a few pages, under the heading of "The Influence of Foreign Mixtures on the Proportions of Steel."

In Kohn's book, which, as stated above, I have taken as my starting point, I can only find among its 270 pages one reference to a paper on the subject of the special alloy steels with which I am dealing, namely, the one by Thomas E. Vickers "On the Strength of Steel Containing Different Proportions of Carbon." This I have always looked upon as being one of the first papers that attempted to deal with the varying influence which a metal, or a metalloid, has upon iron. It was written in 1863 by Mr. Vickers, who, happily, is still with us. We are deeply indebted to him for his very important observations, as this paper was one of the forerunners in the development of this important branch of metallurgy, from which the world to-day is deriving so much benefit.

Effect of Carbon upon Iron.—In the matter of alloys of iron with other elements, the fact should not be lost sight of that carbon still pre-eminently holds a position of supreme importance, for the presence of carbon even in special alloys steel is, with some few exceptions, absolutely necessary. Take as an illustration high-speed tool steel, which has led to such an enormous saving in machine-shop practice, with consequent benefit as regards both quicker and greater output, combined with greater precision of execution, while the carbon present need not be relatively high, yet there must be sufficient carbon present, otherwise the cutting properties of the steel in question would not be attained. The explanation is probably as follows: In high-speed tool steel of modern composition there is usually about 16 to 20 per cent. of tungsten, 2 to 4 per cent. of chromium, and about 0.70 per cent. of carbon. Before treatment of the steel, the carbon exists in the form of carbide carbon and hardening carbon, as defined by Ledebur.

Hardening carbon	0.18 per cent.
Carbide carbon.....	0.56 per cent.
Total carbon.....	0.74 per cent.

After treatment, and with the nose of the tool in the condition required for mechanical work, we find the carbon content to have been transformed into

Hardening carbon.....	0.60 per cent.
Carbide carbon.....	0.14 per cent.
Total carbon.....	0.74 per cent.

Therefore, when we talk about the wonderful effect of the addition of elements other than carbon to iron, we must not fail to recognize that many, if not all, the new properties acquired by the steel are due entirely to the marvelous change that the treatment has wrought in the form of the carbon present; that is, its transformation from the soft, or carbide, carbon into what is believed to be the hardening, or hardenite, form of carbon; that is, the alloy elements which enable the hardening carbon—or call it what you will—to retain its hard cutting properties in the steel. While a similar change occurs by heat-treating any low-carbon steel—that is to say, carbon steel without other modifying elements present, or present in very small percentages only—the chromium and tungsten in high-speed alloy steel maintain, or assist in maintaining, the hardenite present in that form, so that the cutting edge does not lose its hardness even when quite a considerable degree of heat caused by frictional contact is reached, which would entirely soften ordinary carbon steel.

The Author's Experiments on Tool Steel.—These remarks of mine are based on experiments which I carried out as far back as 1903 with two grades of steel. The results of these experiments have not until now been published, so I should like to draw your attention specially to them, for the reason that they illustrate the very great importance of the metalloid carbon.

One of the alloys I made and experimented upon had the following composition:

C.	Si.	S.	P.	Mn.	Cr.	W.
0.74	0.10	0.04	0.03	0.28	2.84	18.00

The physical characteristics of this material (No. 73) were as shown in the accompanying Tables II, III, and IV. It was tested in two conditions: (a) annealed; (b) air hardened at white heat.

Heating and Cooling Curves.—The heating and cooling curves represented by Fig. 1 were taken upon steel No. 1,908D, of similar composition to that dealt with in the experiment described. The qualities of the two materials are practically the same.

TABLE II.—*Tensile Tests*

No. of Test Bar	Forging or Casting	Machined or Forged	Taken From	Transverse or Longitudinal	Pattern	Treatment	Permanent Set Lb. per Sq. In.	Breaking Load Lb. per Sq. In.	Elongation Per Cent.	Reduction in Area Per Cent.	Ball Test
6,517F	Forging	Machined	Bar 1 in. sq.	Longitudinal	No. 2	Annealed	24,640 Extensometer	116,480	14.5	18.64	255
5,563F	Forging	Machined	Bar 1 in. sq.	Longitudinal	Brayshaw 2 by 0.564 in.	Air hardened at white heat	104,160 Extensometer	155,000	nil	nil	627

TABLE III.—*Static Frémont Test*

No. of Test Bar	Forging or Casting	Machined or Forged	Taken From	Transverse or Longitudinal	Pattern	Treatment	Maximum Load Kg.	Energy Static Kgm.	Angle
6,699F	Forging	Machined	Bar 1 in. sq.	Longitudinal	10 by 6 by 30 mm.	Annealed	2,100	0.18	3°

TABLE IV

No. of Test Bar	Forging or Casting	Machined or Forged	Taken From	Transverse or Longitudinal	Pattern	Treatment	Elastic Limit				Ball Test
							25 PerCent. Set Tons	24 PerCent. Set Tons	5 PerCent. Set Tons	Total Set Per Cent.	
5,695F	Forging	Machined	Bar 1 in. sq.	Longitudinal	0.7 by 0.564 in.	Annealed	18	53	61	25.7	235
5,104F	Forging	Machined	Bar 1 in. sq.	Longitudinal	0.7 by 0.564 in.	Air hardened at white heat	135			0.143 Smashed at 284 tons per sq. in.	

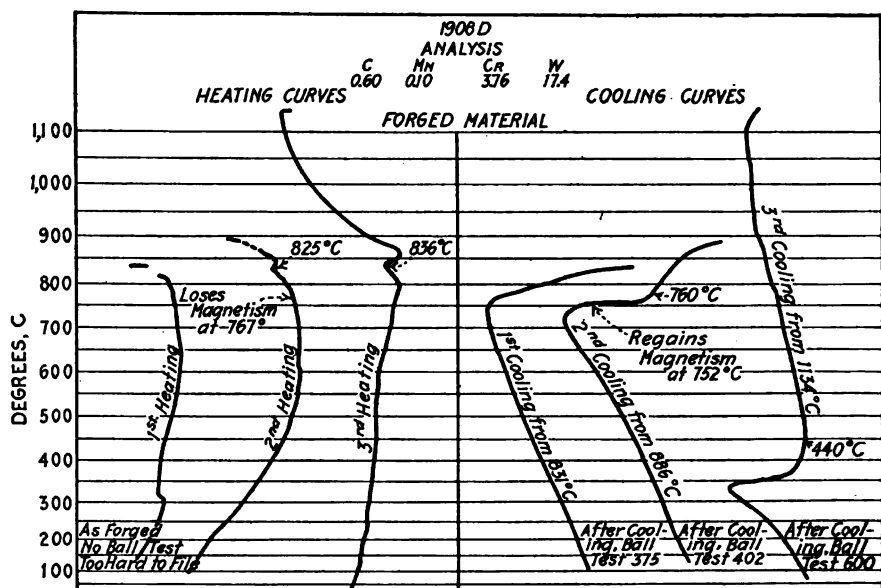


FIG. 1.—HEATING AND COOLING CURVES.
Taken with Saladin Differential Galvanometer.

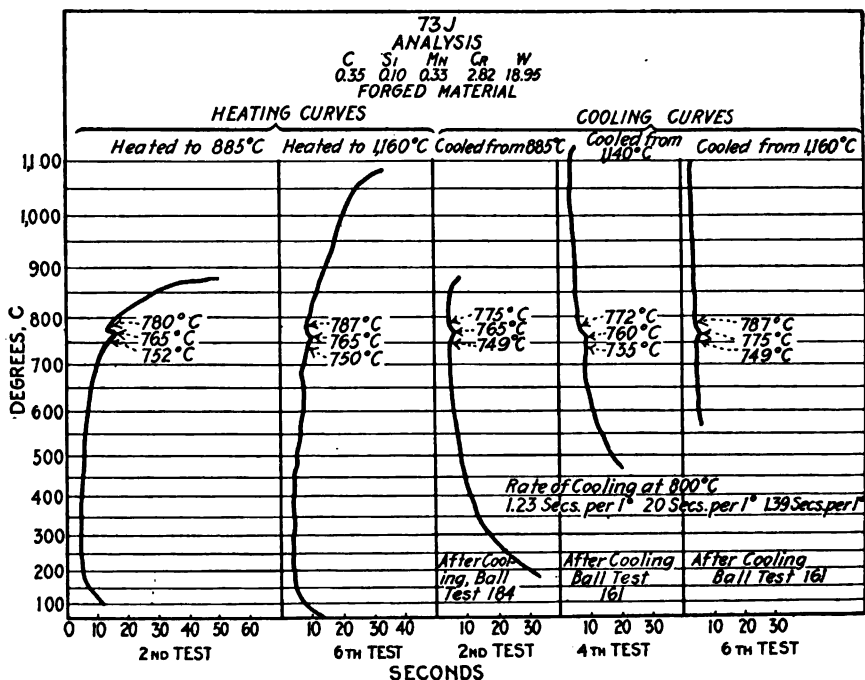
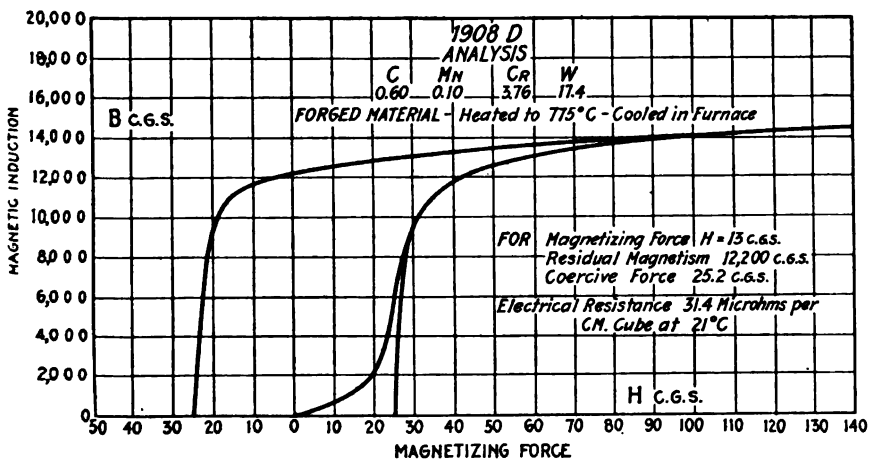
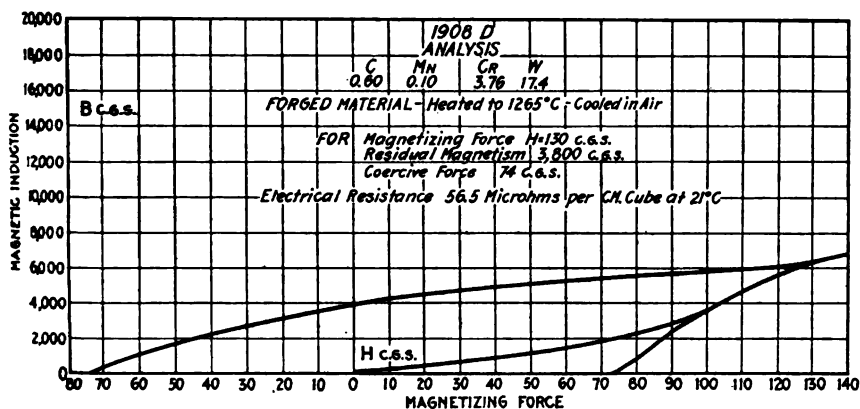
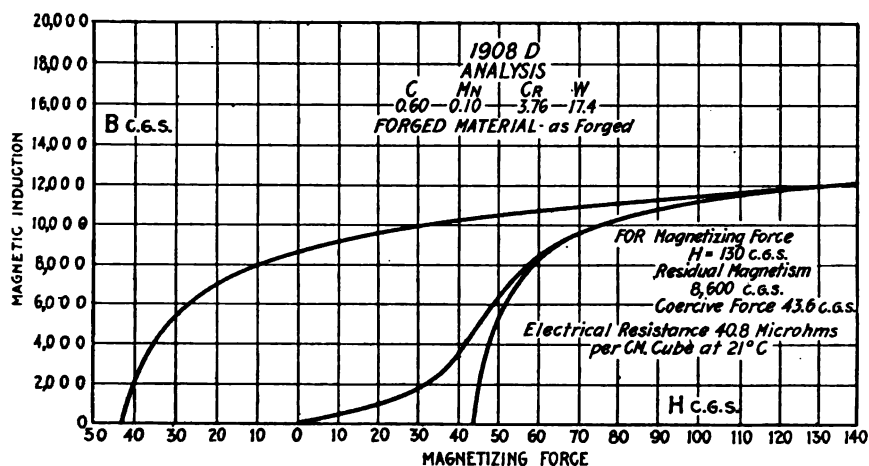


FIG. 2.—HEATING AND COOLING CURVES.
Taken by "Inverse Rate" method.



FIGS. 3 to 5.—TEST IN KOEPSER INSTRUMENT ON BAR 6 MM. IN DIAMETER.
 Curve uncorrected for demagnetizing effect of joints.

Fig. 2 represents heating and cooling curves of steel No. 73J.

Electrical Qualities.—As regards the electrical qualities of this material, these are as follows. The permeability and retentivity or coercivity were determined by the Electrical Department of the Hecla Works Laboratory.

Tests were carried out on the steel No. 1,908D, the following results being obtained:

TABLE V

Treatment	Electrical Resistance Microhms Per Cubic Centimeter
"As forged"	40.79
775° furnace	31.37
1,200° air	56.53

Magnetic tests on the same material are shown in Figs. 3, 4, and 5.

It will be seen from the heating and cooling diagrams that the magnetism probably disappears at a temperature of about 800° C. on heating and 750° to 735° C. on cooling; that is, in the Ar 1-2-3 point.

Tool Steel of Lower Carbon Percentage.—With a view to ascertain the effect of a lower percentage of carbon upon the alloy, the steel marked No. 73J was also prepared, having the following composition:

Mark	C.	Si.	S.	P.	Mn.	Cr.	W.
73J	0.35	0.10	0.03	0.03	0.33	2.82	18.95

It will be observed that this steel, containing 0.35 per cent. of carbon, was thus of the same composition as regards the "determining" elements; that is to say, those elements which are known as the special elements entering in the composition of speed tool steels, but the carbon percentage say about half. It was heat-treated in a manner similar to that carried out upon the specimen marked No. 73. It was, however, found to be quite useless for tools, being soft; it would not carry any cutting edge. There was practically no hardenite, or hardening carbon, present in this steel; the carbon present was almost entirely in the form of carbide carbon.

On the other hand, the steel No. 73, after heating to about 1,100° C. and cooling in air, was found to contain no less than 0.60 per cent. of hardenite, or hardening carbon, a circumstance which fully explains its valuable cutting properties. Table VI, which refers to this steel No. 73, will be found interesting.

It will be seen from the curves in Fig. 2 that this material cannot be properly self-hardened. This is very remarkable in view of the fact

TABLE VI

Mark	Before Treatment		After Treatment		Ball Hardness	
	Carbide Carbon	Hardening Carbon	Carbide Carbon	Hardening Carbon	Before Treatment	After Treatment
73	0.56	0.18	0.14	0.60	255	565/760

that it contains nearly 19 per cent. of tungsten and nearly 3 per cent. of chromium, that is, nearly 22 per cent. of special elements, leaving only 78 per cent. of iron. The reduction from about 0.70 per cent. carbon to 0.35 per cent. has made it impossible for the material to become hard.

It will be noticed that this steel No. 73J, containing 0.35 per cent. carbon, possesses in the forged condition a ball hardness of 202; after cooling from 885° C. its hardness is 184; after cooling from 1,140°, also 1,160° C., its hardness is 161. In other words, there is no increase in hardness.

We have thus in the above a very clear proof of the correct value and action of carbon. From this it will be seen that the same element, carbon, which is so important in carbon steels and in other special steels, is of equal importance in the modern high-speed tool steels. If you leave carbon out of the latter the material is practically useless for the purpose in view. There could not be a stronger evidence of the importance of this element carbon, acting as it does in the presence of no less than three other important constituents: namely, manganese, chromium, and tungsten.

As will have been noticed, the second specimen, steel No. 73J, contained high percentages of chromium and tungsten, nearly 22 per cent. of these two elements together (2.83 and 18.95 per cent. respectively), but the carbon content had been reduced to 0.35 per cent. This percentage of carbon in itself may be considered fairly high, and one might have expected that under the influence of the high percentages of chromium and tungsten present there would have been some hardening effect upon a tool made from a steel of such a composition. When cooled in an air blast from white heat (this treatment being the same as that carried out upon the specimen No. 73), also quenched in water from a bright red or a white heat, the tool made of the steel No. 73J in each case remained quite soft. This illustrates in a marvelous way, as I have said, the influence of carbon. In other words, the effect of this lower percentage of carbon, as compared with the proper percentage for high-speed tool steel, is much the same at this low figure as the effect that would be obtained were there no chromium or tungsten present. After quenching there was no hardening carbon or hardenite formed. On the other hand, by increasing the carbon to the not very high point of 0.74 per cent., (specimen No. 73), then the remarkable properties of high-speed tool steel were obtained.

The physical characteristics of the steel No. 73J were as follows:

TABLE VII.—*Tensile Tests*

No. of Test Bar	Forging or Casting	Machined or Forged	Taken From	Transverse or Longitudinal	Pattern	Treatment	Permanent Set Lb. per Sq. In.	Extension at Permanent Set Inches	Breaking Load Lb. per Sq. In.	Elongation Per Cent.	Reduction in Area Per Cent.
6,516F	Forging	Machined	Bar 1½ in. diameter	Longitudinal	No. 2	Air 780° F.	49,280	0.002	109,760	20	33½

TABLE VIII.—*Frémont Tests*

No. of Bar	Forging or Casting	Machined or Forged	Prepared from	Transverse or Longitudinal	Pattern	Treatment	Nick Test		Ball Test
							"C" nick	No nick	
7,725 F	Forging	Machined	Test bar 6,516 F. 1½ in. diameter	Longitudinal	10 by 8 by 30 mm.	Air 780° F.	2 kgm. by 3°	20 kgm. by 45°	
A5,195 A5,196	Forging	Machined	Bar 1½ in. sq.	Longitudinal	10 by 8 by 30 mm.	700° F.	2.5 kgm. by 2°	8 kgm. by 22°	217

This steel No. 73J showed an extremely fine crystalline fracture, about 20 per cent. being finely granular. The ball hardness of about 217 was obtained whether in its original condition (700° C. furnace) or in its heat-treated condition (heated to about 1,100° C., cooled in air, then reheated to 780° C. and cooled slowly in furnace).

I have specially devoted considerable attention to this particular steel because it illustrates in a clear and decisive manner the very great importance of the influence of carbon in determining hardness.

The work carried out by Abel, Muller, Arnold, and others with regard to the forms of carbon existing in steel created a great impression upon me, and it is largely their conclusions which have made me a carbonist rather than an allotropist.

The Great Work of Ledebur.—The late lamented Dr. A. Ledebur, of Freiberg, cast more light upon this dark place in metallurgy—namely, the relationship of iron and carbon—than any one with whom I have been acquainted. So important do I consider this work that I present in Table IX a list of his papers in case reference is desired to be made by any of those who feel interested in the subject, especially as some of

TABLE IX.—*List of Papers by Dr. A. Ledebur Relating to Iron and Carbon*

Title.	Source	Date
Some New Experiments and Theories on the Various Forms of Carbon observed in Iron and Steel.	Translated from <i>Stahl und Eisen</i>	June, 1886
On the Nomenclature of the Various Forms of Carbon occurring in Iron.	Translated from <i>Stahl und Eisen</i>	June, 1888
Sur la Denomination des Differentes Formes du Carbone dans les Fers Carbures.	<i>Revue de Métallurgie</i>	Sept., 1893
Nouvelles Recherches sur la Teneur en Carbone des Fers Carbures.	<i>Revue de Métallurgie</i>	Sept., 1893
On the Processes Enacted in the Hardening of Steel.	Translated from <i>Stahl und Eisen</i>	Oct., 1895
Various Forms of Carbon and Hardening of Steel.	Translated from <i>Stahl und Eisen</i>	June, 1897
Manual of the Metallurgy of Iron (4th Revised Edition, First Part)		
Annealing and Change in the Forms of Carbon by Heating.	Reprint from <i>Stahl und Eisen</i> No. 15.	1897
Cementation Process.		
On the Modifications of Carbon in Iron.	Iron and Steel Institute.	
On the Forms of Iron and Carbon present in Malleable Iron, Steel and White Pig Iron.		
Determination of Carbide Carbon.	Translation from <i>Guide Book for Steel Works Laboratories.</i>	

them are not well known. I believe that one of the present members of our Institute, Dr. Allerton S. Cushman, received his early metallurgical training under Ledebur. It would be of great interest if Dr. Cushman could be persuaded to give some reminiscences of this most remarkable German who did so much for metallurgy.

Allotropic Theory.—As you are all aware, bearing in view the papers quite recently read before the Iron and Steel Institute, it appears to me to have now been clearly proved that, as strongly indicated by my own earlier work many years ago, and by that of other experimentalists, including those to whom I have just referred, there is no such form of iron as an adamantine "beta" form. It is important that this should be made widely known, because, now the allotropic theory has been shown to be untenable, more attention will be devoted to other research; for example, in the direction of more fully determining the part played by carbon in that marvelous transition of the steel in its present extreme softness to great hardness; that is, from say about 100 Brinell ball number to 600, 700, or even 800, met with when we heat high-carbon or other suitable steel to a sufficient degree and water-quench or otherwise suddenly or quickly cool it.

The Author's Papers on Alloys of Iron.—The subject of steel alloys is such an important one in all its bearings that I thought it might be found interesting, and also useful for reference purposes, to trace back some of the work done in this connection as indicated by various research papers which I have had the pleasure of presenting to the metallurgical world from time to time during the last 25 years. These papers give in chronological order the history of this particular branch of the metallurgy of iron and steel. In these papers, too, I trust full reference and credit has been given to the splendid work done by the scientists of all nations who have helped on the good work, whether in England, America, France, Germany, Austria, Belgium, Sweden, or Spain. As I have before said, not a little credit should be allotted to the early work of the Swedish scientists for first separating for our use many of the elements which we are to-day using in our special steel. It is largely from their work we are to-day benefiting so greatly.

TABLE X.—*Table Showing in Chronological Order the Various Papers and Scientific Addresses by the Author of this Address, from the Year 1888 to the Present Time*

No.	Title	Name of Institute Where Research was read, or Press Reference	Year of Publica- tion
1.	Manganese Steel	Institution of Civil Engineers	1888
2.	Some Newly Discovered Properties of Iron and Manganese	Institution of Civil Engineers	1888
3.	On Manganese Steel	Iron and Steel Institute	1888

TABLE X.—*Continued*

No.	Title	Name of Institute Where Research was read, or Press References	Year of Publica- tion
4.	Alloys of Iron and Silicon	Iron and Steel Institute	1889
5.	Alloys of Iron and Silicon (Abstract of Paper)	British Association	1889
6.	Aluminium Steel	Iron and Steel Institute	1890
7.	Alloys of Iron and Chromium	Iron and Steel Institute	1892
8.	Notes on the Chicago Exhibition	Iron and Steel Institute	1893
9.	Iron Alloys, with special reference to Manganese Steel	American Institute of Mining Engineers	1893
10.	Huntsman, the Inventor of Crucible Steel	American Institute of Mining Engineers	1893
11.	Presidential Address	Sheffield Society of Engineers and Metallurgists	1893
12.	The Early History of Crucible Steel	Iron and Steel Institute	1894
13.	The Results of Heat Treatment on Manganese Steel and their Bearing upon Carbon Steel	Iron and Steel Institute	1894
14.	The Production of Iron by a New Process	Iron and Steel Institute	1895
15.	Steel and Iron Alloys	Institution of Civil Engineers	1897
16.	Foreign Technical Progress	Iron and Steel Institute	1897
17.	Alloys of Iron and Nickel	Institution of Civil Engineers	1899
18.	The Influence of Casting Tempera- ture upon Steel	Institution of Civil Engineers	1899
19.	Address as Master Cutler to the Cut- lers' Company	Sheffield Press	1899
20.	Address and Distribution of Prizes to Students at Technical Department of Sheffield University College	Sheffield Press	1899
21.	On the Electrical Conductivity and Magnetic Permeability of Various Alloys of Iron*	Royal Dublin Society	1900
22.	Researches on the Electrical Conduc- tivity and Magnetic Properties of upwards of One Hundred different Alloys of Iron*	Institution of Electrical Engi- neers	1902
23.	On the Magnetic Properties of an Extensive Series of Alloys of Iron*	Royal Dublin Society	1902
24.	Alloys of Iron, Manganese and Nickel	Institution of Civil Engineers	1903
25.	Alloys of Iron and Tungsten	Iron and Steel Institute	1903
26.	Iron and Steel Alloys	<i>Iron and Steel Metallurgist and Metallographist</i>	1904
27.	Researches on the Physical Properties of an Extensive Series of Alloys of Iron*	Royal Dublin Society	1904
28.	Magnetic Alloys from Non-Magnetic Metals	British Association	1904
29.	Production of Magnetic Alloys from Non-Magnetic Metals	<i>Chemical News</i>	1904

TABLE X.—*Continued*

No.	Title	Name of Institute Where Research was read, or Press Reference	Year of Publica- tion
30.	On the Effect of Liquid Air Temperatures on the Mechanical and other Properties of Iron and its Alloys*	Royal Society	1904
31.	Seventh Report of the Alloys Research Committee, On the Properties of a Series of Iron-Nickel-Manganese-Carbon Alloys*	Institution of Mechanical Engineers	1905
32.	On the Magnetic Qualities of some Alloys not containing Iron*	Royal Society	1905
33.	Presidential Address	Iron and Steel Institute	1905
34.	Experiments relating to the Effect on Mechanical and other Properties of Iron and its Alloys produced by Liquid Air Temperatures	Iron and Steel Institute	1905
35.	Effet de la Température de l'Air liquide sur les Propriétés mécaniques et autres du Fer et de ses Alliages	Congrès International des Mines, de la Métallurgie, de la Mécanique et de la Géologie Appliquée (Section de Métallurgie)	1905
36.	"James Forrest" Lecture, on Unsolved Problems in Metallurgy	Institution of Civil Engineers	1906
37.	Presidential Address	Iron and Steel Institute; Joint meeting with American Institute of Mining Engineers	1906
38.	Address to Students at Columbia University, New York	<i>Iron Age</i>	1907
39.	The Magnetic Properties of Iron and its Alloys in Intense Fields*	Institution of Electrical Engineers	1910
40.	A Research on the Hardening of Carbon and Low Tungsten Tool Steels*	Institution of Mechanical Engineers	1910
41.	"Expériences sur la Ségrégation dans les lingots d'Acier"	<i>Revue de Métallurgie</i>	1910
42.	"Les Progres de la Métallurgie"	<i>Le Génie Civil</i>	1910
43.	Address at Sheffield University on the Conferment of Honorary Degree of Doctor of Metallurgy (D. Met.)	Sheffield Press	1911
44.	Sinhalese Iron and Steel of Ancient Origin	Royal Society	1911
45.	Sinhalese Iron and Steel of Ancient Origin	Iron and Steel Institute	1911
46.	Abstract from paper on Sinhalese Iron and Steel of Ancient Origin	<i>Nature</i>	1912
47.	Abstract from paper on Sinhalese Iron and Steel of Ancient Origin, with Addendum regarding Delhi Pillar	Royal Society	1912

* Joint paper with others.

TABLE X.—*Continued*

No.	Title	Name of Institute Where Research was read, or Press Reference	Year of Publica- tion
48.	Analysis of the Iron Pillar, made about 300 A. D., at Delhi	<i>Engineering</i>	1912
49.	Contribution to the discussion (International) on The Magnetic Properties of Alloys	Faraday Society	1912
50.	The World's Progress in Metallurgy	<i>Iron Age</i>	1912
51.	The Production of Sound Steel	<i>Iron Age</i>	1912
52.	The World's Production of Pig Iron	<i>The Times</i>	1912
53.	On a New Method of Revealing Segregation in Steel Ingots	Iron and Steel Institute	1912
54.	Method of Producing Sound Ingots	Iron and Steel Institute	1912
55.	Plant for Hadfield Method of Producing Sound Steel Ingots	American Institute of Mining Engineers	1913
56.	Méthode pour Produire des lingots d'acier sains et Déceler la Ségrégation dans les lingots d'acier	<i>Revue de Métallurgie</i>	1913
57.	Nouvelle Méthode pour Déceler la Ségrégation dans les lingots d'acier	<i>Revue de Métallurgie</i>	1913
58.	Heating and Cooling Curves of Manganese Steel	Iron and Steel Institute	1913
59.	The Magnetic and Mechanical Properties of Manganese Steel*	Iron and Steel Institute	1914
60.	Research with regard to the Non-Magnetic and Magnetic Conditions of Manganese Steel	American Institute of Mining Engineers	1914
61.	Sound Ingots	American Institute of Mining Engineers	1914
62.	Manganese-Steel Rails	American Institute of Mining Engineers	1914

I give this Table X, and in one sense regret that it covers only my own personal study. During the early progress of this branch of metallurgy, and up to the time I commenced my researches, there were practically no papers of this kind on record. These therefore truly represent the direction in which the leading lines of progress in this branch have been made during the last 30 years. They give historically the gradual progress of this wonderful art of alloying various elements with the master-metal of the world, iron. Moreover, as no complete list of these papers has before been published, the one now presented may prove a useful bibliography to those interested.

Among other matters, these papers deal with manganese steel and its most remarkable properties; with alloys of iron and silicon; iron and aluminum; iron and tungsten; iron and nickel; iron and chromium; also with a large number of other binary, ternary, and quaternary steels.

* Joint paper with others.

This address does not attempt to consider and deal with more than certain portions of thought with regard to the progress of metallurgy. Otherwise it would require not one paper of a few pages, but many books, for so great has been the development on all sides of modern metallurgy. Our ancient friend before mentioned, Tubal Cain, or his confrère of the last century, would indeed be surprised with what he would now see in the advance of this branch of science.

In order to make the special subject of alloys of iron more clear, I may also add the following description in Table XI of some of the researches dealt with in my papers, from which will be seen the large number of alloys of iron and other elements which have been dealt with. The results obtained will be found in the various papers detailed in the foregoing Table X.

TABLE XI

Number	Binary Alloys	Number	Ternary Alloys
1	Carbon	16	Nickel-manganese
2	Boron	17	Nickel-chromium
3	Manganese	18	Nickel-tungsten
4	Nickel	19	Nickel-silicon
5	Copper	20	Nickel-aluminum
6	Cobalt	21	Nickel-molybdenum
7	Chromium	22	Quaternary Alloys
8	Tungsten		Carbon-nickel-manganese
9	Molybdenum		
10	Vanadium		
11	Silicon	23	Quinternary Alloys
12	Aluminum		Carbon-tungsten-chromium-vanadium
13	Sulphur		
14	Phosphorus		
15	Arsenic		

It must not be forgotten that although the above table of alloys refers chiefly to the influence of one or more special elements upon iron, yet the qualities of such alloys are largely affected by the presence of what may be termed the leading element, carbon, which is usually also present. Although in the case of high-speed tool steel the carbon percentage is not high, its influence is literally enormous, for, as I have shown, reduce the carbon below certain limits and you have a comparatively useless product.

Beyond a small number of exceptions—important ones, it is admitted; for example, the material known as “ingot iron,” “low-hysteresis steel,”

and some others—all the known “steels,” including even mild steel, largely depend for their properties and consequently for their commercial value upon the percentage of carbon present.

Thermal Treatment.—While there are still left elements to be alloyed with iron, and upon which there is ample field for investigation and research, we are, so to speak, nearing the top of the hill. It cannot be very long, unless new elements are discovered, which is not probable, before we shall be able to say that there is little or no more room for discovering new alloys or combinations, and attention will then have to be turned still more closely than even now as to the effect of thermal treatment. This may quite possibly imply treatment at lower grades of temperature than those which we now imply by the term “heat treatment,” which leads me to make the following remarks:

Low-Temperature Experiments in Great Britain and America.—The recent prosecution of research in the direction first dealt with by our great scientist Sir James Dewar, that is, the behavior of metals and their alloys at temperatures approaching absolute zero, shows that there is much to be learned from an examination of the physical properties and behavior of metals and their alloys at such temperatures.

In the joint research carried out by Sir James Dewar and myself, a full description of which will be found in my paper on Experiments relating to the Effect on Mechanical and other Properties of Iron and Its Alloys produced by Liquid Air Temperatures, read before the Iron and Steel Institute in September, 1905, it was shown that great increase of tensile strength in steel alloys could be obtained; in one case a tensile strength of over 150 tons per square inch was obtained, not merely from a wire but from a forged bar.

Recent experiments in America show that at a temperature of about -435°F. , or close upon absolute zero, metals lose their resistance to such a remarkable degree that the energy developed at the power house at Niagara, so it is stated, could be transmitted over a quite small metallic wire, simply because resistance vanishes at these temperatures. It will therefore be seen that there is good ground for my statement, that before long we may have to consider not merely heat treatment effects, that is, an increase over normal temperature, but also treatment below normal temperature.

Field for Research.—It will be seen that even to-day the field for research has been and still is an immense one; it is full of difficulties and perplexities. Contradictions often seem to present themselves. Nevertheless, the advance made in this branch of alloy steels since the date of my first researches in 1882 and onward, also my paper on Manganese Steel in 1888, has been of a startling nature.

While it is true there may not be at the present time room for such abnormal discoveries in ferrous metallurgy as in the past, yet workers all over the world are quietly and steadily adding to our stock of knowl-

edge on points, some of which may not seem so important at the time, but all of which tend to enable us to better understand and therefore control the desired qualities of iron and its alloys.

Melting Points.—Another example of attempts being made to widen our field of knowledge is the work of Dr. G. K. Burgess, of the Bureau of Standards, Washington, D. C., who is at present engaged with myself in a joint research regarding the melting points of the various iron alloys I have produced. We are already greatly indebted to Dr. Burgess for the work he has done in connection with high temperatures, melting points, and other kindred subjects. One of his latest researches has been to clear up once and for all the melting points of what he terms the "refractory elements," chiefly those of atomic weight from 48 to 59. We now have for the first time fully recorded the following information, which I have taken from his paper on Melting Points of the Refractory Elements—Elements of Atomic Weight from 48 to 59, detailed in the *Bulletin of the Bureau of Standards*, vol. x, reprint No. 205.

TABLE XII.—*Melting Points of Elements of Atomic Weight 48 to 59*

Metal	Melting Point with Micropyrometer	Purity	Probable Melting Point of Pure Element.
Nickel	^a 1,452 ^b 1,449	99.83	1,452 ± 3
Cobalt	1,477 ± 2 ^c 1,478 ± 1		
Iron	^d 1,533 ± 1 ^b 1,528	99.95	1,478 ± 5
Manganese	1,255	99.98 ± 0.01	1,530 ± 5
Chromium	1,520	97-98	1,260 ± 20
Vanadium	1,720	98-99	1,520 to Fe?
Titanium	1,794 + 12	97-98	1,720 ± 30
		99.9 +	1,795 ± 15

^a Assumed value on platinum strip; also observed value on iridium strip.

^b Crucible melts in electric furnace.

^c Crucible melts in electric furnace; also on iridium strip with micropyrometer.

^d Five samples all agreeing to within 3°.

Dr. Burgess and J. J. Crowe also read another interesting paper on The Critical Ranges A2 and A3 of Pure Iron, at the meeting of the American Institute of Mining Engineers held in October of last year, which will largely help us to settle and co-ordinate the many apparent inconsistencies in past research work on this subject.

Microscopy.—In an address of this nature, mention can only be made briefly of the important branch of metallurgical research known as micro-

scopy. At the time of Kohn's book very little was known. Sorby had practically originated this line, but there were few who had the prophetic eye to foresee the importance this branch of investigation would reach.

As Prof. W. C. Fearnside, M. A., F. G. S., pointed out in a recent and able lecture upon Dr. Sorby's work, he was without doubt the first to originate metallography, or the study of the microstructure of iron and steel, as well as other metals. Sorby's earliest research work was in 1849, when he prepared the first rock "slice" ever made, and his first microscopical study of "igneous" rocks was presented in his great paper, read before the Geological Society of London on Dec. 2, 1857. Showing the incredulity existing at the time as to the value of Sorby's work, Professor Fearnside says:

"Smiled at by Professor Tinnant when about to begin the research, jeered at by his friend Leonard Horner (Vice-President of the Geological Society), who, at the meeting to which the results were presented, remarked from the chair that 'he had been a member of the Geological Society ever since its foundation and during the whole of that time he did not remember any paper having been read which drew so largely on their credulity,' Sorby lived on to have those same results acknowledged and to be acclaimed by the geologists of all nations, assembled to celebrate the Centenary of the Foundation of the Geological Society of London, as the founder of modern petrology."

Sorby's first work on the microscopical study of metals was in 1863. In 1864 he lectured on this topic in Sheffield, later in the same year communicating his results to the Bath meeting of the British Association.

The writer of this address well remembers in 1886 taking some of his first specimens of manganese steel to Dr. Sorby, in order to ask his views with regard to the microstructure of this curious alloy of iron and manganese. Owing, however, to the comparatively imperfect methods of examination then prevailing, Dr. Sorby was not able to definitely report any special characteristics that would throw light upon this non-magnetic material with its many peculiar physical properties.

It was in 1885 that Dr. Sorby was first able to show the true composite nature of the "pearly constituent" of steel, as an aggregate of parallel plates. This discovery was the earliest recognition of the formation of crystals from a solid solution, and may be considered as the crowning achievement of his microscopical research. It was announced to the Iron and Steel Institute in 1886, and in 1897 in his paper on The Microscopical Structure of Iron and Steel. Soon afterward, these discoveries appeared to be the signal "for great activity in the metallographic field, which he so brilliantly had started to explore."

It is not always possible to show who is the real originator and discoverer of lines of new thought. In this case, however, the pioneer's honor in leading the way in the study of the microstructure of metals is due to Dr. Sorby, and to him alone.

Happily, the mantle of Sorby as regards microscopy fell upon young

and able shoulders in Sheffield, and Arnold, who has continued and greatly enlarged this important science, was for many years in close touch with the great master Sorby, of whom Sheffield is so proud.

Stead, Osmond, Sauveur, Heyn and many others have assisted the good work. Kohn in his book devoted a couple of columns to the consideration of "Steel under the Microscope," but Dr. Percy, to whom we all owe a deep debt of gratitude, in his classic work of more than a thousand pages, *The Metallurgy of Iron and Steel*, published a little earlier than Kohn's work (1864), never even referred to this subject of the study of the microstructures of iron and steel! When we consider the army of microscopists of to-day working throughout the world in this branch alone of metallurgical work and research, preparing daily thousands of photomicrographs with a view to studying the proper structure of steel products for the guidance of both producer and user, it can well be understood why we have advanced so much in our general knowledge of this subject.

Dr. Percy's Work.—The work of Dr. Percy just referred to was indeed a classical one; even now one can obtain from it valuable knowledge and aid. His careful and well-prepared descriptions of plant, processes, and research are still of high value, and must have been specially appreciated when so little knowledge existed on the subjects he dealt with. It has been men like Dr. Percy, who have spent their whole life in furthering the particular science with which they have identified themselves, who have conferred such large benefit upon the world. All honor to such men, whose only reward, the only one they ever really appreciated, was the satisfaction of knowing that they were following their calling to the best of their ability. Their labor was in the true sense a labor of love.

Dr. Percy devoted some consideration to the alloys of iron. His master mind, like that of Faraday 40 years earlier, saw some of the possibilities of these combinations of iron with various elements. Unfortunately, Dr. Percy's alloy experiments were most of them on such a small laboratory scale, the product generally weighing at most a few ounces, that the results he obtained were naturally more of academic than of practical value. The whole of his observations with regard to what is probably the most important of the iron alloys—namely, iron and manganese—are summed up in a few lines: "We have already," he says, "considered carburized alloys of iron and manganese, but the presence of carbon, it need hardly be remarked, may be expected to modify the properties of any alloy in a material degree. With regard to alloys of pure iron and pure manganese, I do not know whether anything satisfactory has been published; I have not met with anything of the kind."

This statement was indeed prophetic. Naturally the highly carburized alloys of iron and manganese to which Dr. Percy referred were

really spiegeleisen containing at the most 7 or 8 per cent. of manganese.

There were but few ferro alloys made before about 1865, and those which were produced contained high percentages of carbon, the presence of which completely altered the character of the product. This was the reason why the properties of that extraordinary alloy, manganese steel, were disguised and masked under the predominance of the carbon present in the manganiferous alloys formerly produced. Thus the production and manufacture of an alloy which has proved of such great advantage to mankind remained hidden for another 20 years. It was the study of this particular alloy, which I discovered about 30 years ago, which may be truly said to have been the first systematic study of alloys of iron with other elements. Mushet had made what was then termed self-hardening alloy steel, but its manufacture was shrouded in mystery and its application of comparatively small nature. Moreover, its qualities were rather in the nature of a superior cast iron than steel.

"The Art of Scientific Discovery."—In the foregoing, I have referred, but very briefly, to some of the research work carried out in one branch of science, ferrous metallurgy. This research question is one of the greatest importance, and covers an immense field. Gore has dealt generally with this in a most interesting and full manner in his remarkable book on *The Art of Scientific Discovery*, published some 30 years ago, which every one interested in research should read.

Gore pointed out that there are even among scientific investigators those who consider that methods of discovery are incommunicable. But, as he added, original scientific research is not an incommunicable subject; it is a natural process, and being such it must have laws according to which it operates. It is effected by means of our mental powers, and is therefore subject to the rules of mental action, and is communicable by ordinary natural methods. It is also being reduced as knowledge advances to rules of action, and will continue to be one of the noblest of all intellectual employments.

Why Research Work is so Valuable.—The value of research work is generally recognized. A large portion of the admirable Presidential Address delivered by Sir H. Frederick Donaldson, K. C. B., in April last, before the Institution of Mechanical Engineers, was devoted to the consideration of this subject. This Institution has proposed to start a movement to co-ordinate research work so as to prevent the considerable amount of overlapping now existing. While fully agreeing with this step, I believe that co-ordination in research work should not be made to interfere in any way with individual research. Many works dealing with metallurgy and engineering in all its branches have fully equipped laboratories for carrying our experiments in connection with their own manufacture, these experiments forming part, as it were, of their ordinary routine work, and they have always been found ready to make known

many of the important results at which they have arrived. A simple glance at the list of contents of the various journals of our different societies suffices to show that metallurgists and engineers in this country cannot be taxed with working in secret, and while agreeing in the main, as stated above, with the movement which Sir Frederick Donaldson would like to see initiated, individual effort, to my mind, should not and must not be in the least interfered with.

The Work of the National Physical Laboratory, and the Bureau of Standards.—In this connection I should like to add that we have in England available for various kinds of research work and experiments a most admirable institution, the National Physical Laboratory, whose able Director, Dr. R. T. Glazebrook, F. R. S., has done so much to advance scientific knowledge. In this country the National Bureau of Standards, presided over by Dr. S. W. Stratton, is accomplishing similarly valuable work. The work of these two institutions bids fair to standardize many discoveries. They have provided also a powerful aid to many individual experimentalists. To illustrate my point, it is sufficient to mention the investigations carried out in the Metallurgical Section and at the Froude tank at Teddington. It is no secret that if the experts of the English laboratory had sufficient funds placed at their disposal, their investigations would have a much wider scope, but they have been hampered for want of funds. As an instance of this, it may be mentioned that at the present time they have on hand more than a hundred different investigations waiting to be taken up when funds are available. The laboratory is supported by grants made by the government itself and various British scientific and technical institutions, who thus recognize its value; the establishment itself of the Laboratory was largely due to the initiative of the institutions in question; it is a great British national asset which, it would seem to me, the State should recognize to a greater degree than has been the case, and should grant more substantial financial support out of the public funds. Fortunately, in America you seem to have more public spirited policy, whether as regards help by the State or by private individuals, such as that rendered by Andrew Carnegie, John D. Rockefeller, and others.

Past Workers in Metallurgy.—In the foregoing I have alluded to the future exhaustion of our mineral resources and the necessity which is incumbent upon us for utilizing the metal, of which we have at best only the trusteeship to a maximum degree. My present address, while it deals in the main with researches and discoveries in the matter of iron alloys, would be incomplete were I not to mention the marvelous inventions of Bessemer, Thomas, and Gilchrist, whose work was so ably carried into effect in this country by the revered Holley and Fritz. These men in very truth may be veritably styled benefactors of humanity, for they have largely increased the modern convenience and comfort of the world by making it possible to produce commer-

cial steel at a comparatively low cost, therefore more cheaply, from what were considered inferior products before they put forward their processes of manufacture. On a par with these names are those of Siemens and Martin, who gave us the open-hearth process. The manifold advantages which the world has derived from the discoveries of these men would require more space than is at my disposal for their description. They are patent to all of us, and I am fully persuaded that future generations, while they may or may not give a passing remark to Tubal Cain as being the first metallurgist, will be sure to revere the memory of the above-named scientists, who by their researches accomplished so much, not only for the men of their own period, but also for their children's children. Future generations must surely, too, bear in mind those other scientists who have given to the world special steel alloys, remarkable, among their other characteristic features, by their wonderful toughness and tenacity, by their great durability and many other valuable qualities. They will further give also much more than a passing thought to that body of men of every nationality who are attacking the same problem from another side, the prevention of waste by corrosion, a matter to which our society has given much attention.

The mere mention of these names and subjects opens to our minds an immense vista of consideration, actual realizations and future possibilities. I have here but touched the fringe and will conclude with the following references and remarks.

The Scope of the University.—When His Majesty the King went to Manchester last July, Lord Morley as Chancellor presented to him an Address from the university of that city. To this the King made answer saying: "It was to the universities that the whole Empire looked for men equipped with scholarship and technical knowledge, and with will and energy to devote their gifts to the service of the community in every sphere of action." With this we must heartily agree, whether as regards the universities of America or the British Empire.

A Degree congregation was held on July 3 of last year at Sheffield University, under the Presidency of the Chancellor, the Duke of Norfolk, when H. A. L. Fisher, the new Vice-Chancellor, uttered the following pregnant words: "That University did not exist for the benefit of Sheffield; it was Sheffield's gift to the world. In such an important mining and metallurgical center, it was inevitable that these subjects—mining and metallurgy—should claim special attention, but the life of the mind brooked no territorial barriers, and discoveries made in the laboratories of that University would, no doubt, exercise a sensible influence in every quarter of the globe. They might be quite sure that no endowment lavished on research was ever lavished in vain, and that a wealthy community which devoted a part of its opulence to subsidizing clever men to think at large was discharging a manifest duty to the progress of the world."

Men such as Faraday and other scientists referred to in this address were, to quote Mr. Fisher's words, among those having "will and energy to devote their gifts to the service of the community," and who represented "clever men who thought at large."

The Faraday Society.—The society in England over which I have the honor to preside, by taking the name of Faraday had shown its earnest desire to continue in the footsteps of this great scientist. The object of the Faraday Society in England, and the American Electrochemical Society in America, is to promote the study of electro-chemistry, electro-metallurgy, chemical physics, metallography, and kindred subjects. These two societies also aim to be international in the character of their work.

The Objects of Our Institute.—The objects of the American Institute of Mining Engineers possess even a still wider scope. Their work and their workers have penetrated all lands, specially with regard to work done in metalliferous mining. Moreover, we Britishers do not forget the help rendered us by some of the members of your Institute during the troublous times, now happily past, in South Africa. We shall never forget that some of your members even laid down their lives for my country. A man can do no more.

The evidences quoted in this address of the work our society has done in the past, are a happy augury for its future increasing value, not only in this country, but internationally. In your Secretary, Bradley Stoughton, you have one thoroughly devoted to the cause of metallurgy, which we all have so much at heart.

The Franklin Institute.—Among other scientific societies in America, I should like to bear testimony to the excellent work also done by the Franklin Institute, whose able Secretary, Dr. R. B. Owens, has done much to bring together scientists of many types. This society is cosmopolitan in its awards, quite a number of which have reached, during the last few years, men of science who have done work of note in Great Britain and the European Continent. These awards have encouraged many of us, not only as I say in this country, but also in the Old World. It gave great pleasure to all of us to see that the Franklin Institute recently awarded to Prof. Albert Sauveur the Elliott Cresson Gold Medal, thus wisely recognizing the valuable work done by him during many years on behalf of scientific metallurgy.

The Author's Recent Papers to this Society.—Three special subjects have been dealt with in papers I have recently contributed to this Institute. I will not enlarge here upon the topics in question, except to say, the first relates to Research with regard to the Non-Magnetic and Magnetic Conditions of Manganese Steel. In the paper entitled Iron Alloys, with special reference to Manganese Steel, which I had the honor of reading before this Institute in 1893, it was shown that manganese

steel could be made magnetic. In the further paper just contributed it is shown that this non-magnetic product, almost as inert as lead or copper, can be made magnetic and to possess a specific magnetism of nearly 50 per cent., or about half that of pure iron.

The next paper, on Sound Ingots, describes the continuation of the researches which I have made with regard to the production of sound steel ingots, whether for use in the form of billets, rails, or other structural shapes.

The third paper, Manganese-Steel Rails, gives the history and progress of manganese-steel rails. In my paper read before this Institute in 1893, now no less than 21 years ago, I showed not only how it was possible to produce manganese steel in the rolled, forged, or pressed form, but gave particulars, including composition and mechanical tests, of manganese-steel railroad axles, tires, forgings of various types. In one case it was shown that a manganese-steel rolled tire and manganese-steel forged railroad axles, some of the tests on which were made in this country, stood drop tup tests in a remarkable manner, giving results far superior to ordinary steel. Although there was no difficulty in furnishing such articles to the required finished shapes, the user had not then been educated to see, as he now does, that for special steels showing great strength and durability higher prices should be paid.

Conclusion.—In conclusion, I feel that this address deals quite inadequately with the important subject which I have attempted to bring before your notice. If, however, it does nothing else than stimulate a desire for research in "*The Art of Scientific Discovery*," to use Gore's excellent term, then I shall indeed feel well repaid. Let us remember that "*Labor improbus omnia vincit.*" Thus as time goes on we shall acquire more and more knowledge in the great field of research. As the poet Prince rightly says:

Who are the great?

Those who have boldly ventured to explore
Unsounded seas, and lands unknown before—
Soared on the wings of science, wide and far,
Measured the sun, and weighed each distant star—
Pierced the dark depths of ocean and of earth,
And brought uncounted wonders into birth—
Repelled the pestilence, restrained the storm,
And given new beauty to the human form—
Wakened the voice of reason, and unfurled
The page of truthful knowledge to the world:
They who have toiled and studied for mankind—
Aroused the slumbering virtues of the mind—
Taught us a thousand blessings to create—

These are the nobly great.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

The Russian Oil Fields

BY A. ADIASSEVICH, ST. PETERSBURG, RUSSIA

(New York Meeting, February, 1914)

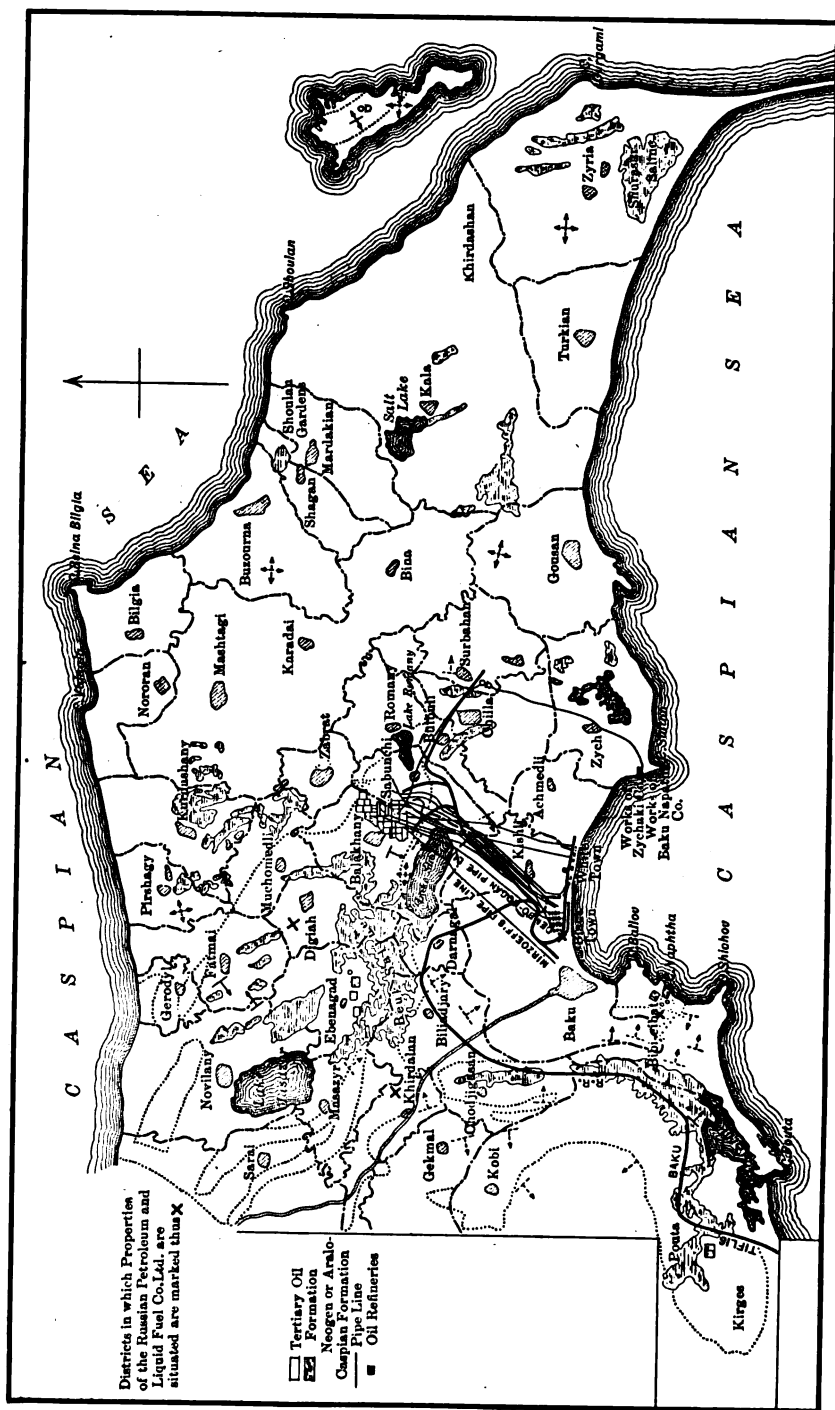
PETROLEUM has been found in Russia in various localities from north to south, as may be seen from the list given below:

- (a) In the basin of the Petchora river, Northern Russia.
- (b) In the basin of the Volga river, in the governments of Samara and Saratoff.
- (c) In the government of Astrakhan and in the Uralsk territory, close to the northern and eastern coast of the Caspian sea.
- (d) At Grozny in Northeastern Caucasus.
- (e) In the Daghestan territory, Eastern Caucasus, on the Caspian sea.
- (f) On the Apsheron peninsula (Baku and other districts) close to the Caspian sea shore.
- (g) On the island of Tcheleken in the Caspian sea, also on the island of Sviatoi.
- (h) In the Transcaspian territory on the Neftianaia Gora (Oil hill) and on the surrounding hills.
- (i) In the Ferghana valley in Russian Central Asia.
- (j) On the Kertch and Taman peninsulas on the Black sea.
- (k) In Siberia it has been discovered in the Transbaikal territory and on the island of Sakhalin.

In the above-mentioned districts petroleum has been found to exist under different geological conditions and to vary in its chemical and physical properties.

At the present time petroleum is produced in Russia at Baku, Grozny, on the island of Tcheleken, in the Ferghana district, and in the Uralsk territory. The yearly production in 1912 throughout the Russian Empire amounted to 69,400,000 barrels, of which the Baku fields were responsible for 52,000,000 barrels. This shows that the Baku fields on the Apsheron peninsula are the chief oil-producing centers in Russia, being at the same time the oldest of all our oil fields. In this paper I shall discuss the Baku fields only, leaving all the others until a later date.

The Apsheron peninsula in the Eastern Caucasus protrudes into the Caspian sea in the shape of a triangle some 47 miles long covering a total area of some 750 square miles. The eastern part of the peninsula is a flat, slightly rolling plain, rising in its highest parts to 140 to 170 ft. above



the level of the Caspian sea. The monotony of the country is relieved by rows of low hills or limestone ridges; on the clayey and sandy surface occur shelly limestone terraces. In the western part are scattered over the rising country elevated plateaus divided by deep ravines; separate ranges of rugged hills make their appearance and their summits rise to 840 to 1,200 ft. (Shaban-dag). In the lowlands accumulations of brine water form regular lakes; these shallow lakes dry up during the hot season

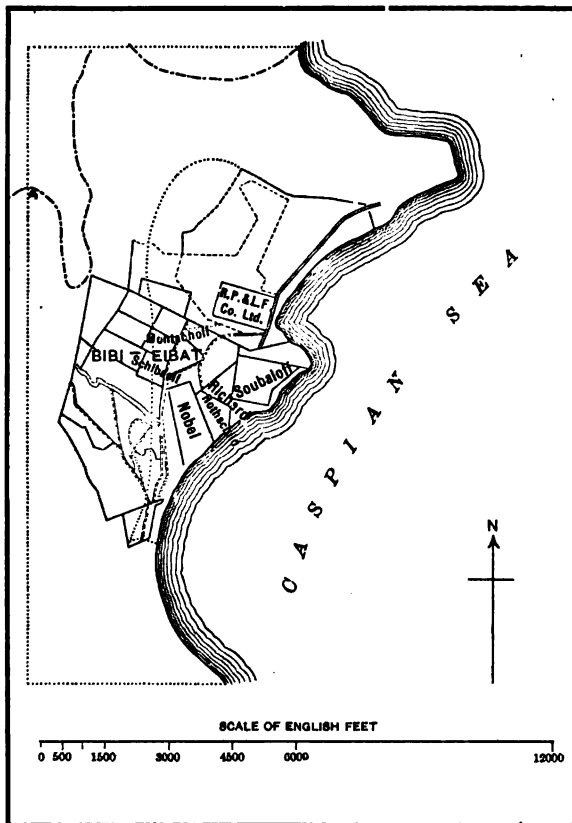


FIG. 2.—THE BIBI-EIBAT OIL FIELD.

leaving numerous salt marshes. The peninsula has the aspect of a sandy desert owing to lack of fresh water and partly also to sandy soil. In early spring, after the winter rain, the country is covered with low grass and bright flowers; even some meager crops are raised at the time; but by the end of May all vegetation is burned by the scorching rays of the southern sun. However, on the northern coast of the peninsula, where fresh water is more plentiful, there are cultivated fields, vineyards, and orchards.

The sedimentary rocks composing the Apsheron peninsula belong to the post-Tertiary and Tertiary periods.

The post-Tertiary is composed of loess, loam, blown sands, gravel formations, and silt, which spread throughout the peninsula. The marine deposits are Caspian terraces of friable shelly limestone, conglomerate, and gravel. These sedimentary deposits, some 140 ft. thick, are horizontal and lie unconformably on the underlying beds.

The Tertiary is represented by:

Pliocene deposits:

- (a) The Baku series, consisting of limestones, sands, clays, and conglomerates, deposited unconformably on the lower series. The series contains the following fossils: *Cardius crassum* Baeri; *Adacna plicata*; *Adacna edentula*; *Adacna vitrea*; *Monodacna caspia*; *Dreissensia Tschandæ* Andr.; *Dreissensia bugensis* Andr.; *Dreissensia polymorpha*; *Neretina luturata*; *Clessinia variabilis*; *Micromelania*.
- (b) The Apsheron series, consisting of limestones, oölitic limestones, marls, sands, sandstones, dark-colored clays with thin interlayers of volcanic ash and sand and deeper dark clays with interlayers of sands and black clays; among these strata are occasionally met hard sandstones and boulders. The following fossils are characteristic of these series: *Apsheronia propinqua*; *Apsheronia Euridesma* Andr.; *Apsheronia raricosta* Andr.; *Didacna intermedia*; *Monodacna nitida*; *Monodacna Sjogreni*; *Monodacna trapezinum*; *Dreissensia anisoconcha*; *Dreissensia rostriformis*; *Dreissensia latro*; *Limnæa obtusa*; *Cypris*.

The Pliocene is some 1,225 ft. thick.

Miocene deposits:

Black clays underlain by marls, light and dark clays with interlayers of volcanic sand, sands, sandstones, and limestones. These deposits are characterized by the following fossils: *Mactra carabugasica* Andr.; *Mactra subcaspia*; *Cardium Vog.* Andr.; *Cardium dombra* Andr.; *Cardium eduleformis*; *Potamides caspius*; fishes.

The formations are classified by Professor Andrussoff as belonging to the Akchagil series, their thickness being from 125 to 175 ft.

Under these series lie thick gray, blue, and brown clays, sometimes with sand, also thick marls interstratified with sands. The upper part of these deposits contains marine and lacustrine fauna (*Limnæus arma-nensis* Naulet., *Planorbis cornu*, *P. costatus*, *Glandina*, *Succinea*, *Paracypris*, the plant *Chara*), while in the lower part no fossils have ever been found. This is the oil measure, some 5,600 ft. thick.

This latter is underlain by sandy clays and at the bottom by char-

acteristic striated clays, in their turn underlain by layers of siliceous limestone containing numerous fossils of spiralis species, also *Lucina*, the fish *Leptolepis* and scales of *Meletta crenata*. Farther down follow shaly sandy clays with remnants of *Cedroxylon*, ferruginous sands, and dark chocolate clays containing fish remains (*Amphisyle*, *Lepidopus*, scales of *Meletta*). Then at last are found shaly clays belonging to the lower Tertiary, probably to the Oligocene.

This completes the succession of rocks comprising the Apsheron peninsula, as far as could be ascertained up to now by means of boring logs, samples of rock from drilled wells, natural outcrops, and the prospecting work carried on by the Geological Committee.

Both the Pliocene and Miocene strata are dislocated, leading to the formation of anticlinal and synclinal folds and dome-shaped uplifts, disturbed in their turn by a number of thrusts and faults in various directions.

The general trend of all uplifts on the peninsula corresponds to the direction of the main Caucasian ridge from northwest to southeast. But besides this upheaval there are others in various directions—viz., one in a west-east direction and another from northeast to southwest. These uplifts in various directions make the geotechnic of the peninsula very complicated.

As an example of typical uplifts in the shape of an elongated dome we can cite the Kirmaku-Balakhany-Sabunchy-Ramany and also the Bibi-Eibat folds. To the south of Ramany we meet the Surahkany fold and at a distance of 1 mile to the north of Bibi-Eibat lies the Atashka-Shabana anticlinal, of a north-south direction. Farther westward we notice a fold beginning from Mount Lock Botan and stretching beyond Mounts Kergez and Kizkala, where outcrop to the surface the strata composing the southern and western wings of this anticlinal. In the northern part of the peninsula are also to be found several distinct folds.

Following the line of main uplift, we notice a series of mud volcanoes, many of which, even at the present time, spout mud and water, gases, and petroleum. The natural oil issues scattered throughout the peninsula, mainly in its central and western parts, occur in the places which have been mostly subjected to the influence of the disturbing forces. Oil has been observed to emanate from layers of all the above described series from the Akchagal to the lower green clays. However, in commercial quantities oil is found only in several localities and its accumulations are dependent on some specially favorable geological conditions, such as are presented in this locality by the dome-shaped folds of the Balakhany-Sabunchy-Ramany and Bibi-Eibat fields.

The presence of petroleum on the Apsheron peninsula around Baku was known from very olden times and the local Persian population, before and after the Russians came to the country, used to dig shallow

wells or produce the oil from natural seepages, mostly near the village Balakhany, where the oil-bearing strata (since extensively worked) rise very close to the surface. In an eastern direction near the village of Surakhany the natural gas used to escape so abundantly from the fissures of the earth that a fire-worshippers' temple was erected on the spot, where the gas burned night and day in lofty flaring torches, and pilgrims flocked to the temple from Persia and even from distant India.

The first borings were started in 1870, in the central part of what was later known as the Balakhany fields, close to the hand-dug tests, and the first big gushers were obtained from depths averaging 120 to 350 ft., the oil spouting at the rate of some 30,000 barrels per day. After the first successful gushers were brought in the boring operations extended eastward and soon spread over the lands of the adjoining Persian villages of Sabunchy and Ramany, all three lying at a distance of some 10 miles inland in a northwest direction from Baku city (and port on the Caspian sea). These three fields have since borne the names of the three villages, although they have long since merged into one continuous field covering some 1978 acres. This is the territory from which comes the main part of the Russian oil. In an eastern direction from these lies the Surakhany field, which ever since the fire-worshippers' days was recognized as a very good gas producer, but where later vast amounts of light oil have been discovered and a good many big gushers were brought in.

Close to Baku city in an eastern direction, on the very shore of the Caspian, lies another small but very productive field—the so-called Bibi-Eibat field. Production was started here in 1880.

The Balakhany-Sabunchy-Ramany fields, and the Kirmaku field adjoining Balakhany on the west, which although worked up to the present only by hand-dug wells on the outcrops must be discussed as a whole with the others, are situated in the central section of the eastern end of the Apsheron peninsula on a slightly undulating tableland stretching from northwest to southeast. The country slopes here from northwest to southeast toward the former Ramany lake (salt). The central part seems to be slightly uplifted. The fields as a whole are surrounded by limestone ridges from north, east, and south, rising from 70 to 140 ft. above the surrounding country. The highest points within the fields territory are the mud volcano Bog-Boga (287 ft.) and the Kirmaku range, with a summit of the same name (330 ft.). This range follows a north-south direction. The strata dip here from west to east at an angle of 70°, forming an anticlinal, in the central part of which outcrop to the surface the lower series of the oil measure, composed of siliceous limestone and comprising spiralis specia. These siliceous formations are overlain by thick oil-bearing sands with thin interlayers of clays and sandstone of a total thickness of 900 ft. In its southern part the Kirmaku anticlinal bends eastward toward the Balakhany-Sabunchy-Ramany fields. On

these latter fields the strata composing the oil measure form a wide quaquaversal anticlinal, the strata dipping north, northeast, east, southeast, and south. In the central part of the oil fields the upper layers have been washed off and therefore the natural denudations or artificial shafts and shallow trenches reveal to our eyes all the consequent strata, beginning from Mount Kirmaku, in an eastern, northern, and southern direction.

The thick oil-bearing sands, which are outcropping near Mount Kirmaku, are the lowest pay sands in this part of the oil fields and these sands are worked on the slopes of the mountain by the local Persian villagers by means of shallow hand-dug wells. These sands are covered by some 100 ft. of clays, overlain in their turn by some 300 ft. of sands and sandstone comprising thin streaks of pay sands. On the top lie again clays about 200 ft. thick. These are followed by thick water-bearing sand and sandstone over 700 ft. thick and again some 400 ft. of clays, sand, and sandstone. Farther eastward, a series of oil sands interstratified with clays and barren sands outcrop to the surface, the total thickness being some 600 ft.; here we find as many as 15 separate and distinct pay-sand layers. These sands are worked on the outcrops by the villagers by means of shallow hand-dug wells. Still farther eastward sands interstratified with sandstone outcrop; this water-bearing formation is 700 ft. thick. This thickness is overlain by a series of pay sands with clay interlayers, barren sands, water-bearing sands, and sandstones, of a total thickness of 1,300 ft.; here we find some 22 separate pay-sand layers. Over it we meet a series of clays and sands barren of oil, of a total thickness of some 600 ft., and above the characteristic Akchagil layers, composed of dark clays with interlayers of volcanic ash and of marls. The Akchagil is overlain by clays, marls, sands, sandstones, and limestones belonging to the Apsheron. The middle Apsheron limestones, as was stated above, surround the producing fields, but are washed off in the central part. The Akchagil series rise to the surface from under the Apsheron layers in the northern part of the fields, turn southward, and are sporadically met with in the southern section of the fields. The clays and sands overlying the pay-sand beds thin out westward and within the limits of the Balakhany field disappear altogether, and the upper oil sands are met here quite close to the surface.

It is evident from the above description that within the boundaries of the Balakhany-Sabunchy-Ramany oil fields we have three divisions of oil-bearing sands; the first and topmost having a thickness of some 1,225 ft. together with the interlying clays, while the thickness of the pay sands alone reaches some 600 ft., whereas separate pay-sand beds vary from several feet to 70 ft. This first division, as shown above, is divided from the second by thick water-bearing sands and sandstones. The second oil division out of a total thickness of some 600 ft. comprises

280 ft. of oil sands, and at last we come to the lowest and thickest Kirmaku oil division, which contains hardly any barren interlayers at all.

Of these three oil divisions only the first has been extensively worked, and all the oil which has been obtained from these fields during the 40 years since the first boring wells were sunk here has been produced from these 600 ft. of oil sand. As stated before, the first gushers on the Balakhany field were brought in from a depth of 120 to 350 ft. At the present time the average depth of producing wells at Balakhany is over 900 ft. (minimum, 300 ft.; maximum, 1,800 ft.) and the production has been reduced from gushers to an average of 8,000 barrels per well per year, or some 25 barrels per day.

On the Sabunchy field the average depth of producing wells at the present time is about 1,200 ft. We know from the description and the sketch given above that the oil-bearing layers on this field are to be found under a greater thickness of overlying barren beds, and the depth of wells varies from about 700 to 2,300 ft. We shall further see that on the Ramany field, which lies still further along the dip of the strata, the wells have to go down to still greater depths. The production on the Sabunchy field averages 55 barrels per well per day, or some 17,000 barrels per year. A considerable drop of production has been noticed here during the last few years; 10 years ago the average production stood at 45,000 barrels per well per year.

On the Ramany field the average depth of producing wells is 1,600 ft. (minimum, 1,000; maximum, 2,450 ft.). The average production per well per day is 110 barrels, or per year some 26,500 barrels, whereas some 10 years ago the yearly production per well was 80,000 barrels.

Below I give a short summary of the Baku production, also the number of producing wells, average production per well, and the number of gushers. In these statistics I include the Bibi-Eibat fields, which will be discussed on the following pages:

Besides the above mentioned number of producing wells in 1913, about 500 wells were being drilled on all the four fields, about 400 wells were being deepened to the next producing stratum, and in about 1,800 wells work had been temporarily stopped. This brings the number of wells on all four fields to 5,590.

All this enormous amount, 1,113,700,000 barrels of oil on the three fields, Balakhany-Sabunchy-Ramany, has been drained from the 600 ft. thick sands of the first oil division. The second oil division has been only lately reached by a few wells on the Balakhany fields, but these wells have only served as tests and no oil has been extracted from those still virgin beds; the wells will have to reach a depth of 3,500 ft. and deeper. After this division is exhausted we have still the deepest, or Kirmaku, oil division. Thus we see that, although the production of oil from the first oil division threatens to drop from year to year, we

Baku Statistics

Years	Balakhany		Sabunchy		Ramany		Bibi-Eibat		Total		Production Per Well	Number of Gushers	Spouted Oil Per Cent.
	Wells	Production Barrels	Wells	Production Barrels	Wells	Production Barrels	Wells	Production Barrels	Wells	Production Barrels			
1899	610	14,350,000	543	28,840,000	138	12,350,000	58	10,100,000	1,349	65,640,000	48,500	23	15.3
1900	736	15,600,000	665	31,450,000	185	14,350,000	112	13,630,000	1,698	75,030,000	44,000	25	11.3
1901	775	14,700,000	780	36,920,000	213	15,520,000	143	16,690,000	1,911	83,830,000	43,500	35	14.6
1902	720	12,680,000	751	33,390,000	219	17,490,000	135	15,930,000	1,825	79,490,000	43,000	42	14.9
1903	693	11,080,000	747	28,800,000	221	14,990,000	174	19,690,000	1,835	74,560,000	40,000	33	8.9
1904	732	10,250,000	791	27,260,000	253	16,680,000	222	22,640,000	1,998	76,830,000	38,000	26	5.8
1905	750	7,040,000	730	17,400,000	245	10,900,000	248	15,830,000	1,973	51,170,000	25,000	26	3.3
1906	739	8,500,000	712	19,600,000	253	11,920,000	250	15,950,000	1,954	55,970,000	28,000	20	2.4
1907	836	8,910,000	873	23,000,000	278	11,200,000	305	16,400,000	2,292	59,510,000	26,000	28	2.5
1908	879	8,790,000	981	24,860,000	279	9,780,000	356	14,950,000	2,495	58,380,000	23,400	18	2.0
1909	917	9,110,000	1,059	25,880,000	285	10,960,000	371	15,350,000	2,632	61,300,000	23,300	19	4.2
1910	962	8,530,000	1,177	24,390,000	305	12,010,000	396	14,860,000	2,840	59,790,000	21,000	19	3.2
1911	937	7,860,000	1,235	22,000,000	330	10,350,000	390	12,810,000	2,892	53,020,000	18,300	13	2.4
1912	1,008	8,160,000	1,358	21,300,000	373	9,850,000	403	13,070,000	3,142	62,380,000	16,700		3.1

The total production of the Balakhany-Sabunchy-Ramany fields since the beginning of their exploitation amounts to:

	Barrels
Balakhany field, area 857.5 acres, since 1870.....	279,100,000
Sabunchy field, area 877.5 acres, since 1878.....	594,100,000
Ramany field, area 243.0 acres, since 1890.....	240,500,000

Total	1,978 acres.....	1,113,700,000
The total production of the Bibi-Eibat fields, area 520 acres, since 1880		268,500,000

have at Baku some deeper oil horizons which are practically virgin ground. We shall further see that on the Bibi-Eibat field there are also unexhausted deeper horizons, while all around Baku on the coast of the Caspian and further inland prospecting work reveals new oil-bearing territories.

The Bibi-Eibat field lies in a valley of the same name stretching from Cape Bailoff to Cape Shikhoff at a distance of a couple of miles from Baku city. The valley extends in a narrow belt for a mile along the sea coast; on the north and west it is bordered by lofty cliffs in the shape of a horse-shoe. The width of the section under actual exploitation is about 5,000 ft., its area being about 520 acres, out of the 800 acres comprised in the valley limits. The general incline of the valley is from northeast to southwest.

The rocks forming the Bibi-Eibat valley, as elsewhere on the Apsheron peninsula, belong to the post-Tertiary and Tertiary formations. The surrounding cliffs are shelly limestone, marl, clay, sandstone, and sand of the Apsheron series.

On the surface we find horizontal sheets of recent Caspian sediments overlying the lower beds of the Apsheron series. These latter reach in different parts of the valley a thickness of 500 ft. They outcrop in the central part, where they are washed off to a certain extent. The Apsheron is underlain by the Akchagil, of a total thickness of some 175 ft.; this is supposed to stratify unconformably with the overlying Apsheron beds. The Akchagil is underlain by a number of interstratified sands, sandy clays, clays, and marls, composing the oil measure, which has been exhaustively prospected and worked by numerous wells to a depth of 2,500 ft., while several wells have gone below 2,800 ft. Most of the sand layers are saturated with oil and gas, the distinguishing feature being that on the Bibi-Eibat field the series overlying the oil measure and even the Akchagil beds, which are always barren on the other group of fields, are mostly impregnated with oil.

Down to a depth of about 2,900 ft. we meet over 30 separate oil-bearing sand layers, from 7 to 90 ft. thick, with a total thickness of about 600 ft. In the upper horizons the pay sands are divided by impervious clay interlayers. The number and thickness of these clay layers decrease with the depth and the oil horizons are isolated in a far inferior way from the circulating water. Below the 2,900 ft. prospecting has revealed the presence of a number of oil-bearing sand strata. They outcrop in the Jassamal valley about 1 mile west of Bibi-Eibat and are actually worked by shallow hand-dug wells on the slopes of Mount Shaban-dag.

Long observations have established the fact that at Bibi-Eibat in its eastern part the strata dip north-northeast, east, and southeast, and in the western part, southwest and south-southwest. This indicates that

these strata had been uplifted into an elongated dome with its axis running in a north-northwest and south-southeast direction. The angle of dip is 7° to 22° . The strata are further fractured by faults chiefly of a northwest-southeast direction, but also running from northeast to southwest. The origin of these faults is the subject of some controversy. Some authorities maintain that the formation of the fold is due to the plicative forces; i.e., that the strata were uplifted into a dome-shaped anticlinal and the faults are of subsequent formation, especially as they do not reach any considerable depth. Other geologists are of opinion that the Bibi-Eibat uplift is the result of upward faulting (horsts) which played a prominent part in the formation of the field, and the oil deposits of the valley, as well as the productiveness of the various sections, depend on the presence of such faults.

As stated before, the Bibi-Eibat field has been worked since 1880 and 268,500,000 barrels of oil have been drained from that field. The average depth of wells on this field at the present time is over 2,000 ft. (minimum, 1,400; maximum, 2,900 ft.), while the production per well per day averages 110 barrels or per year about 33,000 barrels, having dropped during the last 10 years from 118,000 barrels.

On the Bibi-Eibat field some of the wells show up to now a considerable gas pressure, and in sinking wells to horizons exceeding 2,000 ft. big gushers are frequently brought in. We have reason to believe that the deeper strata, below 2,900 ft., contain virgin oil reserves.

As far as the petrography of the oil fields of the Baku district is concerned, it was found that the clays overlying the producing series as well as those interstratified with the sand layers are of various colors—black, gray, blue, different shades of brown; they merge into one another and in neighboring wells differently colored clays are found at corresponding depths. The clays are pure or with an admixture of sand. Sometimes gravel or boulders are imbedded in clays. Generally the clays contain salt and lime.

It has been observed that at Bibi-Eibat as well as on the Balakhany-Sabunchy-Ramany fields beds of clay occur extensively in the upper horizons and often reach a considerable thickness, while in the lower horizons sands predominate.

The clay layers play an important part at the oil fields, as they are the impervious partitions between the oil-bearing and water-bearing layers and the water is usually shut off in these clay beds.

The sand layers interstratified with clays are recorded in boring logs at Baku as water, gas, or oil bearing sands. The first is usually grit which is frequently water bearing. The second is a fine sand with a large admixture of clay. The third, or oil sand, is a fine-grained sand impregnated with oil. If such a sand is barren of oil it is recorded as oil sand or washed oil sand.

Oil sands from some of the wells were subjected to analysis and it was found that all of them contain quartz and mica and most contain carbonate of lime. Traces of copper and iron pyrites were discovered in some. In one case a feldspathic mineral was observed, in another rutile, in a third case a single crystal with the character of beryl.

Reference has been made above to the water-bearing sands and sandstones, dividing the first and second oil divisions. But unfortunately this is not the only water horizon at the Baku fields. Water-bearing layers are met throughout the thickness of strata, but mostly in the lower producing horizons. As a rule, especially in the lower horizons, all barren sands, or those from which oil was drained, are saturated with water, which seems to take the place of oil in such sands. These water-bearing sands are usually incased between beds of clay and should thus be well isolated, except for the numerous faults, which cut through the fields in all directions and serve as a connection between the water and oil sand layers.

On the Balakhany-Sabunchy-Ramany fields water-bearing sands occur, beginning from a very shallow depth of some 70 to 100 ft. and continuing throughout the whole of the first division, while at Bibi-Eibat down to 1,400 ft. there are no well-defined water layers; abundant water outflows occur in lower horizons, especially at 2,100 and 2,650 ft.

The water-bearing sands vary not only in different parts of the fields, but even on the same plot and in the same well, in consequence of varying admixture of clay, the proximity of faults, the character of the overlying and underlying strata, etc.

The waters circulating in the strata of the Baku oil fields may be subdivided into surface waters and deep-layer waters, which in their turn may be subdivided into the waters of upper and lower horizons. All the waters are saline, with varied contents of solid matter. The surface waters usually contain little mineral matter.

The waters circulating in the producing strata are more saline than the surface waters and differ from them in the chemical composition of the solid matter. On the whole, the waters are less saline in the upper horizons, the contents of salt increase with depth, and then again rise less saline waters, which often serve as an indication of the rise of deep water into upper horizons. At Bibi-Eibat to a depth of 2,500 ft. the waters vary from 11° to 9° Bé., while below the density drops to 7.5° and 6° Bé. All the waters analyzed contain alkalies (potassium, sodium, magnesium, and calcium) and also chlorine and carbonic acid. No sulphuric acid has been discovered in the waters from the wells of the Balakhany-Sabunchy-Ramany fields, whereas over 50 per cent. of the Bibi-Eibat waters analyzed contained traces or larger amounts of sulphuric acid.

These and similar observations are of the highest practical importance,

as very often they serve as guides in tracing the origin of the water which floods the wells in one or another part of the fields. Among other very important identifying data we must mention the temperature at the bottom of the various wells. On the Balakhany-Sabunchy-Ramany field such temperature varies between 19.8° and 43.4° C. and on the Bibi-Eibat fields between 34.2° and 52.2° C.

Statistics indicate that the oil-bearing strata which are being worked at the present day on the Balakhany-Sabunchy-Ramany and Bibi-Eibat fields are approaching exhaustion. Some of the layers, which used to be oil bearing and very productive, yield very little oil and large amounts of water. The gradual increase in the depth of producing wells points to the same fact. At the same time we see that the upper pay sands are drained to a larger extent than the lower beds.

As to the lower horizons of the Baku oil measure—*i.e.*, the second and the Kirmaku divisions—these may be considered to be perfectly virgin.

At Bibi-Eibat the lower horizons of the oil measure are also very rich in oil, as the seven or eight wells carried down below 2,800 ft. brought up oil in gushers.

The chief difficulty in drilling to the deeper pay sand is the presence on both groups of fields of thick water-bearing sands dividing the upper pay sands from the lower virgin strata; besides this, the thickness of clay layers in the lower horizons is considerably less than in the upper horizons and therefore with the increase of depth the isolation of water from the pay sands becomes more and more difficult. The third obstacle lies in the depth at which these lower pay sands occur. Over the greater part of the area under actual exploitation the depth at which the lower pay sands may be worked is from 2,800 to 4,200 ft. It is evident that drilling to such depths, considering the presence of thick water-bearing sands, is far from being an easy matter. But no doubt the improvements which are continually introduced in drilling methods and in the cementation of wells will enable the Baku producers to exploit these deep-lying horizons, probably in the very near future.

In conclusion, I must draw attention to the fact, that the Baku oil field is one of the most remarkable fields on the face of the earth. We see here a number of separate pay-sand layers following one another, all these oil-bearing beds being of considerable capacity. Up to the present time, from an area of 2,500 acres these oil fields have produced about 1,400,000,000 barrels, or 560,000 barrels per acre, and still production continues at the rate of about 52,000,000 barrels per year. And of these 2,500 acres there are still plots of Crown lands which have been left intact.

This enormous accumulation of oil on such a comparatively small area is due to the very high holding capacity of the sands, and also to a very favorable geological formation of the locality—namely, the fact that

the sand layers form a quaquaversal anticlinal, with gently dipping strata on the wings.

Besides the Balakhany-Sabunchy-Ramany and Bibi-Eibat fields, there are other fields on the Apsheron peninsula which are likely to prove very good producing territory. Thus I have already mentioned the Surakhany field, where work was started a few years ago. At the present time this field produces yearly about 4,000,000 barrels of oil, nearly every well sunk to 1,400 to 2,100 ft. proving to be a gusher. The area of this field is not yet defined, but we have abundant data to show that this field is also a dome-shaped anticlinal with its axis running in a north-south direction.

A few miles further east there is another similar dome-shaped uplift where prospecting work is carried on at present.

In the western part of the Apsheron peninsula oil has been found in several localities and prospecting work is carried on. This will show what we have to expect from the western section.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

The Origin of Petroleum

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APART from the hypothesis of a cosmic origin (which failed of acceptance because it was not adequately supported by facts), the only important controversy concerning the origin of petroleum has been, for a long time, between the advocates of inorganic and of organic origin respectively. Each of these theories has had a long history of development, and is still being perfected, under the influence of two causes: (1) the increasingly extensive and thorough study of the oil-fields (of which new examples are periodically discovered and opened); and (2) the progress of synthetic experiments devoted to this question. Moreover, the advance in our physical and chemical knowledge of the properties of this peculiar natural product has necessarily modified all criticism of conflicting views.

I. THE HYPOTHESIS OF INORGANIC ORIGIN

That the notion of an inorganic origin of petroleum, first set forth by Berthelot in 1866, and afterward ingeniously developed and formulated by Mendelejeff, should thus have proceeded chiefly from chemists, is quite natural; for the question was one of *possible* chemical processes in the earth's interior, and of imagined chemical reactions to be verified by experiment. Hypotheses of this kind were suggested by many chemists, of whom two, P. Sabatier and J. H. Senderens, may be specially named by reason of their highly interesting chemical experiments.

Among geologists, Mendelejeff's hypothesis was received at first with much interest and favor; for it rested on the assumption of a central terrestrial mass of iron carbide, and the geologists had good reasons for adopting that assumption. Yet comparatively few of them attempted to furnish geological proofs of the hypothesis: the majority either silently believed in it, or for one or another reason rejected it altogether.

An apparently weighty support of Mendelejeff's view was furnished by the American, G. F. Becker, who found in the oil-regions of the United States important and abnormal disturbances of the isogons of terrestrial magnetism, and inferred that in these regions the central iron mass must

come nearer to the surface than elsewhere. To my mind, the better explanation is, that in these oil-fields great quantities of magnetic iron have been placed. All iron pipes, especially when set vertically and hammered, notoriously become magnetic; and this is the case with the tubings of the oil-wells—sometimes to such a degree that iron screws, lowered by ropes into the bore-holes, are so strongly attracted by the iron linings that they stick fast, and will not descend further. Moreover, the disturbances of the isogons are not uniform—a circumstance easily explained by the varying amount of iron tubes. Finally, violent irregularity of these curves is shown in places where deposits, either of oil or of magnetite, are not known.

Mr. Becker's conclusions were disputed by W. A. Tarr for other reasons.¹

The most zealous advocate in America of the inorganic origin of petroleum, so far as I know, is Eugene Coste, of Toronto, who has collected with praiseworthy industry all the facts which support this hypothesis. To give to his many arguments the serious, thorough, and critical examination which they deserve, is the principal purpose of the present paper, as will be seen further on.

During recent decades, no European geologist of authority has advocated the inorganic origin of the petroleum occurring in large deposits.

II. THE HYPOTHESIS OF ORGANIC ORIGIN

As regards organic origin, the view may first be mentioned, that petroleum is the product of distillation from coal—from which, in fact, artificial distillation had obtained photogene and other products having considerable physical resemblance to kerosene. This view was first expressed in Europe by F. von Beroldingen (1778). Von Kobell, Anstedt, Leon Malo, Romanowsky, Noeggerath, Huguenet, v. Hochstetter and others accepted the hypothesis; but the proofs adduced by v. Beroldingen were soon recognized as inadequate. Since petroleum often occurs not underlain by coal-beds—occurrences of coal and oil being, in fact, usually (as on the northern border of the Carpathians) mutually exclusive;—and since, moreover, the distillation-products of coal are entirely different from petroleum chemically (as Baron Reichenbach proved, as early as 1833), this hypothesis had to be abandoned. So far as I know, it received no support in America, because in Pennsylvania, the mother-land of the oil-industry, there were no coal-beds under the deposits of oil.

Animal Origin

Hypotheses of an organic origin were thus narrowed to the direct transformation of animals or plants to petroleum. Already in 1794, Haquet suspected that the oil of Galicia came from marine mussels, "dis-

¹ *Economic Geology*, vol. vii, No. 7, p. 647 (Oct.-Nov., 1912).

solved" in salt water. L. v. Buch, Quenstedt, Volger, Naumann, Dufrenoy, Posepny, Verbeck, Fennema, and many other eminent geologists, accepted this hypothesis, mostly in view of the circumstance that the bituminous rocks carry the fossil remains of animals. Bertils found in the Kuban district of southern Russia, in a bunch of mussel-shells, a substance partly "petroleous," partly animal remains still undecomposed. R. A. Townsend reported a similar observation from the Tertiary oyster-banks of Assam, in Asia. Ch. Knab somewhat extended the hypothesis; and in North America it found in Whitney, for the petroleum of California, and the brilliant Sterry Hunt, for the oil-deposits in the ancient limestones of Canada, Pennsylvania, and Ohio, its most influential advocates, with whom others allied themselves.

When, in 1876, I visited the oil-fields then under exploitation in the eastern United States and Canada, the question of the genesis of these deposits received my earnest attention. I adopted the hypothesis of their animal origin, and in my report of this journey, entitled, *Die Petroleum-industrie Nordamerikas*, I briefly argued in its favor. Since that report was the first account given in the German language of the geological, technical, and commercial features of this interesting and economically important field, the hypothesis of animal origin (as well as the theory of anticlinals, advocated by me) was tested by observations in the European oil-fields, and, confirmed by geologists such as Tietze, Paul, Uhlig, O. Fraas, v. Gümbel, H. Credner, C. Zinken, and many others, it continued to win more and more advocates.

In 1888 appeared my book, *Das Erdöl und seine Verwandten*, in which I examined critically all the genetic hypotheses known to me, and not only demonstrated that of animal origin, but pointed out certain important factors in the process of transformation, particularly the comparatively low temperature and high pressure, as indicated by geological evidence. An important landmark in genetic hypothesis was thus established, furnishing for the guidance of synthetic experiments two important conditions of the transformation of animal bodies into petroleum.

A few weeks later, C. Engler, of Karlsruhe, tested my theses experimentally, by heating in retorts, under a pressure of 10 atmospheres, and to a temperature of from 300° to 400° C.,² first fish-oil, and afterward fishes and mussels. He obtained in this way a product very like petroleum, from which he isolated several members of the methane series. My hypothesis, thus further corroborated, became a theory, known as the Engler-Höfer theory, since both of us had equally contributed to its establishment. It won the support of chemists as well as geologists—whose numerous names it is not necessary to catalogue here.

² It is well known that time may replace temperature; so that nature performed the transformation at lower temperature. Moreover, processes of fermentation are involved; and these often produce very high temperatures.

It is noteworthy that the geologist and the chemist must co-operate in the solution of genetic problems in geology—the former by investigating the natural conditions attending a given formation, while the latter synthetically performs the process itself. Only when thus confirmed is the theory adequately established.

Vegetable Origin

A great additional service rendered by C. Engler was his tireless labor through many years in the investigation of the various transformations undergone, up to the present time, by the original materials and primary forms of petroleum—a labor which still further perfected the Engler-Höfer theory. These investigations showed that vegetable fats also, by dry distillation under high pressure and at comparatively low temperature, could be transformed into petroleum, as American chemists (Warren and Stoner, Sadtler, etc.) had already proved. Our theory could therefore be extended to include those plants which, like animals, contain fats and albumens, but no cellulose—namely, the microscopically small diatoms. T. Fegräus and A. F. Stahl, who first pointed out this source of petroleum, were followed by G. Krämer and A. Spilker, and, still later, by Potonié. This hypothesis is applicable only where, as in California, the minute and delicate siliceous shells of these fossil algæ occur in connection with oil-deposits.

Many American geologists and chemists like Lesley, Newberry, Ashburner, Shaler, Orton, Peckham, Mabery, etc., had already either advocated a common animal and vegetable origin, or (like the two highly esteemed investigators last named) assigned an animal origin to the nitrogenous California product, and a vegetable one to the non-nitrogenous oil of Pennsylvania. The contrary appears, however, to be the truth, since the diatoms are found in Californian, and not in Pennsylvanian, oil-fields. Probably the advocates of the double origin were not thinking of these microscopic vegetable forms at all; they speak in a general way of "plants." But the more highly organized plants contain cellulose, which would leave after distillation a carbonaceous residuum, such as is not found in petroleum and its deposits. For this reason, the above hypothesis, once so generally favored, could not be finally accepted.

According to our present knowledge, the original material of petroleum is principally fat, and subordinatedly wax, resin, and albumen. These substances, especially fat and albumen, occur chiefly in both the lower and the higher animal organisms. Petroleum, therefore, is mainly of animal origin, though it may have been formed, here or there, from fatty plants, particularly diatoms.

The foregoing is a brief review of the development and present condition of our knowledge on this subject. In Europe, as in America, most

geologists and chemists hold to the organic genesis of petroleum, and L. V. Dalton,² in his thorough essay, declares himself an adherent of the Engler-Höfer theory.

III. THE HYPOTHESIS OF VOLCANIC ORIGIN

As already observed, Eugene Coste, of Toronto, seems to be the most zealous American defender of the inorganic origin of petroleum. Three of his papers on this subject lie before me: (1) *The Volcanic Origin of Oil* (*Trans.*, xxxv, 1904, p. 288); (2) *The Volcanic Origin of Natural Gas and Petroleum* (*Journal of the Canadian Mining Institute*, vol. vi, 1903, p. 73), and (3) *Petroleums and Coals Compared in Their Nature, Mode of Occurrence and Origin* (*idem*, vol. xii, 1909, p. 273). He sees in petroleum "the product of volcanic solfataric emanation." In the following critical examination of his proofs, I refer chiefly to the interesting paper last named, which is the most recent, and, moreover, takes account of the arguments previously stated in the two others.

I must express in advance my surprise at his statement (p. 275): "The petroleum series includes all the natural hydrocarbons with the exception of the marsh gas." This hydrocarbon is found in every petroleum, though it escapes as a gas very rapidly, as soon as the oil is exposed. Of natural gas, it is generally the essential constituent.

The Proofs Advanced by Coste

Mr. Coste says in one of his papers: "The vital point is to actually show the carbon and hydrocarbon in the igneous rocks, lavas and emanations proceeding from these internal fluid magmas."

As illustrative instances, he cites (on p. 278 and following pages of his latest paper) the following:

1. "Oil in crystalline gneiss: In Placerita Canyon, five miles east of Newhall, Los Angeles county, California, a very light oil, almost naphtha, of a gravity between 50° and 60° B., is produced from crystalline gneisses which overlay the San Gabriel granite."

According to the investigations of G. H. Eldridge and Ralph Arnold, this so-called crystalline gneiss is not gneiss at all, but a metamorphic crystalline schist, perhaps Jurassic, and hence by no means an Archæan rock which could possibly be regarded as eruptive. It is a metamorphosed sedimentary, and can give Mr. Coste no help—rather the contrary. The oil of this locality may be in its primary deposit, or, more probably, it may have found its way thither from the neighboring Tertiary oil-field.

2. "Oil and bitumen in the quicksilver deposits of California."

² *Economic Geology*, vol. iv, No. 7, p. 608 (Oct.-Nov., 1909).

Being unacquainted with the geological relations of that district, I can offer no suitable explanation of this occurrence, and will only remark that these bitumens may have been extracted and transported from deeper bituminous rocks by the ascending ore-bearing solutions. Spirek ascribes to bituminous rocks the occurrence of bitumen in the Tuscan quicksilver-mines. Since the bitumens occur very seldom in ore-deposits—particularly in deep-seated veins,—they cannot be regarded as a general product of deep volcanic zones. They must have a purely local cause. At all events, such scanty occurrences of bitumen of all sorts prove nothing as to the formation of rich petroleum-deposits. This has been emphasized by Prof. Dr. L. Mrasek, of Bukarest, who is inclined, moreover, not to admit a deep source for the rare occurrences of bitumen in ore-deposits.

3. "Graphite and natural gas in the metalliferous vein of Silver Ialet, and graphite in the veins at Cobalt and Ducktown, Tenn."

To these instances also, what I have already said is applicable. No one would dare to infer from the sporadic occurrence of graphite in mineral veins that *deposits* of graphite are of volcanic origin—still less when (as in the Kaisersberg, Styria) the graphite is accompanied by plant-fossils. It seems to me equally audacious to argue the origin of the great deposits of petroleum found in sedimentary rocks, from the isolated and quite insignificant occurrences of bitumen in veins.

4. "Solid petroleums in pegmatite dykes, and other veins, associated with uranium, radium and vanadium."

We are here concerned, not with "solid petroleum" (a contradiction in terms, since petroleum is a liquid), but with a bituminous mineral, the combustion of which left an ash which, in one locality not named, contained uranium, and in another place, in Peru, contained vanadium. The remarks already made under (2) and (3) are applicable here; and I will only add that petroleum is almost entirely free from ash.

5. "Graphite, diamond and hydrocarbons in meteorites."

This phenomenon bears no relation to the origin of petroleum, nor is it at all surprising, since in the original cosmic material carbon must have been present (probably as carbonic acid), and must have been segregated in the meteoric masses, as in that of the earth.

6. "Oil and natural gas in volcanic rocks in Europe, Africa and Mexico."

These occurrences, few in number and always very small in extent, may be due to coal-beds or bituminous rocks which the volcanic eruptive broke through, distilling out and absorbing into its own mass some oil and natural gas. This distillation of bituminous material has long been practiced in the Scotch shale-oil industry.

7. "Natural gas in serpentine, Asiatic Turkey". (Chimaera).

Alexander von Humboldt suggested long ago⁴ that this emission of gas might be connected with petroleum. E. Tietze,⁵ who studied the phenomenon on the spot, adopts this view, and calls attention to the neighborhood of the so-called "Flysch" formation, which so frequently carries petroleum.

8. "The occurrence of oil around volcanic necks, Mexico."

This proves nothing, since the mineral oil of Mexico, and especially of the State of Tamaulipas, named by Mr. Coste, is widely distributed, and occurs both near volcanic necks and far from them. Villarello,⁶ Division-chief of the Mexican Geological Institute, and one of those best acquainted with the oil-occurrences of Mexico, concludes: (a) that the oil comes from a marine fauna; and (b) that, in the districts explored hitherto, it is found only in secondary deposits, situated in highly disturbed terrain, connected frequently with basaltic eruptions. Since the volcanic tuffs are highly porous, it is not surprising that the oil, in its migration, should accumulate there in special abundance.

Of all the foregoing proofs of the volcanic origin of petroleum, only two, No. 1 and No. 8, are really pertinent to the question of the genesis of valuable deposits of oil. The rest are so insignificant that they prove nothing as to the production of oil in large quantities, and have for us only a purely scientific interest. And the two exceptions, adducing the occurrence of petroleum in alleged gneiss, and in connection with volcanic necks, have been shown, I think, to be entirely inadequate as proofs.

As a logical consequence of Mr. Coste's view, deposits of petroleum should always be found in the vicinity of volcanic eruptions. But this is not the fact. In the Carpathians, the "outer bend," through Galicia, the Bukowina, and Roumania, is free from eruptives and rich in oil, while the "inner bend" is rich in eruptives and poor in oil. At Baku, in Alsace, and in North Germany, as in Canada, New York, Pennsylvania, Ohio, West Virginia, Louisiana, Texas, etc., there are no eruptives near the rich oil-deposits. In Java, Sumatra, Borneo, and Burmah, the oil-fields are far from the regions of eruptive activity.

These weighty facts, completely contradicting the volcanic hypothesis, Mr. Coste seeks to deprive of force by the assumption that oil and gases have ascended from greater depth through fissures, and thus were deposited far from eruptive masses. But it is remarkable that the Hungarian Carpathians are much more disturbed than those of Galicia and Roumania, and are, nevertheless, poorer in oil—indeed, for the most part, contain no oil at all. The most important and profound disturbance of the Galician Carpathians—the so-called *Klippenzone*—is barren of oil,

⁴ *Kosmos*, vol. iv, Observation 51.

⁵ *Jahrbuch der k.k. geologischen Reichsanstalt*, Vienna, vol. xxxv (1885).

⁶ J. D. Villarello in *Das Erdöl*, by Engler and Höfer, vol. ii, pp. 643 and 650 (1909).

like its neighbor, the *Weichselbruch*. The Alps are traversed by deep faults and dislocations, many of which still make themselves disagreeably felt as seismic surfaces; yet no noteworthy oil-deposits have been found among them. At the foot of the Alps on the north, the gas-springs of Wels, in Upper Austria, are found in quietly deposited and undisturbed Miocene strata. In the rich oil-bearing flat anticlinals of Pennsylvania, I sought in vain for any dislocations worthy of mention; and not one of the intelligent "oil-men" whom I met could point me to any such thing. East of that oil-region, we find in the Appalachians mighty disturbances of all kinds—deep fissures, sharply arched anticlinals,—but no oil. At Pechelbronn, in Alsace, the slightly inclined oil-bearing sandstones were formerly mined and thus thoroughly explored, without the discovery of a single dislocation, showing that the oil was occupying its primary place of deposit. K. Kalickij⁷ proved the same proposition for the oil-occurrences on the island of Tscheleken. Even the photographs accompanying his paper are conclusive.

The Objections Advanced by Coste

In order to weaken objections to his own theory, Mr. Coste urges⁸ the following objections to the theory of organic origin:

1. "It cannot possibly explain the large petroleum fields below the Carboniferous."

Can the solfataric hypothesis do that? No: Mr. Coste's proofs—except No. 1 and No. 8—rest wholly upon isolated and minute occurrences of bitumen. No one has ever observed at a solfataria any accumulation of petroleum worthy to be mentioned. On the other hand, F. Quenstedt⁹ has shown that 1 square mile of the bituminous Posidonia slates of Suabia, rich in animal fossils, contains 200 million hundredweight of oil. In other words, the animal remains of a single sedimentary bed can furnish enormous amounts of oil. What was possible after the Carboniferous era may have been possible before it also. Biological activity on the earth has been immense, continuous, and widespread; whereas volcanic activity has been local, and often but temporary, discharging here and there comparatively insignificant quantities of hydrocarbon gases.

2. "Neither can it explain the petroleum in the volcanic emanations of today."

No considerable quantity of hydrocarbon has ever been found in a solfataria. And where one or a few per cent. have been found, they can be referred to bituminous strata which have been intersected. Moreover, positive reports of hydrocarbons (usually given as methane, CH₄) are to

⁷ Ueber die Lagerungsverhältnisse des Erdöls auf der Insel Celeken. *Mémoires du Comité Géologique*, St. Petersburg, New Series, livraison 59 (1910).

⁸ *Journal of the Canadian Mining Institute*, vol. xii, p. 295 et seq. (1909).

⁹ *Einst und Jetzt*, p. 57.

be received with caution. The gases of the Hawaiian volcano Kilauea are often cited as an example. But when collected directly by L. Day and E. S. Shephard¹⁰ they were found to contain no hydrocarbons at all. So this proof also fails. The small quantities of marsh-gas, produced by Brun at Geneva through the heating of certain lavas, may have formed themselves during the process through the decomposition of other gases, for instance, according to the equation



3. Nor can the organic theory explain the petroleum "in the volcanic or igneous rocks in all parts of the world."

4. "Nor in crystalline rocks; in California and New Brunswick, for instance."

5. "Nor in meteorites."

6. "Nor in metalliferous veins."

These four points have been already discussed, and shown to be invalid.

7. "It is also at a loss to explain why the petroleum fields in every district are found grouped along certain lines and why the petroleum are found there in many horizons, while outside of the lines in just the same strata and over much larger areas all the horizons are barren."

This raises the comprehensive question of the structure of the petroleum-deposits, which cannot be treated within the limits of this paper. I will only say briefly that the oil is found along certain lines, because it occurs (1) in fissures, (2) in folds, and (3) in long-drawn channels of sand. The fissures are directly connected with the primary deposits. In folds, anticlines, monoclines, etc., the position of the oil is determined by the accompanying natural gas and water. The three substances arrange themselves according to their specific gravity, along the lines and surfaces presented by the shape of the fold. If the oil-bearing sands occur in long, slender bodies, as in Alsace, the grouping of the oil "along certain lines" is not surprising. Since the oil-deposits are coastal formations originating under special conditions, it is comprehensible that they cannot follow throughout the same geological horizon.

8. "It cannot explain either how the petroleum can possibly travel out of their supposed organic-remain source in some impervious clay or shale to accumulate in a few porous receptacles far distant laterally and sometimes hundreds and thousands of feet above, or even below as some assert, and this all through most impervious rocks and without any impelling force behind, or any cracks, joints or fissures to follow since the decomposed products of the organisms must naturally be supposed to come from the whole mass of the strata through which the organisms were and there could not be fissures, cracks and joints to all parts of the strata."

If I understand Mr. Coste correctly, this passage is directed, not so much against the theory of organic origin as against the hypothesis, so popular in America, of the regional migration of petroleum. In this re-

¹⁰ *Journal of the Washington Academy of Sciences*, vol. iii, p. 475 (1913).

spect, I heartily agree with him. I too maintain that the migration of oil can take place only in cracks, joints, and fissures, the source of motive energy being (as has been often demonstrated) the accumulation, in the primary deposit, of natural gas under high pressure.

9. "It cannot possibly explain why the petroleums, although found today in their reservoir-rocks under strong pressures, cannot by means of that pressure, return and disperse back to their original sources; they should be able to return the way they came, nothing is to prevent them and there is plenty of pressure for the return voyage if one admits the first voyage from the organic source."

This question might be applied to Coste's hypothesis also. As already remarked, petroleum is driven by gas-pressure to a considerable altitude in fissures; and its removal leaves in the original deposit a space in which the gases collect and keep the oil above them, as, for instance, in the so-called inverted siphon, when partly emptied, the entrance of carbonic acid gas continues to maintain the height of the water in the discharge-pipe.

10. This objection, based on alleged features of the occurrence of petroleum in California, I must leave to my esteemed colleagues, Ralph Arnold, B. Anderson, G. H. Eldridge, and other distinguished investigators of the oil-geology of that State. It will possess for them no difficulty.

11. "It cannot possibly explain again, if the petroleums can travel so freely through the strata as to be able to accumulate under an anticline from organic remains deposited far and wide laterally (at least a mile or two or much more in order to allow for the quantities obtained in many fields), why they did not escape out into the free air only a few hundred or a few thousand feet away at most; the shales above the sands are not any more impervious than the shales below the sands, which on that theory are supposed to be the source of the petroleums, and if they can travel freely through the shales which are the most impervious rocks of the sedimentary series, I repeat, what is to prevent them from getting out into the atmosphere?"

This question properly concerns, not the organic origin of petroleum, but a hypothesis of its migration, advanced to explain the formation of productive deposits—a hypothesis which I reject, holding that petroleum originated in the sands in which it is found, unless it has passed through fissures to other sand-strata.

12. "It cannot account for the continual absence of petroleums in the hard parts of organisms preserved in the sedimentary strata."

Oil can be formed from the soft parts of animals under certain conditions only, among which is the exclusion of air. We find on the sea-shore many hard parts, such as shells and skeletons, of marine animals, from which their organic contents have totally disappeared, having been destroyed by the oxygen of the air. Since this generally finds access to dead animal matter, we find the hard parts without oil very frequently, and oil itself infrequently in comparison, because only under special favoring conditions.

13. "It cannot explain the evident non-connection of petroleum deposits with coal-beds."

Since the latter are land-formations from plants containing cellulose, while the former are marine estuary-formations from animal remains, there can be no connection between the two organic processes or their products.

14. "It cannot account for the continual association of petroleum with strong salt and sulphur waters."

Since the original materials of petroleum accumulated in marine bays, having but limited relations with the ocean, the presence of strong salt water is not surprising, but constitutes, on the contrary, a proof of our theory. Sea-water is known to contain sulphates also, which, in the process of oil-formation, can be reduced to sulphides, or even to sulphur. As a marine formation, petroleum may be accompanied by salt, gypsum, calcite, and dolomite; and this explanation of their presence seems to be more natural than that of a volcanic source.

Further Considerations

I have thus answered in detail both Mr. Coste's objections to the organic, and his arguments for the inorganic, origin of petroleum. The latter, however, constitute, strictly speaking, an incomplete statement; for he contends only that petroleum was brought by solfataras into the cooler parts of the earth's crust. Concerning the questions, *out of what* and *how* it was formed, he is entirely silent. His explanation, even if we were able to accept it as correct, goes only half way, like those of his predecessors, Lenz (1831), Rozet (1835), S. W. Pratt (1846), Choucourtois (1863), Thore (1872), Fuchs and Sarasin. It is at best a plant without a root.

Even if we had proved, or should hereafter prove (as has never yet been done), the presence in solfataras of large quantities of marsh-gas, CH_4 , such gas would stream into the air, without forming petroleum. Besides, we know of no process by which CH_4 can be converted into the higher members of the paraffine series, or any member of the naphtha series. This circumstance likewise deprives the very rare occurrence of CH_4 in ore-deposits or volcanic rocks, of all significance as to the origin of petroleum.

As a sincere friend of the petroleum industry, I am heartily sorry that I must reject Mr. Coste's emanation-theory; for, if it were true, we might expect our petroleum-supply to prove inexhaustible, new quantities being continually furnished by solfataric activity. Unfortunately, that is not the case.

Mr. Coste mentions an occurrence of hot water with petroleum in Texas. This is a purely accidental phenomenon; since neither in the great

Yellowstone region of thermal springs nor in any of the European hot springs has petroleum, or even marsh-gas, been observed.

Why are oil-deposits lacking in the highly fissured true Archæan rocks of Scandinavia, Bohemia, the central Alps, the Appalachians, etc.; and why do they appear first in the sedimentaries deposited at a time when the earth had become populated with organisms? This can be construed only as a proof of the organic origin of petroleum.

If this oil had ascended from great depths, it would have impregnated all porous strata. But, on the contrary, we find repeatedly, between two oil-bearing horizons, porous rocks containing no oil, like, for example, the Jamna sandstone in the Galician Carpathians. Underlying the oil-sands themselves, there are porous, yet barren, rocks.

If petroleum were the product of distillation at high temperature, it could not contain any primary paraffine, and it would be richer in olefins. Neither of these conclusions is confirmed by the facts.

The occurrence of free nitrogen (not in the form of air) in many petroleum and (often in considerable amount) in natural gas, cannot be explained by any volcanic hypothesis, but furnishes another strong proof of organic origin. The same may be said of the optical properties of petroleum, and of the presence of cholesterin, which seems to be a condition of the polarization, and is a special indication of animal origin. Moreover, the high-molecular pyridin bases, observed in many oil-regions (Galicia, Alsace, Baku, Fergana, Roumania, Sumatra, California, Egypt, Algiers) speak conclusively against a volcanic, and in favor of an organic—particularly an animal—origin. The general chemical character of petroleum as an unstable mixture of hydrocarbons bears similar testimony against any supposed pyrogenic process at high temperature.

All geological and chemical facts concerning the occurrence of petroleum bear unanimous witness in favor of its organic origin, and hence conclusively against its production from inorganic substances, and the collateral hypothesis of emanation. The doctrine of the volcanic origin of petroleum deposits must therefore be pronounced to lack scientific foundation.

To demonstrate this fact in a review of the publications of Mr. Coste, one of the most meritorious and zealous representatives of that hypothesis, has been the purpose of the foregoing remarks. Hence I have adduced proofs of organic origin only so far as they contradicted the opposite view. For a detailed exposition and defense of the former theory, I refer to my two books: *Das Erdöl und seine Verwandten* (3d ed., published by Vieweg at Braunschweig in 1912), and *Die Geologie, Gewinnung und der Transport des Erdöls* (published by Hirzel at Leipzig in 1909), the latter of which constitutes vol. ii of the comprehensive monograph issued by Engler and myself under the title, *Das Erdöl*.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

The Illinois Oil Fields

BY H. A. WHEELER, ST. LOUIS, MO.

(New York Meeting, February, 1914).

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History

ILLINOIS has so recently attained the third place in the oil production of the United States that few realize its great importance, or are aware of its highly profitable character. Since 1907 Illinois has furnished about 15 per cent. of the United States output and about 10 per cent. of the world's production. The value of the output in 1913 is estimated at \$30,000,000, of which about \$20,000,000 is profit.

The present prosperity dates from 1905, but efforts have been made since 1865 to develop production and geologists have been very sanguine as to its future for over 25 years. Had the oil operators been as confident as the geologists, the field would have been opened at least 30 years earlier. For the rich Eastern Illinois oil field is located along the La Salle anticline, that was mapped 46 years ago by Professor Worthen in the

first geological survey of the State.¹ Prof. T. B. Comstock enthusiastically wrote in 1887² about the highly promising character of this anticline, to which he again called attention in 1889.³

The writer's studies in 1888 were convincing as to its oil future, yet to have then advocated drilling would have earned the reputation of being a dreamer. In fact, as late as 1903 the writer's suggestion to a prominent Pittsburg oil operator to prospect in Illinois met with contempt and derision, which was representative of the general feeling then held by the oil fraternity, yet this same party made a large fortune four years later out of a block of Illinois oil leases.

The first commercial wells were brought in at Litchfield, 50 miles northeast of St. Louis, where in 1882, in drilling for coal, gas was discovered that supplied the town several years. Later, some oil wells were brought in that until 1902 produced a lubricating oil that sold for \$5 a barrel.

In 1887, at Sparta, 40 miles southeast of St. Louis, in drilling for water, gas was discovered that supplied the town about 20 years, and in 1906 several oil wells were brought in half a mile northeast of the gas wells.

In 1890, in drilling for water, gas was discovered near Pittsfield, in Pike county, that later developed into a field over 10 miles long.

The present highly prosperous era dates from 1904, when a small gas well was drilled near Casey, in Clark county, by J. J. Hoblitzell, who was induced to prospect at Oilfield on the showing of oil and gas made by some old wells that had been drilled in 1865 by Chicago parties. The latter drilled several wells on the evidence of oil and gas seepages, but, although a little was found, the wells were a failure from not casing off the water; i.e., they were drowned out. Several more small wells were brought in, but they were so discouraging that they failed to interest the scouts that flocked in when leases were only \$5 to \$10 an acre. When a 40-barrel well was subsequently brought in, the "talent" promptly scurried after leases, which rapidly advanced to \$100 to \$200 an acre, and since then the Eastern Illinois field has rapidly developed along the La Salle anticline. The southern extension was discovered in February, 1906, in Crawford county and by midsummer the adjoining Lawrence county began producing. The Allendale pool, at the south end, was discovered in September, 1912. Shipments by tank cars started in June, 1905, and within a year the first pipe line reached the field, followed by four others that had a daily capacity of 112,000 barrels by 1909.

On the western side of the State, the Butler oil and gas was discovered in 1907, the Centralia pool in 1908, the Greenville, Carlinville, and Sandoval pools in 1909, and the Carlyle pool in 1911.

¹ *Geological Survey of Illinois*, vol. iii (1868).

² *Oil and Natural Gas in Illinois, Proceedings of the Illinois Society of Engineers and Surveyors*, vol. ii, p. 92 (1887).

³ *Engineering and Mining Journal*, vol. xlviii, No. 26, p. 565 (Dec. 28, 1889).

The total production to Jan. 1, 1914, is 209,018,914 barrels, valued at \$156,025,398, as shown in Table I. This record for output and values has never been equaled by such a young field, as it requires time, as well as capital, to develop a new field, especially where there was such a lack of confidence. While the latter has disappeared since 1906 on the eastern side of the basin, it still holds as regards the western side of the basin.

TABLE I.—*Summary of Illinois Oil Production*

Year	Output Barrels	Value	Stocks Dec. 31 Barrels	Produc- ing Wells	Average Price
1905	181,084	\$116,561		189	\$0.64
1906	4,397,050	3,274,818	2,509,598	3,093	.75
1907	24,281,973	16,432,947	15,751,305	7,353	.68
1908	33,686,238	22,645,881	29,209,660	10,372	.67
1909	30,898,339	19,788,864	32,343,887	11,152	.64
1910	33,143,362	19,699,383	31,324,784	12,171	.59
1911	31,317,038	19,734,339	24,063,870	12,753	.63
1912	28,601,308	24,332,605	15,709,738	13,222	.85
1913	22,512,522	30,000,000	5,613,372	14,000?	1.32
Total.....	209,018,914	\$156,025,398			

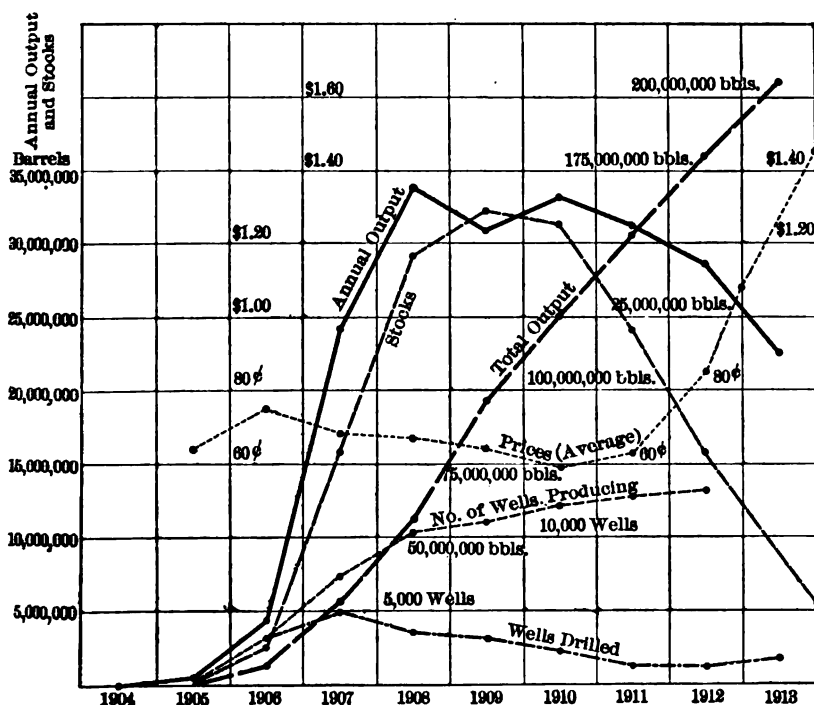


FIG. 1.—CHART SHOWING OUTPUT, STOCKS, PRICES, AND WELLS, ILLINOIS OIL FIELDS.

Since 1904, 21,730 wells were drilled up to Jan. 1, 1914, of which 82.9 per cent. were producers, that came in with an average daily production of 38.5 barrels per well, according to the monthly statistics published by the *Oil City Derrick* (Oil City, Pa.). This average is based on tank gaugings, and while it is much below the popular estimates, it is high compared with the other high-grade oil fields. The value of wells averaging 38 barrels will be appreciated when it is considered that a 100-acre lease in Illinois that averages 10 barrels per well usually nets 100 to 125 per cent. on the investment.

A condensed *résumé* of the Illinois oil statistics is presented in the diagram, Fig. 1, which shows the annual production, the total production, surplus stocks, average annual prices, the oil wells drilled, and the number of producing wells.

Location

The Illinois oil fields are in the southern half of the State and consist of two groups, known as the Eastern and the younger Western fields.

The Eastern Field.—The Eastern field is situated along the eastern edge of the State and mainly in Clark, Crawford and Lawrence counties, with slight extensions into Edgar, Coles, Cumberland, Jasper, and Wabash counties. The strike is quite regular and runs about 20° west of north. The length in Illinois is 66 miles from the Westfield pool at the north end to the Allendale pool at the south end. The field crosses the Wabash river at St. Francisville and extends 45 miles through the southwestern portion of Indiana and continues 50 miles southward to Hartford, in Kentucky, where several wells were brought in last summer. This gives a length of over 160 miles along the La Salle zone of deformation on which oil and gas have been developed. A marked change occurs beyond the Illinois border, as the field becomes very irregular and highly spotty, at least as far as developed, and lacks the remarkable reliability and continuity that characterize it in Illinois. The width of the field ranges from 2 to 4 miles at the northern end and from 5 to 12 miles in the central and southern portions. The field seems to be over 20 miles wide in Crawford county, but this is undoubtedly due to a cross anticline, the influence of which is seen in the new pools being opened in Sullivan county, Ind., which adjoins it on the east.

The accompanying three maps, Fig. 2, illustrate the development of the field when respectively two, four, and nine years old. They show that it is premature to define the limits, as the boundaries are constantly extending, and new sands are being brought in. A study of these maps shows that at the end of two years, when about 4,000 holes had been drilled, apparently less than 25 per cent. of the area along the anticline was productive. At the end of four years, when about 12,000 holes had been drilled, fully 50 per cent. was found to be productive. At the end

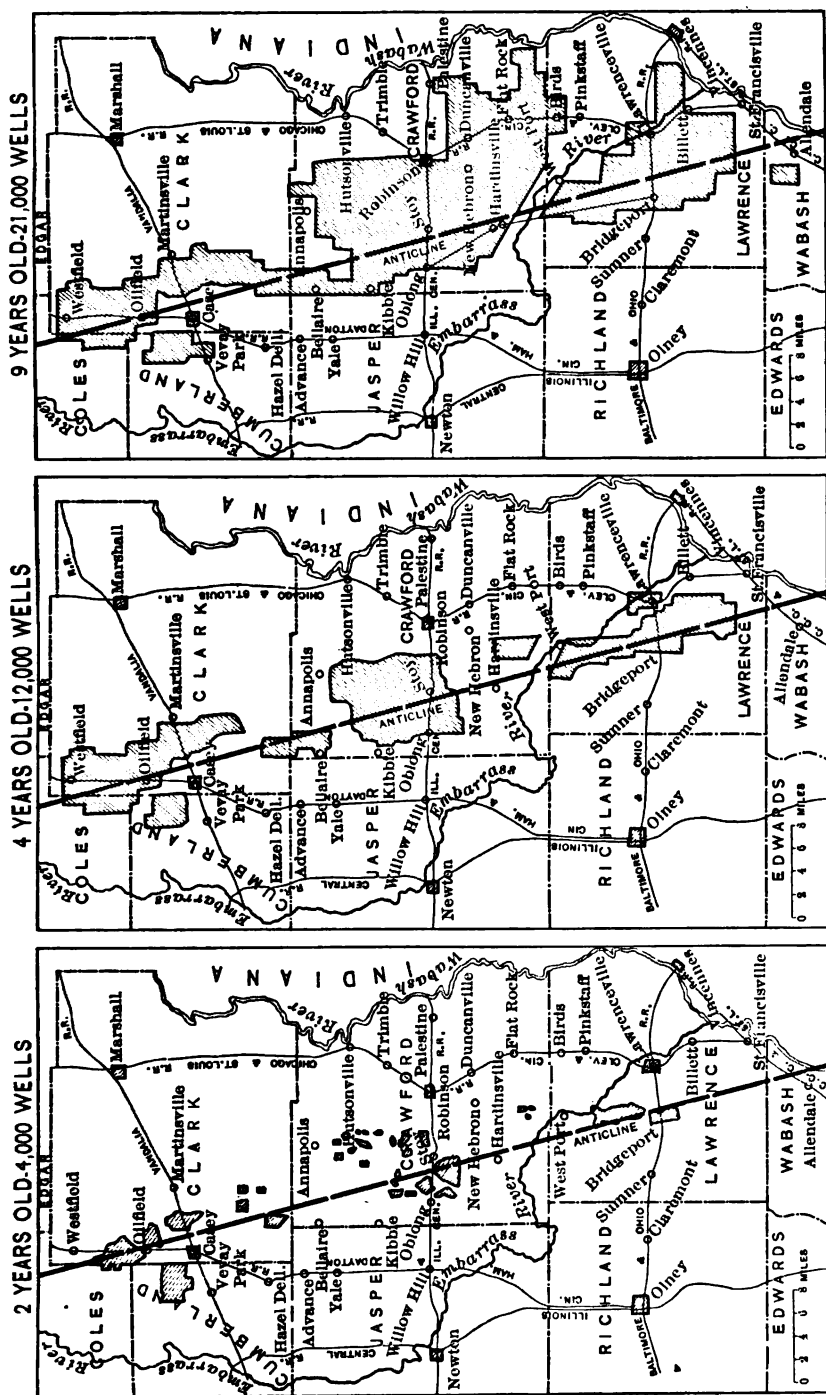


FIG. 2.—DEVELOPMENT OF THE EASTERN ILLINOIS OIL FIELD, OPENED IN 1905.

of nine years, when about 21,000 holes had been drilled, over 80 per cent. had proved productive. The Eastern Illinois field to-day is the largest, most regular, and most reliable individual oil field in the United States. If the Allendale pool is omitted, there is only one break in its

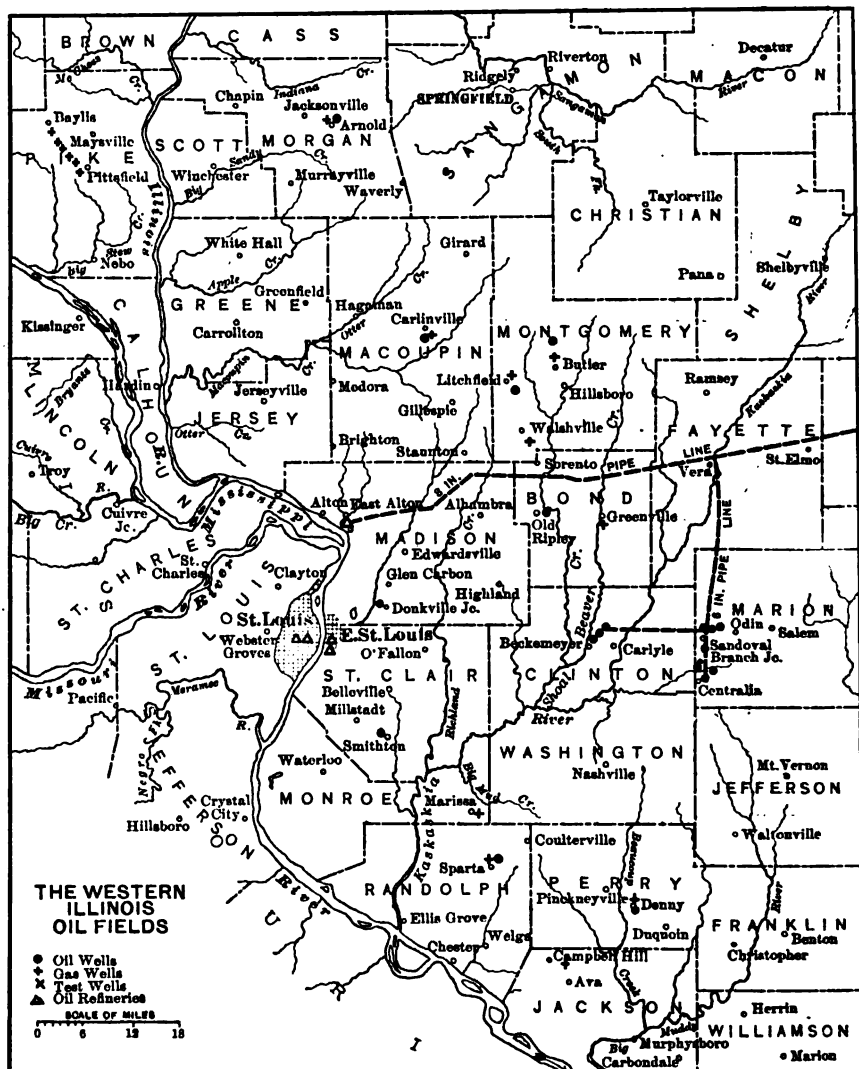


FIG. 3.—MAP SHOWING WESTERN ILLINOIS OIL FIELD.

continuity in 60 miles, or where the Embarrass river crosses the anticline. While to-day several miles separate the Allendale pool from the main field, more thorough drilling may show that the intervening ground

is productive, as this has repeatedly happened in the past eight years' experience.

The Western Field.—The Western Illinois oil and gas field is situated in Marion, Clinton, Bond, Montgomery, Macoupin, Morgan, Pike, Madison, St. Clair, Randolph, Jackson, and Perry counties, as shown on the accompanying map, Fig. 3. As the western flank of the Illinois basin is about three times larger than the eastern, and as there are more anticlines, it will probably develop into a larger field and ultimately produce more oil, although this prediction to-day seems as rash as earlier ones made by the writer that have since made good. For the sands and other favorable formations are practically identical on both sides of the basin, while the more numerous western anticlines have a much larger collecting area to supply them. As the Western field lacks the simplicity and regularity of the Eastern field, it will be more difficult to prospect and the percentage of dry holes will be larger, especially if developed in the haphazard manner in which most prospecting is carried on by "practical" oil men. For the attitude of most oil operators toward the aid that can be rendered by geology and trained experience is similar to that of the steel maker 50 years ago about a chemist—"an amusing scientific chap who uses big words but knows nothing about making steel."

Geology

Structure.—The geological structure of Illinois is comparatively simple and regular, like its topographic features, which latter mainly consist of a very slightly undulating sheet of glacial drift 50 to 150 ft. thick through which occasional sluggish streams have sometimes eroded to bed rock.

Under the flat prairie surface occurs a spoon-shaped basin. Its north-and-south axis is 400 miles long and its width is 140 to 180 miles. The basin is occupied by alternating sediments, mainly of Carboniferous age, that have a maximum thickness of about 2,000 ft. The formations thin out, die out, and outcrop toward the north and thicken and increase in number toward the south. The oil-bearing "sands" tend to decrease in richness toward the north, as a result of this thinning of the beds, and tend to enrich toward the south, from the thickening of same, until they come within the malignant influence of the Ozark uplift. Thinning and impoverishment of the oil sands toward the north also occur in the similar Mid-Continent oil field (Oklahoma and Kansas), in which the present production is also derived from the coal measures, and a similar uplift ruins the southern end of the field.

Upon the flanks of the Illinois basin occur anticlines, or folds, or elongated domes, which in water-charged sediments are the natural traps, or zones of arrest, or concentration areas, of oil and gas. The

anticlines vary considerably in the amount of the doming and are relatively flat and broad compared with those of Pennsylvania. They are usually flatter and less persistent in the Western field and tend more to the type of a series of domes along the uplift, rather than a continuous ridge type as in the main Eastern field.

Coal Measures.—The oil thus far produced in Illinois, with minor exceptions, is derived from the Carboniferous system, or the coal measures, which consist of alternating shales 5 to 150 ft. thick, limestones 2 to 50 ft., bituminous coals 1 to 8 ft., and sandstones 5 to 60 ft. The accompanying well section illustrates the variety and thickness of the formations, in which only the basal sandstone (the Ferruginous, or Benoist) carried oil at this particular place. The six sandstones between 915 and 1,439 ft. frequently contain oil at other places, especially in Lawrence county, while two shallow sands did not show in this well, of which the Dykstra (600 ft.) produced oil 2 miles west of this hole.

The shales fluctuate greatly in color, texture, and thickness and are always the predominating formation. The limestones are usually thin and, with one exception, are not persistent. The coal seams are generally thin, or 1 to 3 ft., excepting No. 6, and occasionally Nos. 5 and 2, which range from 4 to 8 ft. The thin seams are not persistent, while No. 6 is remarkably persistent in western Illinois and is easily recognized by its limestone roof. The sandstones are usually porous, are firm and stand up well, and are generally water-bearing. The porosity varies considerably, and to a less extent the fineness of grain, which largely explains the variations of different wells in the same field when not complicated by a variable thickness.

Type Section of an Illinois Oil Well

The Langewich Well, Centralia; 70 Barrels

Formation	Thickness	Depth
	Feet	Feet
Sandy clays (drift).....	155	155
Shale.....	10	165
Limestone.....	10	175
Shale.....	90	265
Limestone.....	20	285
Black shale.....	67	352
White shale.....	8	360
Black shale.....	115	475
White shale.....	25	500
Black shale.....	115	615
Limestone.....	10	625
Coal—No. 6 seam.....	7	632
Black shale.....	18	650
Limestone.....	25	675

Formation	Thickness	Depth
	Feet	Feet
White shale.....	40	715
Dark shale.....	33	748
Coal No. 5.....	5	753
Black shale.....	42	795
White shale.....	50	845
Black shale.....	30	875
Limestone.....	12	887
Black shale.....	13	900
Limestone.....	15	915
Sandstone—dark.....	15	930
Black shale.....	80	1,010
Sandstone—salt water.....	11	1,021
Black shale.....	16	1,037
Sandstone—salt water.....	43	1,080
Black shale.....	20	1,100
Limestone—hard.....	25	1,125
Black shale.....	51	1,176
Limestone and shale.....	50	1,226
White shale.....	24	1,250
Black shale.....	45	1,295
Sandy limestone.....	10	1,305
Sandstone—dark.....	38	1,343
Dark shale.....	25	1,368
Sandstone—white.....	23	1,393
Limestone.....	17	1,410
Shale.....	14	1,424
Sandstone.....	15	1,439
"Red rock" (shale).....	11	1,450
Dark shale.....	5	1,455
Limestone.....	30	1,485
White shale.....	30	1,515
Limestone.....	7	1,522
"Pencil slate" (caves).....	30	1,552
"Red rock" (shale).....	8	1,560
Shale.....	32	1,592
Oil sand—9 ft. pay.....	39	1,631

(The "big lime" is probably at 1,650 to 1,700 ft.)

Sub-Carboniferous.—The following successively deeper measures occur in Illinois, some of which have an assured future as oil producers:

The Sub-Carboniferous, or the Mississippian, immediately underlies the coal measures and is 500 to 1,000 ft. thick. It consists mainly of limestones and is known as the "big lime" by the drillers. An upper bed (the St. Genevieve) that is known as the "McCloskey sand" is occasionally oölitic and has proved the richest in the State, as the wells occasionally come in at 1,000 to 4,000 barrels.

Beneath the "big lime" occurs a shale bed that when sandy was found to carry oil and gas along a small anticline in St. Louis on the

western fringe of the basin. Like all thick limestones, even when mostly non-magnesian, like this, there is always the possibility of crevices and chambers serving as reservoirs for oil accumulation that are likely to produce good wells. While such reservoirs are erratic and will be difficult to find, they suggest occasional wells. The 200 water wells drilled about St. Louis in this formation occasionally show oil, which indicates that where artesian conditions have not washed away the oil valuable wells will be found.

Where a fault has exposed the "big lime" as a bold bluff near Grafton, 30 miles northwest of St. Louis, it is soaked with oil for about 125 ft. This "oil in sight" stimulated considerable drilling in the neighborhood that resulted in dry holes, as the oil that once filled the limestone escaped millions of years ago through the "knocking out of the head of the barrel" by the fault. This exposure is highly important in almost assuring large wells when this horizon is tapped under a thick cover with favorable structural conditions.

Recently wells of 10 to 25 barrels⁴ have been found in Oklahoma in the "big lime," although the prospect drilling in that field, as in Illinois, has almost invariably stopped at the top of this formation. As the geology of the Oklahoma and Illinois fields is similar in many respects, the developments in either field are likely to prove helpful in prospecting the other.

The Devonian.—The Devonian formation is usually thin, or 2 to 10 ft., where it outcrops along the Mississippi river, on the western edge of the basin, and is generally a black shale. It has been found over 100 ft. thick, and when sandy is likely to be an important, though erratic, producer of oil. Some oil of excellent grade, or 39° B., was found in the Devonian at Old Ripley, in Bond county, 40 miles northeast of St. Louis, at 2,000 ft., and also at Peters, 10 miles northeast of St. Louis, at 1,300 ft., but at both places the sand was very thin and hence the wells were small.

The Niagara Limestone.—The Niagara limestone is 50 to 200 ft. thick. The Pike County gas field at 150 ft. in western Illinois and the recently developed deep sand (2,800 ft.) at Casey, in eastern Illinois, are in this formation. The famous old crevice well at Terre Haute, Ind., 30 miles east of Casey, also is in the Niagara lime. Where this limestone is dolomitic, and consequently porous, as in Pike county, it is likely to prove a productive horizon if anticlines or other structural conditions are favorable.

The Trenton Limestone.—The Trenton is 200 to 400 ft. thick and is partly a porous dolomite and partly a compact limestone. A shale that occurs with it is so heavily soaked with oil as to burn like a candle and is known as "oil rock" where it outcrops for about 300 miles along the

⁴Since the above was written, a 250-barrel well has been struck at Big Heart, Okla., at a depth of 300 ft. in the "big lime" and with a surface depth of 2,200 ft.

western fringe of the basin. This is the formation that carries the oil and gas in the adjoining Indiana field and also in the Lima field, of western Ohio. As it is usually 2,500 to 3,500 ft. deep in southern Illinois and has a thick shale cover, it is likely to be a rich gas producer, as the great depth has probably prevented leakage. It is a promising formation for both oil and gas and underlies the entire oil field, though at varying depths.

Very Deep Sands.—About 200 to 400 ft. below the Trenton occurs the St. Peters sandstone, which is usually 100 to 200 ft. thick and generally coarse and very porous. Still deeper are the "second" and "third" sandstones, that are interbedded with magnesian limestones. While these are possible oil horizons, there are features that are not encouraging, while the depth will be 4,000 to 5,000 ft.

Oil Conditions.—The chemical, physical, and structural conditions in Illinois are ideal for the occurrence of profitable oil pools, and the geologists incurred little risk in predicting their discovery many years in advance of the drill. For there is an abundance of material to produce the oil in the bituminous shales, organic formed limestones, and coals. There are numerous porous beds to act as reservoirs in the porous sandstones and dolomites. There are many shales to act as impervious covers, or "stoppers to the bottle." There is a continuous dip on the sides of the basin to permit gravity to rearrange the original concurrent association of water, oil, and gas. There are anticlines to trap or arrest the ascending oil and gas and concentrate them into pools, and the magnitude of the basin is sufficient to insure large pools. Stratigraphic conditions have prevented artesian circulation, at least in the lower sands, thus preventing the washing away and dissipation of the oil after reaching the anticlines. Faults and dikes are either absent, or are so slight in the main part of the basin as not to interfere with the accumulation of the oil, or to cause its dissipation. While these conditions have been known to the geologist for a long time, it is only since 1905 that they have been demonstrated to the satisfaction of the oil operators on the eastern side of the basin. The oil men are still skeptical as to the western field, which is only in the infancy of its development, and they have thus far lacked the courage to test the deeper horizons below the coal measures.

Description of the Eastern Illinois Field

The Eastern field is situated along a strong anticlinal fold known as the "La Salle," as it passes through the town of La Salle in northern Illinois, where it is a heavy fault. As the axis of the anticline dips south, the sands deepen and increase in number and thickness to the southward. The No. 6 coal seam, which is well marked and persistent in western

Illinois, serving as a valuable datum plane, is frequently absent in eastern Illinois, or is difficult to recognize, so that the correlation of the "sands" is uncertain. All of the sandstones (or "sands") are more or less lenticular, as usual, or thicken and thin, come in and die out, although two of them are surprisingly persistent.

The field is divided into three districts, each of which has marked individuality, which are known as: 1, the Clark County, or Northern, or Shallow, district; 2, the Crawford County, or Central, district; and 3, the Lawrence County, or Southern, or Deep, district.

The Northern District.—The Northern district is mainly in Clark county, with slight extensions into Edgar, Coles, and Cumberland counties. Casey is the commercial center, which, like the other towns in the oil belt, has greatly grown and prospered since the field opened in 1905.

Most of the wells are 350 to 400 ft. deep and are in a light gray, porous, dolomitic limestone that has 8 to 30 ft. of pay. The wells come in at an average of 25 barrels, ranging from 5 to 50 and occasionally at 100 to 200 barrels per day. The small cost of a well resulted in considerable over-drilling, with a consequent rapid decline, and many wells were abandoned when they fell off to one to two barrels with oil selling at 60c. to 68c. Recently the abandoned leases have been re-drilled and again made very profitable, especially as another oil horizon has been found 40 to 100 ft. deeper, that comes in at 10 to 50 barrels.

In the Siggins pool, 2 miles west of Casey, a sand occurs at 600 ft. that is 20 to 60 ft. thick and came in with 10 to 100 barrel wells and plenty of gas.

Oil shipments by rail began in June, 1905, and in the following three years the field developed rapidly. Since then the extensions have been slight and there was little activity until last year, when a second pay was found at about 450 ft. The district is producing about 7,000 to 8,000 barrels daily.

Recent deep drilling near Westfield, at the northern end of the field, has found a dark-green, high-paraffine oil at 2,750 ft. As the wells are small for such a deep sand, or 10 to 20 barrels thus far, only a few have been drilled and they are on the western edge of the pool. They are probably in the Niagara limestone, as are the Terre Haute well, 30 miles east, and also the Pike County gas field, in western Illinois.

The Crawford County District.—The Central, or Crawford County, district, with Robinson as the commercial center, is the largest in area and has made the highest output, as 100,000 barrels a day were produced in 1907. It occupies the greater part of Crawford county, with a slight overlap into Jasper county on the west, and extends to Indiana on the east. The great width of over 20 miles that the field seems to have is undoubtedly due to a cross anticline that extends into Sullivan county, Ind.

The main producing horizon occurs at about 900 ft. and is known as the Robinson sand. It ranges from 8 to 30 ft. in thickness, averaging 20 ft., and the wells come in with an average of about 30 barrels per day, ranging from 5 to 100 and occasionally 200 to 500 barrels. A sand known as the Kibbe occurs at 750 ft. in the northern part of the county, and deeper sands occur at 1,100 and 1,400 ft. (the latter resting on the "big lime") over portions of the field, but they are narrower and less reliable than the Robinson sand.

The district was opened in February, 1906, by the discovery of the rich Shire pool, near Oblong, as the result of wild-catting started by a salted well. The latter opened the field at least a year earlier than would have resulted from conservatively edging southward from the Casey district. Four very active years of drilling followed, that rapidly enlarged the field, and for the past six years it has been a consistent producer, as the average well came in at 22 to 28 barrels. There has been considerable over-drilling, especially where the early wells were large, and the wells have therefore declined to a very moderate production. The daily output of the district is about 12,000 to 15,000 barrels.

A local, but unsuccessful, company drilled several tests about Robinson in 1901 to 1903, that found trifling amounts of oil and gas. Developments since then show that some of the holes were located in the pool, and had intelligent advice been secured the efforts would have been rewarded with huge profits. This experience of the pioneers failing where later followers have reaped generous rewards has been repeatedly duplicated in Illinois. Within the past few months town-lot wells are being drilled in Robinson, which was condemned by the earlier oil men from the neighboring dry holes. Another very reliable pool, where 42 good wells were consecutively brought in on one lease, was discovered 18 months ago in the heavily condemned territory 3 miles north of Robinson.

The Lawrence County District.—Lawrence county, at the southern end of the field, is probably the richest producer of high-grade oil of any county in the world. A few pools have attained a larger initial production, but the nine sands and their great staying power have enabled Lawrence county to maintain a daily production of 40,000 to 60,000 barrels for the past seven years. Yet the first three tests were dry holes in what has since proved to be the heart of the pool. The old oil operators were afraid of it and promptly sold out in its early development, as there was no gas. It occupies the greater part of Lawrence county, with a small extension to Allendale on the south, in Wabash county. Bridgeport is the commercial center, where the field was opened in 1906 with town-lot wells. The latter are still profitable producers, which is one of the rare exceptions where such excessive well density has paid.

The development of the field has been steady and consistent, as the depth and high cost of drilling have discouraged overdrilling. The

first wells were opened in the Bridgeport sand at about 900 ft., which came in at 20 to 100 barrels; later, other lenses of Bridgeport sands were developed at 780 and 1,000 ft. Later drilling brought in the Buchanan sand at 1,300 to 1,400 ft., the "Green Oil" at 1,525 to 1,575; the Kirkwood at 1,590 to 1,625; the Tracy at 1,700 to 1,790, which has produced wells of 1,000 to 2,000 barrels; and the McCloskey at 1,840 to 1,900 ft. The latter is an oölitic limestone, the St. Genevieve, in the upper portion of the Mississippi lime, and is very rich, as it has produced 2,000 to 4,000 barrel wells; the oil, like the Tracy sand, has a gravity of 37° to 39.5° B. Recent developments have brought in the Richey sand, at 550 ft., that was overlooked in the earlier work.

Three to five of the above sands are often found productive on one lease, to each of which separate wells were formerly drilled. In recent practice, one well is made to serve several sands by perforating the casing at each horizon.

The Allendale pool, in Wabash county, was discovered in September, 1912. Only one sand has thus far been found, at 1,500 ft., which produces a 37° gravity oil, but with no gas. The wells come in at 20 to 800 barrels, but decline rapidly, and the pool is small and spotted.

The daily output of the Lawrence County district ranges from 42,000 to 50,000 barrels.

Description of the Western Illinois Field

The Western Illinois oil field is so much younger than the Eastern field, with two minor exceptions, that the output is much less, although the sands are essentially the same and also the grade of oil. The developments of the next five years will probably witness a large increase in the output, and ultimately it will probably surpass the eastern side of the basin. For the western flank is about three times as large as the eastern limb of the Illinois basin and the anticlines are more numerous. It will develop less rapidly, however, as it lacks the simplicity of the Eastern field, especially if prospected in the erratic, unscientific method usually pursued.

Marion County.—In sinking a coal shaft 2 miles north of Centralia, in 1908, oil seeped in at 600 ft. from a small fault, which stimulated drilling in the vicinity. A strong north-and-south anticline at this place had been previously mapped by the Illinois Geological Survey, but, as usual, this was ignored by the oil men. Several 5-barrel to 20-barrel wells were brought in near the shaft in the Dykstra, or 600-ft., sand.

In the following year two light wells were brought in 2 miles east of Centralia in the 1,600-ft., or Benoist, sand and 75-barrel wells since then; the log of one of these, the Langewich well, is given on a preceding page. In the Dunn well, 2 miles northwest of Centralia, the oil was found in the Stein, or 1,400-ft., sand. Another oil sand, though light, was reported

at 1,300 ft. in the Wireback well that was drilled last fall $2\frac{1}{2}$ miles east of Centralia, in which the Benoist sand was found at 1,685 ft.; this is unusually low and explains why the oil was light and the salt water heavy, as it was too far down on the anticline (too far east) to expect oil. It is difficult to obtain reliable information about the Centralia pool and it has the earmarks of being held back until more leases are obtained.

Prospecting for the extension of the Centralia pool in 1909 brought in a good well at Sandoval, about 6 miles north on the same anticline, and in the following year a rich pool was drilled in. A sand was struck at about 1,400 ft. (the Stein) that came in at 25 to 50 barrels and at 1,550 ft. the Benoist sand came in with wells of 50 to 400 barrels and averaged 110 barrels per day. As the Benoist sand is 20 to 40 ft. thick and has great staying power, the wells are all sunk to this horizon and production in this field is the highest priced in the State. There is plenty of gas in both sands, and the No. 6 coal seam is found at 600 ft. The Benoist sand rests on the "big," or Mississippi, lime.

The Centralia district is connected by a 4-in. and the Sandoval district by a 6-in. pipe line with the Ohio Oil Co. lines, while Sandoval also has a 4-in. independent line to loading racks on the B. & O. R. R.

Marion county has the highest record in the State for initial production, as the U. S. Geological Survey statistics⁵ for 1910 show that 60 wells came in with an average daily output of 110.6 barrels; Lawrence county is second, at 102.7 barrels in 1910 and 105 barrels in 1912, when the very rich McCloskey sand was discovered.

Clinton County.—In prospecting in 1910 along an anticline near Carlyle, to which attention has been drawn by the Illinois Geological Survey, a 100-barrel well was brought in at 1,020 ft. after three previous failures. Hundreds of oil men rushed in and bought up leases in every direction, paying \$5 to \$10 an acre 10 miles distant up to \$250 an acre near the discovery well. A rich pool was rapidly developed and the wells came in at 20 to 1,000 barrels per day, with plenty of gas. Another sand at 800 ft. that was overlooked in the first rush has since been developed and came in at 10 to 30 barrels.

The Carlyle pool is connected by a 6-in. pipe line with the Ohio Oil Co. lines.

The 123 wells brought in during 1911 in Clinton county came in with an average daily output of 95 barrels, the highest in the State that year, and which has only been exceeded by the adjoining Marion county at 110.6 barrels (the highest in the State), and by Lawrence county (eastern Illinois) at 105 barrels in 1912.

Bond County.—Drilling at Greenville in 1910 developed a gas sand at 950 ft. that has since furnished light and heat for that town, the wells coming in at 2,500,000 cu. ft. per day.

⁵ *Production of Petroleum in 1912*, U. S. Geological Survey, p. 68 (1913).

Subsequent drilling at Old Ripley, 10 miles west of Greenville, found oil at about 2,000 ft., in the Devonian formation. The sand was very thin, or about 2 ft., and the wells were so small, or 5 to 25 barrels, that they were abandoned, as oil was then selling at 60c.

Montgomery County.—The first commercial gas and oil found in Illinois were developed at Litchfield, at about 600 ft., in 1882. The gas was discovered in drilling for a lower coal seam and it supplied the town for several years. Further drilling in 1886 found several small oil wells that produced until 1903 a lubricating oil that sold for \$5 a barrel. In drilling for coal at Butler, 8 miles east of Litchfield, in 1907, gas was discovered at 600 ft., and two miles north a similar heavy lubricating oil was found in the same sand, but developments were stopped by litigation.

Macoupin County.—Drilling at Carlinville in 1909 found gas and a little oil (2 to 5 barrels) at 400 ft. in the same sand as at Litchfield and Butler, that has since supplied that town. On drilling half a mile east of the gas wells last summer wells of 40 to 100 barrels were brought in from the same sand and a local pipe line was built to the field.

Pike County.—In drilling for water near Pittsfield, in 1890, gas was struck at 168 ft. in the dolomitic Niagara limestone, but no use was made of it until 1905. Since then a field 10 miles long by 4 miles wide has been developed along an anticline and nearly every farm has its own gas well.

Randolph County.—In drilling a water well at Sparta, in 1887, a strong flow of gas at 350 lb. pressure was struck at 900 ft. that supplied that town for over 20 years from some 20 wells. In 1906 oil was found in several wells over a small area on drilling about half a mile northeast of the gas field. While one well came in at 100 barrels, most of them were small, and shipments have about ceased. An investigation of the well records by the Illinois Geological Survey shows that the wells were drilled on a local "wart" or very small dome in a synclinal basin, which clearly explains the large number of dry holes that surrounded the small oil pool.

Morgan County.—Small amounts of oil and gas were found near Jacksonville, in 1910, at 300 ft., which is the most northerly occurrence that suggests commercial value. As the sands thin and die out going northward, the prospects are not encouraging in northern Illinois.

Madison County.—At Peters, 10 miles northeast of St. Louis, excellent oil was found in 1905, in the Devonian, at about 1,300 ft., but as the sand was very thin the wells were too small to pay.

St. Clair County.—At Smithton, several small oil wells were brought in during 1911, at 150 ft., at the base of the coal measures, along an anticline that extends northeast through O'Fallon. As the oil sand outcrops a few miles southwest it is surprising that any oil was found, but prospecting northeastwardly along the descending axis of the anticline should be successful when deep enough to be beyond the leakage zone.

At Marissa, gas and oil were noticed in 1895 in drilling a water well, which led to further drilling in 1912. Two good gas wells were brought in at 600 ft. As the neighboring coal mines show a strong, wide anticline, a field will probably be opened in the vicinity.

Jackson County.—A small gas well was brought in near Campbell Hill at 580 ft. in 1912 that led to considerable drilling in the vicinity; a little oil was found, but thus far not in paying quantities. This county is so far south as to be within the discouraging zone of faulting caused by the Ozark uplift and hence there is grave danger that the oil and gas have escaped, at least from the large pools that undoubtedly once existed.

Quality of the Oil

Over 92 per cent. of the Illinois oils grade above 30° B., which is the commercial line between low and high gravity oil. The Flat Rock oil is about 22° gravity, and occasionally oil in the Robinson and Casey sands grades at 28° B., on which there was formerly a dockage of 8c. a barrel; but for over a year all grades of Illinois oil have commanded the same price.

Broadly, the gravity of the oil improves with depth in Illinois, as the oils from the shallow sands in Clark, Crawford, and Macoupin counties are relatively heavy, or from 28° to 32° B., while the oils from the deeper sands in Lawrence, Marion, and Clinton counties will range from 34° to 39.5° gravity.

The color of the Illinois oils is very dark green to dark brown, except the deep Casey oil, which is light green and high in paraffine.

The Illinois oils have a paraffine base, except in the Bridgeport and Flat Rock sands, which have as much as 5 per cent. of asphalt.

The oils are "sweet," or essentially free from sulphur, as the latter averages 0.2 per cent., ranging from 0.1 to 0.5 per cent.

The gasoline content, like the gravity, improves with depth, as the Clark County or shallow oils have 8 to 18 per cent., the Crawford county oils have 8 to 20 per cent., and the deep Lawrence county oils have 10 to 23 per cent., while the heavy, asphaltic Flat Rock oil has only 1 to 4 per cent. The average gasoline content is about 15 per cent.

The kerosene content ranges from 25 to 38 per cent. and averages about 31 per cent.

The Oil Sands

All but two of the so-called "sands" in Illinois are sandstones, and, on account of the dip of the basin, the depth of the same sand will vary according to the distance from its outcrop. On the eastern side of the State the depth increases toward the west, and on the west side, which is much the longer flank, the depth increases eastwardly at 10 to 15 ft. per

mile. There is a secondary general dip of the northern three-fourths of the State to the south, due to the dip of the axis of the spoon-shaped basin to about Jefferson county, when the axial dip reverses, with a much sharper dip to the north, and the southern, or stub, end of the basin comprises about a fourth of the State.

When not carrying oil or gas, most of the Illinois sands are saturated with salt water and dry sands are local. Even on the anticlines, if a sand is thick, oil and gas frequently occur in the upper portion, while the lower part of the sand may be full of salt water.

Two of the oil-bearing sandstones are quite persistent and fairly uniform over large areas of Illinois, while the other sandstones are lenses of greater or less magnitude and consequently are more or less uncertain. The lenticular, uncertain character of most of the oil sands harmonizes with the other formations of the Illinois coal measures, as, barring No. 6 coal seam and one or two limestones, the limestones, coals, sandstones, and shales fluctuate greatly in persistency and thickness—in fact, to such an extent that it is often difficult to correlate reliable logs of different wells on the same lease.

Basal Sand.—The most persistent and easily recognized “sand” rests on the “big lime” and was formerly known as the Ferruginous sandstone. It generally ranges from 20 to 50 ft. in thickness, is usually coarse and porous and is the Sparta gas and oil sand, the Benoist sand of Sandoval and Centralia, the “deep” sand of Crawford county, and the Tracey sand of Lawrence county. The wells usually come in at 50 to 150 barrels, ranging from 5 to 1,500 barrels per day.

Robinson Sand.—About 200 to 300 ft. above the basal, or Ferruginous, sandstone occurs another sandstone that is very persistent and more frequently carries oil or gas. This is the “big-pay” sand at Carlyle, the gas sand at Greenville, the oil and gas sand at Litchfield, Butler, and Carlinville on the west side, and is the equivalent of the Robinson sand in Crawford county and the Kirkwood in Lawrence county. This sand is 5 to 50 ft. thick, averaging 15 to 25 ft., is usually quite uniform, rather coarse, and stands up fairly well. It produces wells of 20 to 100 barrels, with extremes of 5 to 1,500 barrels per day.

Casey Sand.—The 400-foot, or Casey, sand in Clark county is a dolomitic limestone, which is unusual in the coal measures, as the limestones are seldom magnesian. The dolomitization has rendered it porous and therefore capable of serving as a reservoir. The wells come in at 17 to 30 barrels, with extremes of 5 to 200 barrels per day.

McCloskey Sand.—The McCloskey sand in Lawrence county is an oolitic horizon (the St. Genevieve) in the upper portion of the Mississippian limestone. Its porosity is due to the spaces between the spherical granules of which this formation is composed, and as these are large and excessively numerous it produces very large wells, or from 50 to 4,000

barrels per day, the largest thus far found in Illinois. The excessive, coarse porosity that is responsible for such large wells also results in the oil soon draining out; hence the wells soon decline to a modest output.

Drilling Features

Contracting.—Drilling is generally done by contract, as few companies operate their own tools. The contractors furnish the equipment and deliver a completed hole on a footage basis, also including the casing if desired, though usually the latter is supplied by the producer. "Machines," or portable rigs, are generally employed up to 1,200 ft., although some contractors use derricks for the 400-ft. wells. The derrick is used when over 1,200 ft. and is usually wood, though a few composite, or "turn-buckle," rigs are employed, and rarely the all-steel derrick. The derricks are erected by special rig-builders at an expense to the operator of about \$700, as they are usually left on completion of the well for subsequent cleaning and pulling.

As the fire clays under the coal seams frequently cave and as there are several water sands above the oil, a large amount of casing is required. For wells not over 1,000 to 1,200 ft., the hole is started with 10-in. or 12-in. drive pipe, followed by 8½-in. casing, and the oil sand is entered with 6½-in. casing; deeper holes are started with 16-in. drive pipe and sometimes finished with 5½-in. casing. The gas pressure is usually so light that it is imperative to enter the oil sand with a "dry" hole; that is, the overlying waters must be shut off by casing. For if the hole is full of water a rich oil sand might be passed through and reported as a trace, or a "showing," from the hydrostatic pressure holding back the oil in the sand, or "drowning the oil." This is what delayed the opening up of the Casey district some 40 years. When the hole caves, under-reaming bits are used, which permits the casing to follow closely after the tools, thus preventing them from getting buried under the cavings. Crooked holes occasionally give trouble from the tools being diverted by striking a "nigger-head," or the tough, hard, spathic iron concretions that occur in some of the shales.

The prices paid for drilling in the Casey and Crawford districts, where machines are used and there is no caving, range from 70c. to 80c. per foot. In Lawrence county, the rates are 90c. to \$1 to the Bridgeport sand (900 ft.) and \$1.30 to \$1.50 to the deeper sands, where more or less under-reaming is required. In western Illinois, where there is more or less caving, the rates are \$1.15 to \$1.25 per foot up to 1,100 ft. and \$1.40 to \$1.50 to the 1,600-ft. sand. These rates are subject to modifications according to haulage and whether the operator furnishes the fuel and water. For cleaning out the hole, etc., after shooting, the charges are \$20 to \$25 a day.

Drillers are paid \$5 and back-hands or tool dressers \$4 per 12-hr. shift, with two men "on tower" while drilling and both crews, or four men, when setting or pulling casing.

For "wild-catting," or drilling at a distance from proven fields, the charges are higher, or usually \$2 per foot, from the inconveniences and greater delays that result in isolated work.

The time required to complete a hole to the 400-ft. sand ranges from 4 to 8 days, as from 30 to 100 ft. are drilled per shift. It takes 14 to 24 days to drill to the 1,000-ft. sand and 30 to 60 days to the 1,600-ft. and 1,800-ft. sands.

Shooting.—Shooting has been the salvation of many Illinois wells that made only a feeble showing until shot, after which they came in at 25 to 50 barrels. Formerly heavy shots, or from 100 to 200 quarts of nitro-glycerine, were used, but so much sand trouble resulted that 20 to 80 quarts are now employed, depending on the thickness and softness of the sand. Several feet of bottom anchor are used where there is danger of breaking into water below.

The shots are fired with a battery, on account of caving trouble, and the charge is placed and wired before the 6½-in. casing is pulled.

Well Density

Acreage Per Well.—The usual practice in Illinois is to allow about 5 acres to each well, or they are spaced 400 to 425 ft. With 10 to 25 ft. of fine-grained sand this insures a slow decline and a reasonably long life. When the sands are thin, or so deep as to make the cost excessive, a well is given 7 to 10 acres, or they are spaced 500 to 600 ft. apart.

In the shallow sands, where the cost is light, the wells have frequently been placed much closer, or with only 1 to 3 acres to the well. While such over-drilling temporarily gives a large output, it results in a rapid decline of the wells and sinks an excessive amount of capital in a needless number of short-lived wells. When the leases are very small, or under 20 acres, or where very large wells are brought in, the rivalry among neighboring operators to secure as much of the oil as possible usually results in over-drilling, to the great advantage of the contractors and supply men, but at the expense of the ultimate profits of the operators. Town-lot wells seldom pay, as four to eight wells are drilled on an acre, which usually exhausts the oil so quickly that the investment is not always repaid. The wells in the town of Bridgeport are an unusual exception, as they have proved quite profitable, which is partly due to the staying power of the Bridgeport sand and partly to there being several sands. For the same reason the 1-acre school leases in Lawrence county, on which two to four wells have been drilled, have paid well.

As an extreme case of excessive well density in a normal field, yet one

that has proved highly profitable, a sketch, Fig. 4, is shown of the McBride $\frac{1}{2}$ -acre tract, 4 miles north of Bridgeport, that has been operated since Jan. 1, 1908, by the Silurian Oil Co. It is surrounded by four leases that call for 22 offset wells and to-day there are 30 wells on an area of less than 2 acres. Yet in 2,190 days it produced 200,000 barrels and to-day is yielding 55 barrels from 8 wells; this gives an average yield to date of 25,000 barrels per well, without including the flush production of 1907. Five wells are in the Bridgeport sand and three in the Buchanan, which are very thick, for 35 to 45 ft., which explains their

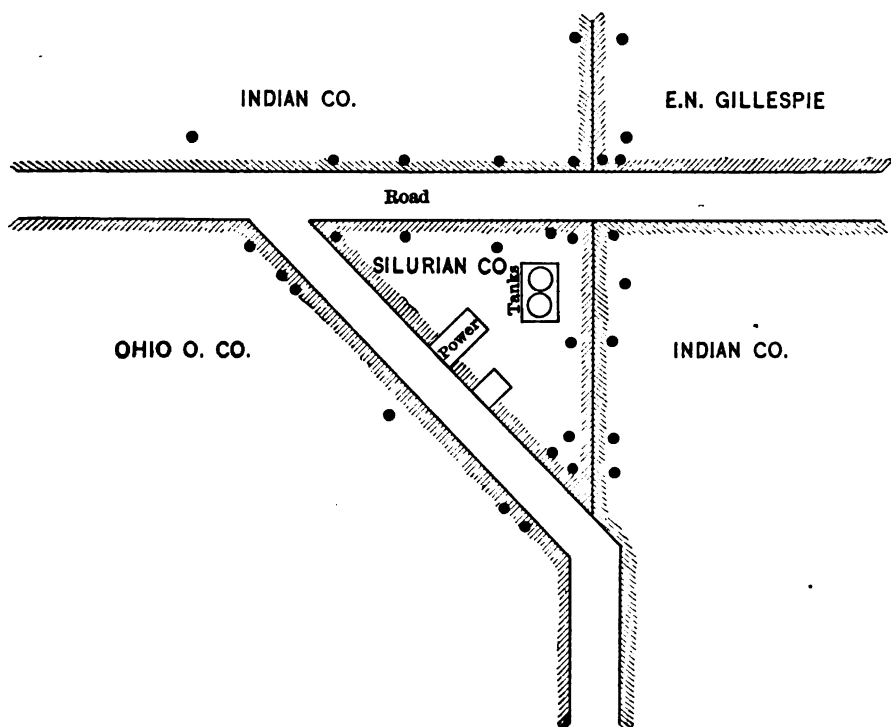


FIG. 4.—THIRTY WELLS ON TWO ACRES, NEAR BRIDGEPORT, ILL.

ability to stand such a very heavy drain; they came in at 35 to 300 barrels. The sales from a gas well in the Kirkwood sand (No. 9) that came in at 5,000,000 ft. exceed \$20,000.

To illustrate the richness of this territory when not overtaxed by excessive well density, a neighboring 40-acre lease produced 1,069,347 barrels in about 2,300 days from 17 wells in three sands, or over 60,000 barrels per well, and the wells are tanking 445 barrels per diem.

Offset Wells.—The unwritten law to locate line or boundary wells 200 ft. back from the property line has been quite well lived up to, and

there have been few line fights, outside of town-lot and similar small leases where disputes are unavoidable. The offset wells have usually been promptly drilled, especially in the early days when the wells were larger, and the drill is generally kept busy in developing a lease until all the line or boundary wells are completed. The inside locations are subsequently drilled deliberately and on many 40-acre and 80-acre tracts they have been omitted, as it is found that the line wells will eventually drain the interior of the lease.

Plant Equipment

Casing.—Casing is the heaviest investment on a lease and ranges from \$1 to \$1.50 a foot up to 1,000 ft., and from \$1.50 to \$2.50 in deep holes, or 1,400 to 1,900 ft. Formerly all the casing was left in a hole, but the larger sizes, or above 8½-in., are now pulled and re-used. Casing requirements vary with every hole, but the following approximations cover most cases: A 1,000-ft. hole will require 50 to 200 ft. of 10-in. to reach the rock; from 200 to 600 ft. of 8½-in. to shut off the upper waters; from 800 to 1,000 ft. of 6½-in. to reach the oil sand. A 1,600-ft. hole will require 50 to 150 ft. of 16-in.; from 200 to 600 ft. of 12½-in.; from 600 to 1,000 ft. of 10-in.; from 1,000 to 1,400 ft. of 8½-in.; and from 1,400 to 1,600 ft. of 6½-in. Occasionally a string of 5⅞-in. casing is employed to finish a hole, if water sands or the under-reaming have been excessive. The price of casing, which is subject to great variation, is about as follows, to which 5 to 20 per cent. should be added for freight, unloading, hauling, etc., to arrive at the cost at the well.

	Per Foot
16-in., $\frac{5}{16}$ thick (drive pipe).....	\$3.25
12½-in., 36 lb. per foot (drive pipe).....	1.20
10-in., 32 lb. per foot.....	0.85
8½-in., 24 lb. per foot.....	0.63
6½-in., 17 lb. per foot.....	0.45
5⅞-in., 13 lb. per foot.....	0.35

Derricks.—If a well is drilled with a derrick, it is usually allowed to remain for subsequent cleaning and pulling; otherwise portable pulling machines are employed when a well needs attention. As the high drilling derricks (64 to 84 ft.)^r are occasionally blown over, they are often replaced by 40-ft. pulling derricks, which are free from this danger.

Pumping Outfit.—The wells are equipped with 2-in. pumps and tubing as the salt water is seldom so heavy as to require a 3-in. pump. The pumps are connected to iron or wooden jacks that are operated by pull rods from a central power station driven by a 25 to 35 h.p. gas engine. The engine and power are housed in an inexpensive corrugated-iron building and from 4 to 24 wells are pumped from one power.

The 2-in. lead lines from the wells to the tank and the 2-in. gas lines that take off the casing-head gas aggregate 4 to 6 miles on a completely drilled 80-acre lease (15 wells). They should be buried over a foot to prevent freezing in winter, or tearing open from expansion in summer, and to be safe from breakage from the farmer's plow.

Tankage.—Liberal tankage is required, compared with Eastern fields, as from 2 to 10 tanks (wood) of 250 barrels capacity are required for a 100-acre lease. A "gun-barrel," or siphon, tank is also required for settling out the salt water, although small amounts of water, that only settles out in cold weather after thinning by steaming, are carried over into the shipping tanks. The tanks are housed in framed buildings to reduce the evaporation losses in summer.

A small boiler house located at least 100 yd. distant, so as not to fire the tank vapors, furnishes the steam for preparing the oil for shipping in cold weather and for operating the pipe-line pump.

Formerly the "B. S.," or the emulsion of oil, water, air, and sediment formed by improper pumping, was run off into special sludge ponds and burned, a very wasteful practice that is still quite prevalent. On the better-equipped leases the B. S. is treated in a sludge tank, by which considerable good oil is recovered. If the services of a capable chemist were retained practically all the B. S. could be recovered as good oil.

Market Features

Oil Prices.—Illinois is the only field that has gone through its "flush," or initial period of heavy production, and secured such high prices as 60c. to 68c. per barrel. Other fields have had to accept 3c. to 30c. during the inevitable heart-breaking interim that intervenes between overflowing field-tanks and the subsequent bringing in of sufficient pipe lines. This great advantage is due to its proximity to St. Louis and Chicago, where the oil would have found a welcome market at 40c. to 50c. in competition with coal.

When shipments began in June, 1905, the oil sold at 60c. and advanced to 83c. by June, 1906; after which it declined, remaining at 68c. throughout 1908 and again reached the low point of 60c. in October, 1909, where it remained until May, 1911. Since then, the price has continuously advanced and to-day, Jan. 29, 1914, it is selling at \$1.45 in the open market, as paid by the Ohio Oil Co. (branch of the Standard Oil Co.). Independent buyers pay a premium of 5c. to 20c. a barrel, as on equal terms most producers prefer to sell to the Standard Oil Co., and it is only by giving a premium that they are able to divert it from the principal buyer. Formerly there was a discount of 8c. per barrel for oil below 30° B. gravity, but for the past year all grades have sold at the same price.

The oil is purchased in the producer's tanks after gauging and run off through the buyer's pipe line, with an extra allowance of 1c. to 2c. per barrel for the steam furnished to operate the buyer's pump that forces the oil into the pipe line. A dockage of 3 per cent. is usually made to allow for losses due to salt water, B. S., evaporation, leakage, etc. A "credit balance" ticket is made out by the buyer three days after the oil is shipped for the net barrels due the operator after deducting royalty and dockage. This ticket can be cashed at the market price for oil ruling that day, or it can be held to be cashed later, if higher prices are expected. Before the oil markets were stabilized by the present powerful interests, there were most violent fluctuations in oil prices that out-rivaled Wall Street stocks, and operators who had the working capital held their tickets for the winter market, when prices generally ruled higher.

The unprecedented prosperity of the oil industry, the enormous increase in the demand for high-grade oils (paraffine base) to meet the rapidly expanding uses of gasoline and lubricants, and the almost complete exhaustion of the Illinois surplus stocks assure much higher prices for Illinois oil in the not distant future.

Storage.—As the Illinois oil is of too high a grade to be sacrificed for fuel, the large buyers built steel tankage to take care of the surplus oil before there were sufficient pipe lines. About 1,000 tanks of 35,000 barrels capacity were erected throughout the field, which contained about 33,000,000 barrels of oil in 1910. As the demand for Illinois oil has been much greater than the supply for the past three years, this surplus has been heavily drawn on to meet the deficit and to-day there remains about 5,500,000 barrels.

Pipe Lines.—No oil field has been as well taken care of in being promptly and completely equipped with pipe lines as Illinois. The magnitude of the field, the richness of the wells, and the high grade of the oil caused the pipe-line companies to go to enormous outlays to run the oil from the tanks of any and every producer and their total daily shipping capacity now exceeds 140,000 barrels.

The Ohio Oil Co. controls most of the lines, having a complete series of gathering lines throughout all the districts in eastern and western Illinois. It has an 8-in. line across the State to the Wood River refinery, 140 miles; two 8-in. lines to the Lima trunk line, 200 miles; and a 12-in. line to New York harbor, 900 miles. There is a pumping station at Bridgeport, in the Southern district, with 60,000 barrels capacity, that pumps the oil at 800 lb. pressure to Stoy, 25 miles, in the Central district. Another station at Stoy pumps the oil to Martinsville, 25 miles, in the Northern district, where all the oil from the Eastern field concentrates. From there it is distributed through the Eastern and Western pipe lines.

The Tidewater Co. has a 6-in. pipe line to the Atlantic seaboard and gathering lines throughout Crawford and Lawrence counties.

The Indian Refining Co. has gathering lines in Lawrence county, as also the Central and Sun Companies, which run to their refineries at Lawrenceville.

The Leader Pipe Line Co. has gathering lines in Clark county for its refinery at Casey, from which the surplus oil is shipped to Pennsylvania refiners by tank cars. The Robinson refinery has gathering lines in Crawford county.

Railroads.—The five railroads that tap the Eastern Illinois field moved large quantities of oil before the pipe lines were completed and they still transport considerable oil for the smaller refineries from loading racks distributed throughout the field. While most of the refiners own their own tank cars, tank-car companies rent out hundreds of cars to the smaller shippers, at usually \$1 per day.

The Western field is well equipped with railroads, which have thus far handled most of the oil.

Refineries.—Although most of the Illinois crude oil is piped to the large refineries on the Atlantic seaboard, there are nine refineries in Illinois that rely on the local output, of which six are on the eastern and three on the western side of the State.

At Lawrenceville is the large Indian refinery, of 9,000 barrels daily capacity, that makes a full line of products (gasoline, kerosene, lubricants, paraffine, etc.); the Central, with a capacity of 3,000 barrels, that co-operates with an asphalt plant, and the Sun Co. operates a 400-barrel skimming plant that makes gasoline and fuel oil.

There is a 600-barrel refinery at Robinson, a 600-barrel refinery at Casey, and a refinery at Chicago, the latter relying on tank-car shipments.

In western Illinois there are two refineries at East St. Louis that depend on tank-car shipments and a large refinery at Wood River, 20 miles from St. Louis, of 10,000 barrels capacity, owned by the Indiana Standard Oil Co.

Production Prices

The value of production and of operating leases has greatly increased as oil prices have advanced, and developed properties are almost unpurchasable. When oil was selling at 60c. production sold at \$300 to \$400 per barrel of daily output. When oil reached \$1 it advanced to \$800 to \$1,000. At present, with oil at \$1.45 to \$1.65, production is held at \$1,250 to \$1,500 per barrel, according to the age and size of the wells, locations undrilled, royalty, condition of plant, equipment, and other modifying operating details. These prices are based on "settled" production, or after the flush period, or when the wells are about a year old.

Pennsylvania production, where the oil is higher grade, or 40° to 46° gravity, and sells for \$2.50 a barrel, is rated at \$2,000 to \$3,000 per barrel, although the wells yield only $\frac{1}{4}$ to $\frac{1}{2}$ barrel a day, and, like Illinois, it is almost impossible to induce a producer to sell at any price.

At these prices, the investment is expected to be returned in 1,000 to 1,200 days.

Royalties and Rentals

Eastern Field.—When the Eastern field was opened, the royalty was one-eighth of the production, or the customary rate in the older fields. When the field proved so rich, the competition among the oil men for leases resulted in the royalty advancing to one-sixth to one-fourth, and even three-eighths to one-half in extreme cases. The royalty on gas wells similarly increased from \$50 to \$100 per well annually to \$150 to \$200.

The rental to hold a lease good until a well is drilled was formerly 25c. to \$1 per acre annually, but lately \$1 to \$4 per acre is demanded, and then only 1 to 2 years allowed, compared with 5 to 10 years in the earlier leases.

Western Illinois.—In western Illinois the royalties and rentals are still reasonable, or one-eighth on the oil and \$50 to \$100 a year per gas well, while the leases can usually be carried 5 to 10 years without drilling on paying an annual rental of 50c. to \$1 per acre.

Cost of Oil Leases

The bonus, or cash price, paid to obtain an undeveloped lease varies greatly according to the distance from production, richness of the discovery well, and terms of the lease. Leases favorably situated on anticlines in an undeveloped territory go begging at a few dollars per acre until a discovery is made in the vicinity, when they quickly advance to \$25 to \$50 an acre if not over 5 miles distant, and to \$100 to \$350 an acre if within 2 miles. The discovery of a new pool promptly brings out hundreds of active, keen buyers from the old oil fields, who purchase at rapidly advancing prices every available lease for 7 to 15 miles in every direction. Leases 10 miles distant will bring \$5 to \$15 an acre from late comers after the nearer leases have been secured by the excited buyers. Within 30 days several million dollars will be invested in leases, of which 60 to 70 per cent. will probably prove unproductive. Yet pool after pool in Illinois (and elsewhere) has been the scene of repeated large investments in undeveloped oil leases around a discovery well by a horde of feverish buyers. While many speculators rush in to every new discovery, most of the buyers are old, seasoned oil operators who can no more resist the fascination of a "wild-cat" well that has made good than a tenderfoot can escape the lure of a mining boom in a gold camp.

Operating Expenses

The field expenses for producing oil in Illinois, including labor and upkeep, vary greatly according to size and conditions of the lease, the size of the wells, and amount of salt water. Excessive salt water requires the wells to be pumped by night as well as on the day shift and Sundays, whereas with little or no salt water day pumping suffices. As a pumper is generally required to every lease, whether it contains 40 or 160 acres, it makes the labor cost per barrel excessively heavy on the small leases, especially if operated double-shift. Similarly, if the wells are small, the labor costs per barrel are heavy compared with large wells.

The extra labor required for cleaning the wells, repacking, etc., will vary according to the condition of the wells, as old wells that have not been over-shot may have to be pulled only once or twice a year, whereas sand troubles caused by over-shooting may constantly require a pulling crew. The pumpers are paid \$65 to \$75 a month and live on the lease in a small cottage provided by the operators.

When an operator has several leases in a district a "farm boss" is employed, who keeps a crew of two to four "rousters" steadily employed in looking after the wells. The rousters are paid \$60 to \$70 a month and a farm boss from \$100 to \$115.

The usual field expense of putting the oil into the tanks ready for the buyer will range from 3c. to 7c. a barrel on leases of 80 to 160 acres, with large wells, or over 20 barrels per day; with smaller leases, or 20 to 80 acres, or where the wells yield 5 to 15 barrels, the cost will range from 7c. to 15c.; with very small wells, or 2 to 5 barrels, the cost increases to 15c. to 30c. per barrel.

When the leases are very small and in town-lot production, one pumper is usually put in charge of several leases and thus the labor cost is brought down to a reasonable amount.

Operating Profits

The net operating profits from Illinois oil production vary greatly according to the size of the wells, their staying power, depth and number of sands, size and age of lease, royalty, oil market, and other local conditions, besides depending on the capital invested for leases, drilling, plant, etc., and the individual operating costs.

The Illinois oil wells usually repay their cost in 30 to 90 days when not over 1,100 ft. deep and in 60 to 180 days in the deep sands, or 1,200 to 1,900 ft. Occasional large wells repay their cost in 5 to 20 days, while very small wells may take a year.

The capital for the acquisition, development, and equipment of an undeveloped property in a proven field is usually returned in the first year and will show the following great range: The lease will cost from \$25

to \$300 an acre, according to the distance from and richness of the discovery well; the wells will cost \$1,000 to \$8,000 each, according to depth; the plant investment for power, rodding, piping, tankage, boilers, buildings, etc., will range from \$2,000 to \$10,000, according to the size of the lease and local conditions. Yet many Illinois leases have returned the total capital invested in 4 to 16 months, when oil sold at 60c. to 80c. a barrel.

The net operating profits at present prices of oil (\$1.45 to \$1.60), after deducting the royalty and operating expenses, usually range from \$1 to \$1.30 a barrel. Small leases (5 to 20 acres), town-lot wells, very small wells (1 to 3 barrels), or excessive royalties (three-eighths to one-half), may reduce the net profit per barrel to 50c. to 80c.

A fair example of operating profits is shown in the following 120-acre lease in the shallow Casey field, where the wells came in at 10 to 50 barrels, seldom at 100 barrels per day, and the oil sold for 60c. to 82c. a barrel: The oil royalty was one-eighth, the cost of the lease was \$100 an acre, and the wells cost about \$1,000 complete, as they were in the 400-ft. sand. In the first year the investment of \$43,000 was returned; in the following 54 months, when the property was sold, the profits ranged from 6 to 40 per cent. per month.

Gas Developments

Paucity Due to Leakage.—Although Illinois has proved an exceptionally rich oil producer, the gas output is disappointing. While sufficient gas is usually produced to operate with and supply the neighboring towns, not enough has been developed to warrant piping to the metropolitan markets of St. Louis and Chicago.

This paucity seems to be due to faulting, as minor faults occur that apparently have allowed more or less of the gas to escape. That the gas has been only partly preserved is shown not only by the small size of the wells, but usually by the absence of the heavy pressures that occur in other fields.

That faulting is responsible for the deficiency is verified at Centralia, where there is no gas; yet 6 miles north, at Sandoval, on the same anticline, the same sands are rich in gas; but at Centralia a fault is shown by the coal mines that has not only vented the gas, but is permitting some of the viscous oil to escape into the coal shafts.

The northern parts of the fields usually have more gas than the southern portions, which harmonizes with the faulting theory, as the faults are more numerous and heavier toward the south, from the closer proximity to the Ozark uplift.

That the oil has been so slightly affected by the faults—at least, in the lower sands—is due to its viscous nature and strong capillary adhesion as compared with the great elasticity and extreme mobility of the gas.

For the gas can move and more or less escape through minute fractures caused by faulting that are more or less impervious to the viscous oil.

Gushers.—On the opening of a new pool the gas pressure is often sufficient to cause the wells to flow intermittently for a short time, or until the pressure is relieved. The gusher type is exceptional, however, and most wells have to be pumped from the outset.

Size of Gas Wells.—Gas wells usually come in at 200,000 to 1,000,000 cu. ft. a day, rarely at 2,000,000 to 5,000,000, which for a new field is modest. There are comparatively few "dry" gas wells, as the gas generally occurs with the oil; hence when a "gasser" is found the oil usually occurs in the immediate vicinity.

Values.—The towns are supplied by local gas companies at 20c. to 30c. per 1,000 cu. ft. for domestic purposes and the gas is metered. The companies either have their own wells, or buy gas from oil producers, who have an excess, at 5c. to 10c. per 1,000 cu. ft. at the wells.

Drillers are furnished gas at a flat rate of \$5 a day for the boiler and lighting.

The value of the Illinois gas output in 1912 is estimated^a at \$616,467, or about 2 per cent. of the oil value, of which 1,236,162,000 cu. ft. was sold at an average of 23.62c. for domestic use and 5,603,318,000 ft. at 7.43c. for industrial purposes.

The actual production was much larger, as the oil operators continue to waste large quantities, especially in new pools and when a flowing well comes in.

Gasoline from Gas

Few Plants.—The Illinois oil field is so young and so lucrative that thus far little attention has been paid to gas gasoline. The old wells of Pennsylvania are so small and the casing-head gas is usually so rich that the gas-gasoline revenue often exceeds that from the oil, whereas in Illinois it amounts to only 5 to 10 per cent. of the oil revenue. As the Illinois wells decline and as the gas grows richer, more attention will be given to extracting the gasoline. Only six plants are operating in eastern Illinois and compound compressors are employed to liquefy the gasoline vapors that are contained in the gas. Two grades are made, or a high-grade 70° to 80° gravity gasoline by the low-pressure cylinder and a very high-grade 80° to 95° B. fluid by the high-pressure cylinder. Both grades are usually blended with naphtha to render them safe for shipping and to reduce the heavy volatilization loss. The 90° to 95° gravity fluid is so extremely volatile that if a cupful is tossed into the air on a warm day, it will volatilize before reaching the ground.

Yield.—The gasoline yield of Illinois gas ranges from 1.5 to 2.5 gal.

^a *Production of Natural Gas in 1912*, U. S. Geological Survey, p. 5 (1913).

per 1,000 cu. ft., and about 75 per cent. of the gas is available for returning to the gas main, if subsequently utilized.

Prices.—The gasoline is frequently retailed from the lease to automobiles at 12c. to 15c. a gallon, although large producers usually sell it to jobbers at 8c. to 10c. It has to be shipped in steel containers, as the loss and risk are too great to permit shipment in wooden barrels.

There is a keen demand for this exceptionally high-grade gasoline and the industry has a bright future that the Illinois operators should more generally appreciate.

Statistics

The data given in the following statistics were obtained from the annual *Mineral Statistics*, published by the U. S. Geological Survey, or from the *Oil City Derrick*.

Table II presents a summary of the wells drilled to Jan. 1, 1914, in Illinois, since the marked activity started in 1906. It shows a tremendous outburst of activity in 1907, when about 5,000 holes were drilled, since which there has been a steady, marked decline until 1913. The high prices for oil stimulated the drilling of many inside locations last year, where the sands had been more or less drained, which accounts for the smaller percentage of dry holes and the much lower average yield.

The average yield declined from 40.4 barrels in 1906 to 26.1 barrels in 1908, as the flush period passed in the very active shallow Crawford County district. The greater activity in the deeper but richer sands of Lawrence county reversed the usual retrogression that nearly all young fields exhibit and the average output of the new wells steadily increased up to 1912, when it averaged 67 barrels per well, which is exceptionally high for a high-grade field.

TABLE II.—*Summary of Illinois Drilling*

Year	Wells Drilled	Produced Oil		Initial Yield Barrels		Produced Gas		Dry	
		Number	Per Cent.	Total Yield	Daily Yield	Number	Per Cent.	Number	Per Cent.
1905	227	189	83.2			..	...	37	16.8
1906	3,283	2,793	85.1	113,012	40.4	..	...	490	14.9
1907	4,988	4,260	85.4	139,163	32.6	..	...	728	14.6
1908	3,574	3,019	84.5	78,960	26.1	..	...	555	15.5
1909	3,151	2,593	82.3	89,756	34.0	70	2.2	485	15.4
1910	2,149	1,681	78.2	93,256	55.8	75	3.4	393	18.3
1911	1,365	1,061	77.7	66,919	63.1	42	3.1	263	19.2
1912	1,260	980	77.8	65,686	66.6	23	1.8	257	20.4
1913	1,747	1,439	82.0	47,558	33.3	40	2.4	268	15.6
Totals	21,744	18,025	82.9	694,310	38.5			3,486	16.0

Table III gives the average initial daily yield of the wells in the producing counties of Illinois and gives an approximate idea of the relative richness of the sands. It also shows the activity in drilling inside wells in 1913 in all the counties, and the consequent drop-off in the yield.

TABLE III.—Average Initial Daily Yield of Illinois Wells by Counties

<i>Eastern Field.—Barrels</i>								
County	1906	1907	1908	1909	1910	1911	1912	1913
Clark.....	26.5	20.9	23.3	24.0	22.5	17.1	23.6	15.2
Coles.....	5.5	7.0	15.3	10.6	16.3	5.0	5.0	20.0
Crawford.....	66.1	34.2	23.5	25.5	27.8	26.6	23.1	17.1
Cumberland.....	29.9	26.0	9.8	24.3	12.5	16.7	19.0	12.0
Edgar.....	4.8	10.7	6.4	5.0				
Lawrence.....	50.6	49.2	36.2	61.5	102.7	86.8	10.5	54.0
Jasper.....				7.1	10.0	6.7		5.0
Wabash.....							50.0	39.0
<i>Western Field.—Barrels</i>								
Bond.....					25.0			
Clinton.....						95.0	32.2	10.0
Macoupin.....				5.0		3.5	3.0	55.0
Marion.....				37.2	110.6	91.5	27.7	23.0
Randolph.....				72.5				
State Average.....	40.5	32.7	26.2	34.6	55.5	63.0	67.0	33.3

TABLE IV.—Wells Drilled in Illinois in 1913

County	Wells Drilled	Produced Oil Per Cent.	Average Initial Daily Output Barrels	Dry Holes Per Cent
Clark (Casey).....	208	81	15.2	19
Clinton (Carlyle).....	19	74	10.0	26
Coles.....	6	67	20.0	33
Crawford (Robinson).....	689	80	17.1	20
Cumberland (Siggins).....	63	80	12.0	20
Lawrence (Bridgeport).....	667	89	54.0	11
Macoupin (Carlinville).....	9	33	55.0	67
Marion (Sandoval).....	22	95	23.0	5
Wabash (Allendale).....	49	50	39.0	50
Miscellaneous.....	15	13	12.0	87
Total.....	1,747	82	33.3	18

1,439 wells = 82.0 per cent. came in with an output of 33.3 barrels daily (average).

40 wells = 2.4 per cent. came in as gas wells.

268 wells = 15.6 per cent. came in as dry holes.

1,747 wells = 100.0 per cent. or the total wells completed in 1913.

Table IV shows that 1,747 wells were drilled in Illinois in 1913, of which 82 per cent. were oil producers that came in at an average of 33.3

barrels per day, and 2.4 per cent. were gas producers. It also shows that almost no prospecting was carried on, as only 15 holes were drilled outside of the producing fields. This insignificant effort to open up new territory was largely due to the great scarcity of drillers, who were attracted by the fierce boom in Oklahoma, where rates are higher and work was very plentiful.

Table V presents a summary of the drilling in 1913 in the States that exclusively produce high-grade oil (paraffine base). It shows their relative activity by the number of wells completed; and while the drill was unusually busy in all, it shows the extent of the unprecedented boom in Oklahoma, where over 9,000 holes were drilled. In spite of bringing in the very rich Cushing and Boston pools in Oklahoma, the average well shows a further decline to 46 barrels per day, although in the old nip-and-tuck race with Illinois for first honors in the highest average well, it leads the latter (33.3 barrels) by nearly as large a margin as Illinois with 67 barrels led Oklahoma in 1912, at 48.6 barrels.

With the exception of New York (where the average well came in at 1.8 barrels, which would be called a dry hole in Illinois), the record of 82 per cent. of the Illinois wells coming in as oil producers is the highest, as the other States range from 80 to 38 per cent.

TABLE V.—*Wells Drilled in the High-Grade Oil Fields in 1913*

State	Wells Drilled	Produced Oil		Initial Yield Barrels		Produced Gas		Dry	
		Num-ber	Per Cent.	Total	Daily Average	Num-ber	Per Cent.	Num-ber	Per Cent.
New York ...	510	443	87	807	1.8	45	9.0	22	4.0
Pennsylvania..	3,719	2,961	80	8,155	2.8	263	7.0	495	13.0
West Virginia..	2,072	1,290	62	34,287	26.6	453	22.0	329	16.0
Ohio.....	1,447	551	38	13,409	24.3	685	47.0	211	15.0
Indiana.....	310	212	68	7,361	34.7	12	4.0	86	28.0
Kentucky....	211	134	64	2,310	17.0	3	1.5	74	34.5
Illinois.....	1,747	1,439	82	47,558	33.3	40	2.4	268	15.6
Kansas.....	2,149	1,583	73	22,985	14.5	292	14.0	274	13.0
Oklahoma....	9,112	7,232	79	331,385	46.0	572	6.0	1,306	15.0

The richness of the Illinois fields is shown in Table V, as its high daily average production of 33.3 barrels exceeds all the other States (1.8 to 24.3 barrels) excepting Oklahoma, as previously mentioned, and Indiana (34.7 barrels); the latter State made an unprecedented record, as its wells usually come in at 10 to 20 barrels. The increase is due to the opening up of the new Sullivan County field, which adjoins Crawford county, Ill., and seems to be influenced by the same cross-anticlinal zone that passes through the latter.

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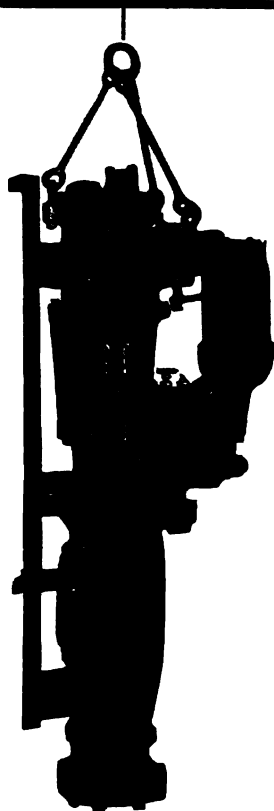
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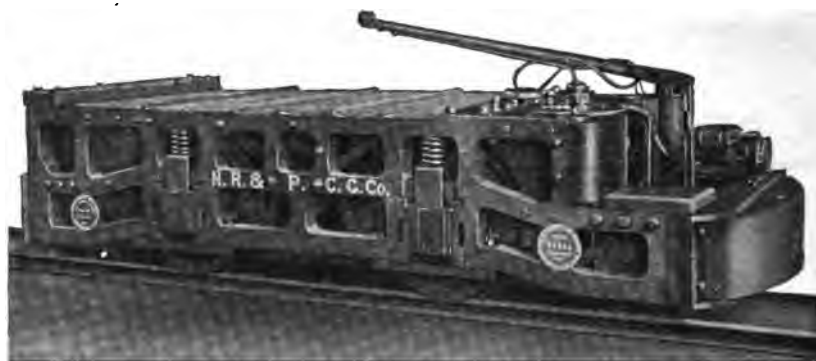
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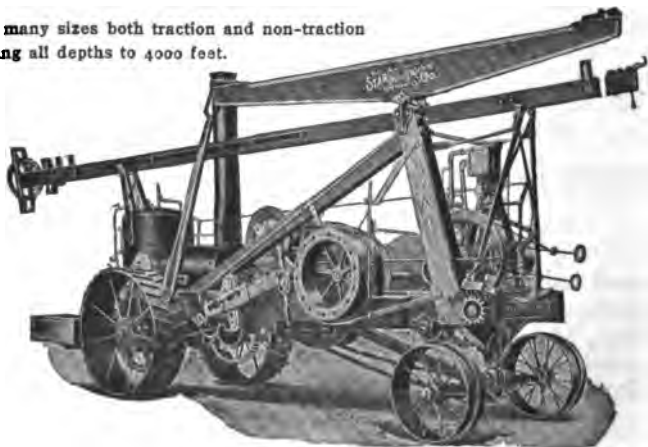
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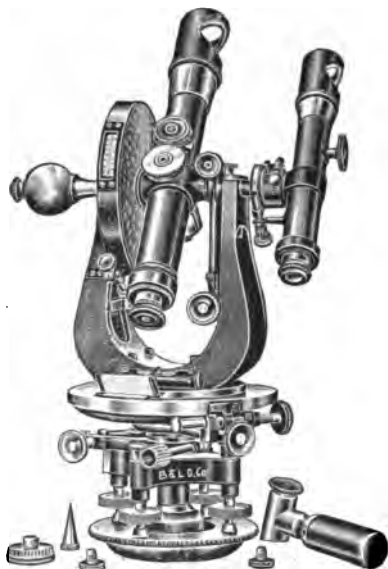
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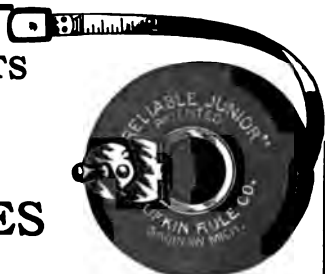
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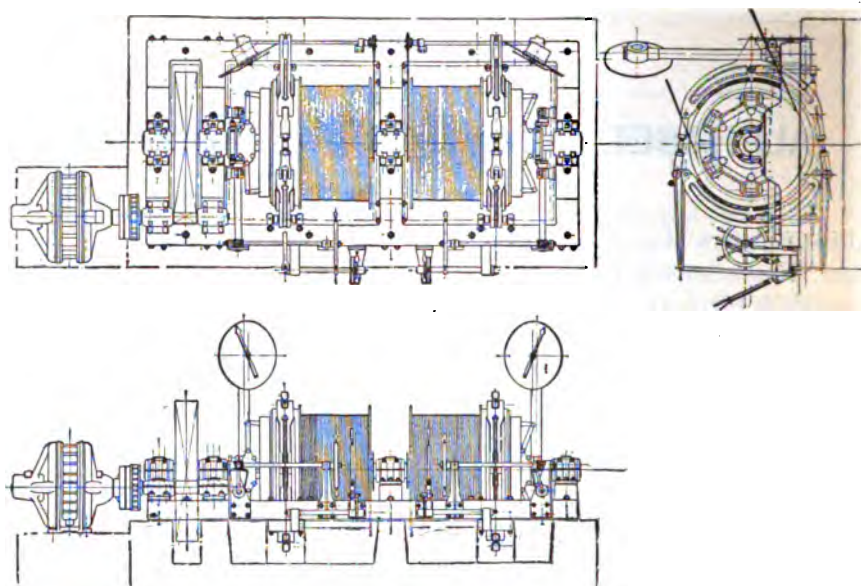
29 West 39th Street

NEW YORK

PROPOSALS FOR MEMBERSHIP. A blank proposal for membership is included in each Year Book and Bulletin. In the event of not being able to secure such a one, applications for membership can be made out on any blank piece of paper, provided they include the class to which the candidate is proposed, whether member, associate or junior member; the signature of three members or associates (junior members must also have the endorsement of two of their instructors); a brief history of the candidate, including date and place of birth, his education and technical record, and finally a signed statement by the applicant that he desires to become a member.

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Typical Nordberg Two-Drum Electric Hoist

Nordberg Electric Hoists

NORDBERG Electric Hoists are built in all sizes, single and double drum, hand operated in smaller sizes, and with oil or air thrust cylinders in larger sizes.

The above drawings show a typical small, hand operated, two-drum hoist of a type built in large numbers. Four were recently supplied for Phelps, Dodge & Co.

Each drum is supplied with a Nordberg axial plate clutch and post brakes suspended by carriers, so as to have absolutely parallel motion.

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Perfect alignment of gear and pinion is assured by the provision of bearings on each side of the pinion and gear, all on the same bed plate, and all arranged at one end of the hoist.

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Manufacturers of High Efficiency Corliss Engines; Uniflow Engines; Poppet Valve Engines; Air Compressors; Blowing Engines; Hoisting Engines; Pumping Engines; and other machinery.





JUNE, 1914



Bulletin of the American Institute of Mining Engineers

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EXAMINE THIS ILLUSTRATION

IT EXPLAINS THE
REASON FOR THE

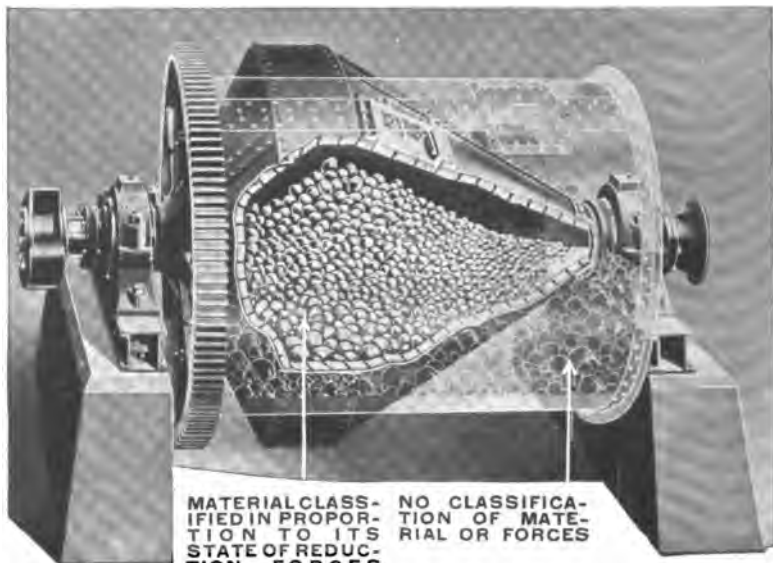
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GETTING ACQUAINTED

The importance to each member of broadening his acquaintanceship and developing personal relationships with fellow members needs no argument, but aside from the annual meetings, and the efforts of local sections, the opportunities for doing this have heretofore been left to chance. To meet this need it is now proposed to establish the custom of an Institute annual dinner for purely social purposes, to be given at the most suitable local center by those members who are within convenient reach of that center.

The plan involves no permanent organization and is so elastic as to fit any conditions from those of the cities to those of the frontier. It merely requires each member to acknowledge his duty to the profession, once a year, by personally meeting the fellow members of his vicinity. The plan has now been in operation for three years at St. Louis, Mo., and for one year at Butte, Mont., and in both cases with such remarkable success that what was begun as a duty is now anticipated with enthusiastic interest. It has been found that members, no matter how successful and influential they may have become, are keenly alive to the importance of keeping in personal touch with the men, and particularly the younger men, of their own profession. The annual dinner has in fact been welcomed and supported by every one as soon as its character and real meaning have been understood.

The President and Board of Directors therefore ask you to co-operate with the other members of your vicinity in establishing the custom at the first suitable opportunity. So far it has been found best to select the season in advance and to fix the actual date by inquiries made at the time. Local sections, or in their absence committees formed to decide upon the most favorable season, place grouping, and other details, may find the experience at St. Louis and Butte suggestive, and inquiries concerning it should be addressed to P. N. Moore, Merchants Laclede Building, St. Louis, and to E. P. Mathewson, Anaconda, Mont., who have kindly offered to furnish any information desired.

In case any such committees feel that the presence of Institute officials or members of the Board of Directors would be of assistance in starting the system, they should correspond directly with those who are known to be within reach and also advise the Secretary of the Institute as to the place and date in view. He may frequently be able to inform them as to officers or Directors who are likely to be in the vicinity at the time. It is believed that the new custom, once firmly established, will have profound and far-reaching effects. Personal relations cannot be created in a day, but their systematic development is certain to bring benefits to the individual member which will exceed all anticipations.

CORRECTIONS FOR MAY, 1914, BULLETIN

Page 738: Fourth, third, and second lines from bottom of page should read:

In the gas-electric station, where the units will be of 2,000-kw. capacity, there will be 32 units.

In the turbo-electric station six 10,000-kw. units will suffice.

Page 743: Eighth line from bottom of page should read:

$$\text{Over-all efficiency} = \frac{398.3 \times 0.70 \times 0.70 \times 0.93}{1,247.4} = 14.6 \text{ per cent.}$$

AMERICAN INSTITUTE OF MINING ENGINEERS
29 West 39th Street, New York, N. Y.

PROPOSAL FOR MEMBERSHIP

Mr. _____
(Name in Full)

Occupation _____

Address _____

is hereby proposed by the undersigned, as a _____

of the American Institute of Mining Engineers.

} Signatures of three
Members or
Associates.

Place of birth _____ *Year of birth* _____

*Education, general and technical, when, where and how acquired,
with degrees, if any.*

<i>Dates</i>	

[illegible]

*Present position*_____

Signature.

Dated _____ 191

ARTICLE II.—MEMBERS.

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

Harvard College Library
Aug. 14, 1916.
Bequest of
Erasmus Darwin Leavitt.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

No. 90

JUNE

1914

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTSKY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STOURGTON, Editor, F. O. PIERCE, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

SALT LAKE CITY MEETING

The One Hundred and Eighth Meeting of the Institute, for the presentation and discussion of technical papers, will be held at Salt Lake City, Utah, August 10 to 14, 1914.

Institute Headquarters will be at the Hotel Utah, where the technical sessions will be held.

In view of the heavy influx of tourists to Utah in August, members who contemplate attending the Salt Lake meeting are urged to communicate with the Secretary of the Institute or the Chairman of the Local Committee, R. C. Gemmell, P. O. Box 1169, Salt Lake City, Utah, at as early a date as possible, in order that provision may be made for suitable hotel accommodations.

NOMINATIONS FOR OFFICERS

The Committee appointed by your Board of Directors to nominate Officers for the Institute for the year 1915 invites the participation of all members in the nominations. The suggestion of names for the various offices will be appreciated and given full consideration when making the selection of candidates which this Committee will recommend.

The officers to be elected at the annual meeting in February, 1915, are:

One officer, known as Director and President.

Two officers, known as Director and Vice-President.

Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The Geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. (N. B.—New York City and district is designated District 0.)

District No. 2. Pennsylvania.

District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.

District No. 4. Minnesota, Wisconsin, and Michigan.

District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.

District No. 6. California and Nevada.

District No. 7. Utah, Colorado, Arizona, and New Mexico.

District No. 8. Louisiana and Texas.

District No. 9. Other Southern States and District of Columbia.

District No. 10. Mexico.

District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:

President, Benjamin B. Thayer (not eligible for re-election).

Past President, James F. Kemp.

Vice-Presidents, Karl Eilers, District 0; Sidney J. Jennings, District 0.

Directors, Edmund B. Kirby, District 3; Joseph A. Holmes, District 9; Robert W. Hunt, District 3; George C. Stone, District 0; Edward L. Young, District 0.

It is the earnest desire of this Committee to obtain the expression of the members, in an endeavor to present to the voters of the Institute as strong and representative a selection of candidates as is procurable.

Communications should be sent to the Chairman of the Nominating Committee, 304 Boston Building, Denver, Colo.

WILLIAM J. COX,
Chairman, Nominating Committee.

BOARD OF DIRECTORS

Meeting of April 24, 1914.—President Thayer announced the appointment of the Committee on Safety and Sanitation.

The Committee on Nominations for officers for 1915 was appointed, as follows: William J. Cox, Chairman; H. Foster Bain, Joseph Daniels, Charles E. Mills, R. V. Norris, E. Gybbon Spilsbury, and Herbert A. Wheeler.

The request of twenty residents of Salt Lake City and vicinity for permission to establish the Utah-Nevada Local Section was granted and R. C. Gemmell was appointed Chairman of the Committee on Organization.

The following resolution was adopted:

Resolved: That under authority of Article V, Section 7, of the Constitution, a special Committee be and hereby is appointed, to consist of the President and six other members of the Institute, to be named by him, who shall be authorized to appear before the proper authorities at Washington and urge that Mining Engineers be appointed upon the proposed Commission for revision of the Federal Mining Laws.

Further Resolved: That the activities of this special Committee be restricted to the authority given in this resolution.

The President then named, as members of this special committee, the following members; Messrs. E. B. Kirby, J. A. Holmes, H. Jennings, C. F. Rand, F. L. Garrison, W. L. Saunders and B. B. Thayer, *ex officio*.

It was voted that no collective Index of the *Transactions* be published this year.

Upon recommendation of the Executive Committee it was voted that all Junior Members in good standing shall be permitted to purchase the *Transactions* of the current year by the payment of \$5 for that year in addition to the regular Junior Membership fee of \$5 per year.

A vote of thanks on behalf of the Institute was extended to Sir Robert A. Hadfield for his generous gift in establishing the Hadfield Research Prize.

Three Associate Members, 126 Members and 5 Junior Members were elected to membership in their respective classes.

Ten Members who had paid their dues for 1913 and previously, were reinstated.

The resignations of four Members were accepted:

The following amendments to the By-laws were approved:

BY-LAW VII, PARAGRAPH 4

Changed to read, "The Committee on Publications shall consist of the Secretary of the Institute, who shall be its Chairman, and at least twelve members of the Institute, to assist in passing on all papers offered for publication. They shall be appointed annually by the Board of Directors."

BY-LAW XVIII, SECTION 6

Add: Any Member or Associate of the Institute whose place of business and residence are outside of the territory of any Local Section, or who is already a member of the Local Section within whose territory he has his place of business or residence, is eligible to become a member of any Local Section of the Institute, under such regulations as the individual Local Section may enact, and shall be designated, when elected, as a "Non-Resident Member" of such latter Section.

SECTION 10, ADD:

This regulation is not intended to forbid any Member of a Local Section from voluntarily enrolling himself as a "Contributing Sectional Member," such agreeing

yearly to be assessed up to a certain maximum annual sum for the expenses of the Section; nor does it exempt either such Members as are eligible to "Resident Membership" who fail to become Members of the Section within three months after due invitation, or "Non-Resident Members" of a Section, from being required to enroll as "Contributing Sectional Members."

SECTION 9, LINE 4:

For "Members" read "Resident Members."

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period April 10 to May 10;

I. N. Knapp, Ardmore, Pa.
Herbert B. Cox, New York, N. Y.
A. McLintock, Brooklyn, N. Y.
Heinrich Ries, Ithaca, N. Y.
R. M. Catlin, Franklin Furnace, N. J.
Percy E. Barbour, Candor, N. C.
A. D. R. Galloway, Oruro, Bolivia.
M. K. Rodgers, Westgate, Cal.

E. P. Mathewson, Anaconda, Mont.
H. B. Barling, New York, N. Y.
M. M. Thompson, Morsemere, N. J.
Howland Bancroft, Denver, Colo.
John H. Adams, Birmingham, Ala.
Henry D. Boddington, London, England.
William J. Cox, Denver, Colo.
W. J. Morrison, Toronto, Canada.

W. J. Richards has been elected President of the Reading Coal & Iron Co., succeeding the late George F. Baer.

H. L. Haldeman has been elected President of the Pulaski Iron Co., Philadelphia.

Charles F. Rand, President of the Spanish-American Iron Co., Past President of the American Institute of Mining Engineers, has been made a member of the Order of Isabella Catolica and decorated by the King of Spain with the Grand Cross of a Knight Commander.

George A. Laird has severed his connection with the Great Cobar Copper Co., and has opened an office in Sydney, N. S. W., to carry on a general consulting practice.

Arthur G. McKee, of Cleveland, Ohio, engineer and general contractor for power plants, cold storage and handling plants, has opened a branch office in New York at 52 William Street.

H. F. E. Gamm has severed his connection with the Oil Furnace Engineering Co., of New York, and will engage in mine examination and engineering work.

James I. Long has resigned his position with the El Dorado Mining & Milling Co., and has taken up his residence at Los Angeles, Cal.

Lawrence Addicks has been engaged by Phelps, Dodge & Co. to take charge of the field work in connection with the hydrometallurgical research of that company.

Robert W. Hunt was elected one of the Vice-Presidents of the National Association for Testing Materials at a meeting of the Council of that Association, held in Turin.

William H. Donner, President of the Cambria Steel Co., has been elected Chairman and Director of the Pennsylvania Steel Co.

George H. Garrey has severed his connection with the American Smelting & Refining Co., and allied companies, as their Chief Geologist, and has opened an office as consulting mining geologist and engineer at 115 Broadway, New York.

G. M. Shoemaker has been appointed Manager of Operations for the LaFollette Coal, Iron & Railway Co., in place of Edward H. Coxe, resigned.

Robert B. Tinsley, Superintendent and Assistant Engineer of the Panama Canal, has tendered his resignation, to accept a position with the Caribbean Petroleum Co. as General Manager of all their oil, asphalt, and railroad interests in eastern Venezuela.

W. D. B. Motter, Jr., until recently Manager of the Canada Iron Mines, Ltd., Trenton, Ont., has accepted a position as Manager of the Benson Mines Co., Benson Mines, N. Y.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Wanted for a mill on the Ivory Coast of Africa an experienced amalgamator and millman. Salary, \$160 to \$200 per month. No. 18.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Chemist-metallurgist of broad technical training, university graduate, with experience in chemical and metallurgical research work, desires position, preferably in the East. Has successful past that stands investigation. No. 73.

Member, mining and metallurgical engineer; thorough technical training, Ph. D., Columbia University, aged 39, married, with an extensive experience in smelting, mining, and economic geological work in U. S., Mexico, and South America, desires position as assistant to the Manager or Superintendent, or as a director of experimental and research work, with a reliable mining or metallurgical company. The West, Mexico, or South America preferred. No. 98.

Member, aged 28, graduate of Lehigh University, with seven years' experience as surveyor and engineer on railroad and construction work, desires position as engineer either in field or office. No. 99.

Member, technical graduate, 32 years old, varied experience from mine surveyor and assayer to mine and mill superintendent, desires position with precious-metal mine. No. 100.

Graduate mechanical engineer (Lehigh, 1910), aged 27, desires position as experimental and test engineer; four years' experience in physical testing, metallography, and heat treatment in a large tire works; familiar with steel foundry work in carbon steels. No. 101.

Member, aged 37, college graduate, married, 14 years' experience in mining and exploring and superintending large low-grade mines; opening, equipping and developing new properties. Has record of decreased costs and increased efficiency. Best references. Desires position with good people where there is a chance for advancement. No. 102.

Member, technical graduate, aged 43, 21 years' experience in connection with coal and coke operations in various parts of the country, 15 years in charge of operations, desires position as general superintendent or general manager of large coal mining company. No. 103.

Member, mining engineer aged 32, technical graduate, with ten years' experience in United States as engineer and general manager, desires management or financial position in United States. Salary not less than \$10,000. Exceptional business experience, does not speak Spanish. Available on 60 day's notice. No. 104.

Member, graduate mining engineer, aged 27, experience as mine surveyor in gaseous and non-gaseous mines of three States, at present Chief Engineer for large coal company, desires position as head of engineering department. No. 105.

A member, technical graduate, with five years' practical experience in iron and steel work, desires a position as Assistant Superintendent of an iron blast-furnace. No. 106.

A junior at Columbia University, with experience in microscopic work on cast irons, is desirous of obtaining a position for the summer doing metallographic work. No. 107.

Member, technical graduate, aged 25, with experience in surveying, sampling, and office work in coal and metal mines, both underground and open pit, desires position. No. 108.

Fuel specialist, with experience in investigating boiler room and fuel troubles, is desirous of obtaining temporary or permanent position in solving power-consumption problems. No. 109.

Member, technical graduate, aged 37, with 14 years' experience in mine and mill operation, consulting work, and as Professor of Mining Engineering, desires a position as manager of development company or metal mine or as Professor of Mining Engineering or Economic Geology. No. 110.

Member, technical graduate, aged 27, four years' experience in coal and iron mines, one year in chemistry, desires position as engineer, assayer or chemist. Has best of reference. No. 111.

NOTABLE GIFT TO THE LIBRARY

UNITED ENGINEERING SOCIETY
29 WEST 39TH STREET
NEW YORK, N.Y.

MR. BRADLEY STOUGHTON,

APRIL 9, 1914.

Secretary American Institute of Mining Engineers,
Engineering Societies Building,
29 West 39th Street, New York.

DEAR SIR:

At the meeting of the Board of Trustees of United Engineering Society, held on March 26, 1914, the Board was honored by a gift from Mr. John W. Lieb, Jr., of a series of volumes, covering the work of Leonardo da Vinci, as engineer, philosopher, and artist. The gift included volumes of the *Codice Atlantico* di Leonardo da Vinci, published under the auspices of the Italian Government; a reproduction of da Vinci's note books, and certain unique monographs, making in all twelve units of the greatest interest.

The Board of Trustees passed formal action of thanks to Mr. Lieb, but directed me in addition, to give voice in each of the Founder Societies, to the hope that such a gift might be exemplary to other persons, and to make public the fact that the Board of Trustees would welcome the de-

velopment in the Library of United Engineering Society, of a department covering the history of the mechanic arts in this and other countries, with biographies and other material covering the great names of men who have labored in this field. It was the hope of the donor and of the Board, that Mr. Lieb's gift might be added to, making a collection of "da Vinciana" in its Library which would be of distinguished interest.

This information is sent to you for transmittal to the governing board of your Institute, and with a request that they be asked to take such action as may be appropriate.

Very truly yours,

F. R. HUTTON, *Secretary*.

The Library Board of United Engineering Society desires to obtain by gift, exchange, or purchase, any of the missing volumes of serial publications listed on page xvi, and requests the co-operation of members. Please notify the Librarian if you have any of these, or know of any one who wishes to dispose of them. In this way you can assist in adding to the resources of the Library.

IRON AND STEEL COMMITTEE

ALBERT SAUVEUR, *Chairman*

A. A. STEVENSON, *Vice-Chairman*

HERBERT M. BOYLSTON, *Secretary*, Abbot Bldg., Cambridge, Mass.

Guilliaem Aertsen,	Henry M. Howe,	E. Gybbon Spilsbury,
John Birkinbine,	Robert W. Hunt,	J. S. Unger,
William H. Blauvelt,	J. E. Johnson, Jr.,	Felix A. Vogel,
Charles A. Buck,	Charles Kirchhoff,	Leonard Waldo,
Ernest F. Burchard,	Richard Moldenke,	William R. Walker,
F. D. Carney,	Charles F. Rand,	William R. Webster,
James Gayley,	Joseph W. Richards,	Frederick W. Wood.
Henry D. Hibbard,	C. F. W. Rys,	

There will be a meeting of the Institute in Pittsburg on Oct. 8, 9, and 10, 1914. This meeting will be under the joint auspices of the Iron and Steel Committee, the Committee on Coal and Coke, and the Committee on Petroleum and Gas. The iron and steel papers will probably be presented on Oct. 8.

The following papers have been received and accepted and will be read at the October meeting.

John Birkinbine: Iron Ore Reserves.

G. K. Burgess: Observations on the Finishing Temperatures and Properties of Rails.

J. V. Emmons: Surface Decarbonization of Tool Steel.

J. C. H. Ferguson: Notes on Roll Shells; Their Manufacture, Their Wear in Service.

R. H. Rice: Turbo-Blowers.

Albert Sauveur: Manganese Steel and the Allotropic Theory.

A paper has been promised on the results obtained by the Snyder electric furnace, and the Pennsylvania Engineering Co. has promised a paper giving plans of duplex steel process plants.

H. M. BOYLSTON, *Secretary*.

COMMITTEE ON SAFETY AND SANITATION**ARTHUR WILLIAMS, *Chairman*****E. MALTBY SHIPP, *Secretary*, 2 Rector Street, New York, N. Y.**

A preliminary meeting of the Committee on Safety and Sanitation, recently formed, was held on May 27, 1914, to discuss plans for the work of the Committee. It was decided that the scope of this Committee's activities should include physical safety, sanitation, and education, and that sub-committees should be formed to cover the various phases of the general subject of safety. As an initial step, it was decided to collect information concerning the safety regulations in force at various mines, and arrange for the compilation of a standard set of rules to meet the different conditions which prevail in mining and metallurgical work.

Members of the Institute can aid the Committee greatly in its work by sending to the Secretary of the Committee the rules now in force at their mines, together with suggestions based on experience in the conduct of safety work.

E. MALTBY SHIPP, *Secretary*.

NEW YORK LOCAL SECTION*Executive Committee***L. W. FRANCIS, *Chairman*****WILLARD S. MORSE, *Vice-Chairman*****THOMAS T. READ, *Secretary*, Woolworth Bldg., New York, N. Y.****P. A. MOSMAN, *Treasurer*****LOUIS D. HUNTOON**

Since the annual meeting in February, the New York Section of the Institute has held three meetings. The meeting on March 10 was held in conjunction with the American Society of Mechanical Engineers and the American Electrochemical Society, at the Engineering Societies Building. The papers of the evening were devoted to records of progress in the Processes for the Photographic Reproduction of Natural Colors. The speakers of the evening were F. E. Ives, one of the earlier workers in color photography, Westley Allison, C. W. Robinson, George F. Clyfton, John B. Taylor, and Henry Hess.

The April meeting was held on April 22 at the Institute headquarters, and the topic for discussion was Sanitation in Mining and Metallurgical Work. The principal paper was by J. W. Luther, chief surgeon of the New Jersey Zinc Co. This was followed by a similar paper by Charles L. Close, secretary of the Welfare and Sanitation Committee of the United States Steel Corporation, and by an account of the sanitation work done in Cuba by the Spanish-American Iron Co., which was read by the Chairman, in the absence of Charles F. Rand. Following these papers general discussion was indulged in.

The concluding meeting of the year was held at Browne's Chop House on May 21. There was held a brief business session, at which the reports of officers were presented and accepted and officers were elected to serve for the year 1914-15.

Following the business session a beefsteak supper was served and at

the close addresses were made by Dr. C. L. Kolbeck, Director of Freiberg Bergakademie; T. A. Rickard, delegate of the Royal School of Mines to the celebration of the Fiftieth Anniversary of the Columbia School of Mines; B. B. Thayer, John C. Montgomery, J. F. Kemp, Franklin Guiterman, and C. F. Chandler.

PENNSYLVANIA ANTHRACITE SECTION

At the first regular meeting of the Pennsylvania Anthracite Section of the American Institute of Mining Engineers, held Saturday evening, May 9, 1914, officers and members of the Executive Committee were elected as follows:

R. V. NORRIS, *Chairman*

Vice-Chairmen

C. F. HUBER,

EDWIN LUDLOW,

W. J. RICHARDS,

A. H. STORRS,

CHARLES ENZIAN, *Secretary-Treasurer*, U. S. Bureau of Mines, Wilkes-Barre, Pa.

DOUGLAS BUNTING,

R. J. FOSTER,

F. A. HILL,

J. M. HUMPHREY,

A. B. JESSUP,

R. A. QUINN.

By-laws for the Section were adopted.

Douglas Bunting presented a tentative outline of a paper to be prepared by the Committee on Mining Under Heavy Wash, of which he is Chairman. This outline was discussed by members, William Griffith, Frank A. Hill, I. A. Stearns, G. W. Engel, K. M. Smith, H. W. Montz, H. H. Otto, R. J. Foster, and A. H. Storrs.

CHARLES ENZIAN, *Secretary*.

BOSTON LOCAL SECTION

Executive Committee

HENRY L. SMYTH, *Chairman*

ALFRED C. LANE, *Vice-Chairman*

AUGUSTUS H. EUSTIS, *Secretary-Treasurer*, 131 State St., Boston, Mass.

TIMOTHY W. SPRAGUE

HENRY A. WENTWORTH

Eighteenth Meeting

The Eighteenth meeting of the Boston Local Section, being the third annual meeting, was held at the Engineers' Club, 2 Commonwealth Avenue, Boston, Monday evening, March 9. Eighteen members were present.

The Chairman, Robert H. Richards, called the meeting to order.

The officers for the ensuing year were elected as follows:

Chairman, **HENRY L. SMYTH,**

Vice-Chairman, **ALFRED C. LANE,**

Secretary-Treasurer, **A. H. EUSTIS.**

A. W. Stickney was then introduced, and delivered an address on The Pyritic Ore Deposits of Kyshtim, Russia, which was discussed by Messrs. Wenstrom and Batchelder.

Nineteenth Meeting

The Nineteenth meeting was held at the Engineers' Club, Monday evening, Apr. 6. There were 24 members present.

Prof. Waldemar Lindgren gave an illustrated talk on his recent trip to Cuba. At the close of the lecture, the members present had an opportunity to examine the specimens which Professor Lindgren had with him.

AUGUSTUS H. EUSTIS, *Secretary*.

CHICAGO LOCAL SECTION*Executive Committee*

ROBERT W. HUNT, *Chairman*

J. A. EDE, *Vice-Chairman*

HENRY W. NICHOLS, *Secretary-Treasurer*, Field Museum of Natural History, Chicago, Ill.

F. K. COPELAND

G. N. DAVIDSON

A meeting of the Chicago Local Section was held at the Sherman House on Tuesday evening, May 12, 1914. Nineteen members and guests were present. After an address by the Chairman, papers were read by H. W. Nichols, on The Field Museum and Its Collections, and by Dr. F. W. DeWolf, on The Work of the State on the Mineral Resources of Illinois. Both papers were thoroughly discussed by the members present.

H. W. NICHOLS, *Secretary*.

COLORADO LOCAL SECTION*Executive Committee*

FRANK BULKLEY, *Chairman*

S. A. IONIDES, *Vice-Chairman*

C. LORIMER COLBURN, *Secretary-Treasurer*, 614 Ideal Building, Denver, Colo.

JAMES A. McCLAVE

DAVID G. MILLER

A meeting of the Colorado Local Section of the Institute was held in the lecture room of the Colorado Scientific Society, in the Boston Building, Denver, on Apr. 23, 1914. The chairman, Mr. Frank Bulkley, called the meeting to order. About 40 members and friends were present.

Dr. R. B. Moore, of the U. S. Bureau of Mines, delivered a very instructive and entertaining lecture upon the subject of Radium and Radioactivity. He exhibited several radium compounds and showed, by the use of the lantern, the effect of the several rays upon the electroscope.

C. L. COLBURN, *Secretary*.

INTERNATIONAL ENGINEERING CONGRESS, 1915

Rapid progress is being made in working out the final program of papers for the International Engineering Congress to be held in San Francisco in 1915.

The first volume of the publications of the Congress will consist of a series of articles descriptive of the various technical features of the design and construction of the Panama Canal. The various topics which will be treated are noted in the following list:

- | | |
|---|--|
| Introductory Chapter. | Design of Lock Walls and Valves. |
| Dry Excavation. | Design of the Spillways. |
| Dredging. | Gates of the Panama Canal Locks. |
| Terminal Works, Dry Docks, and Wharves. | Electrical and Mechanical Installations. |
| Permanent Shops. | Emergency Dams above Locks. |
| Coaling Plants and Floating Cranes. | Municipal Engineering and Domestic |
| Meteorology and Hydrology. | Water Supply in the Canal Zone. |
| Design of Locks, Dams, and Regulating Works. | Reconstruction of the Panama Railroad. |
| Method of Construction of the Locks, Dams, and Regulating Works in the Atlantic Division. | Aids to Navigation of the Canal. |
| Method of Construction of the Locks, Dams, and Regulating Works in the Pacific Division. | Geology of the Canal Zone. |
| | The Working Force. |
| | Sanitation in the Canal Zone. |
| | Purchase of Supplies. |

Each of these topics will be treated by someone on the canal force who has been responsible for the design and construction described. The introductory chapter as well as the topic of Dry Excavation for the Panama Canal will be handled by Colonel Goethals himself.

The program of papers for the various sections of the Congress is practically completed. Among the general subjects to be treated, probably the one having the broadest interest is that of Materials of Engineering Construction, which enters into all phases of engineering activity. The list of topics which will be treated is as follows:

- | | |
|--|--|
| Timber. | The Life of Iron and Steel Structures. |
| Preservative Treatment of Timber. | The Employment of Special Steel in Engineering Construction. |
| Substitutes for Timber in Engineering Construction. | The Place of Copper in the Present Engineering Field, and the Economics of the World's Supply Thereof. |
| Brick in Engineering Structures. | Alloys and Their Use in Engineering Construction. |
| Clay Products in Engineering Structures. | Aluminum in Engineering Construction. |
| Probable and Presumptive Life of Concrete Structures made from Modern Cements. | The Influence of the Testing of Materials upon Advances in the Designing of Engineering Structures and Machines. |
| Aggregates for Concrete. | Cement Testing. |
| Slag Cement. | Testing of Metals. |
| Waterproof Concrete. | Testing Full-Sized Members. |
| Cements containing Additions of Finely Ground Foreign Material. | Proof Testing of Structures. |
| Economics of the World's Supply of Iron. | |

The aggregation of papers which will be presented will constitute a broad review of the field and be of the highest value.

Full information concerning the Congress, the price of subscription, and the arrangement for purchase of volumes of the proceedings, may be obtained by addressing the Committee of Management, as follows:

INTERNATIONAL ENGINEERING CONGRESS, 1915, Foxcroft Building,
San Francisco, Cal., U. S. A.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM B. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining-maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

SERIAL PUBLICATIONS NEEDED TO COMPLETE SETS IN THE LIBRARY OF THE ENGINEERING SOCIETIES

- Acetylene Journal, 1-13, 1899-1912.
 Acetylen in Wissenschaft und Industrie, 1-12, 1898-1909.
 Acetylene, 1-5, 1903-1908.
 Aéronautical Journal, 1-12, 1897-1908.
 Allgemeine Automobil Zeitung, 1-9, 1900-1908 (Berlin).
 Allgemeine Bauzeitung, 1-date, 1836-date.
 American Lumberman, 1-date, 1899-date.
 Annalen der Chemie (Liebig's), 1-364, 1832-1909.
 Annales des Mines de Belgique, 1-8, 1896-1903.
 Annales des Ponts et Chaussées. Memoires et documents. Ser. 1-5; Ser. 6, vols. 1-16, 19-end of ser.; Ser. 7; Ser. 8, vols. 1-30, 36.
 ——— Lois decrets, etc. Ser. 1-5; Ser. 6, vols. 1-8, 10-end of ser.; Ser. 7; Ser. 8, vols. 1-7, 8 pt. 1.
 ——— Personnel. All before 1889, and any issued to date.
 Annales des Travaux Publics de Belgique. 1-date, 1843-date.
 Archiv für Eisenbahnwesen. 1-30, 1878-1907.
 Autogene Metallbearbeitung. 1-date, 1908-date.
 Berg und Hüttenmännische Zeitung. Vols. 1-21; 44, no. 36; 49, nos. 14, 45, 51;

- 50, nos. 8, 47; 51, no. 19; 52, nos. 39, 41, 42, 44-52; 54, nos. 2, 3, 4, 25, 26, 29, 30, 32; 55, no. 10.
- Beton und Eisen. 1-2, 1902-1903.
- Braunkohle. 1-7, 1902-1908.
- Brick, 1-42, 1894-1913.
- Cement and Engineering News. 1-21, 22, no. 2, 1896-1909, 1910.
- Centralblatt der Bauverwaltung. 1-33, 1881-1913.
- Chemical Engineer 1-4, 1904-1906.
- Chemical Society of London. Journal. 1-26, 1849-1873.
- Chemiker Zeitung. 1-10, 12, 1877-1886, 1888.
- Chemische Industrie. 1-date, 1878-date.
- Chemisches Centralblatt. 1-date, 1830-date.
- Ciment (Paris). 1-date, 1896-date.
- Connecticut Society of Civil Engineers. Papers. 1-25, 27, 1894-1909.
- Contractor (Chicago). 1-date, 1898-date.
- Deutsche Chemische Gesellschaft, Berichte. 1-6, 1868-1873.
- Eisen Zeitung. 1-24, 1880-1903.
- Electricien. Ser. 1, vol. 10 to end of series, 1886-1890.
- Elektrische und Maschinelle Betriebe (Leipzig). 1-date, 1898-date.
- Elektrochemische Zeitschrift. 1-8, 1894-1901.
- Elektrophysikalische Rundschau. 1-3, 1910-1912.
- Elektrotechnische Nachrichten. 1-date, 1905-date.
- Feuer und Wasser. 1-date, 1894-date.
- Fonderie Moderne. 1-4, 1908-1911.
- Gas World. 1-58, 1884-1913.
- Gazetta chimica Italiana. 1-date, 1871-date.
- Geographical Journal. 33-date, 1908-date.
- Geological Magazine. 4 to new ser. decade 5, vol. 5, 1867-1908.
- Geologisches Zentralblatt. 1-11, 1898-1908.
- Gesundheits Ingenieur. 1-31, 1878-1908.
- Glückauf. 1-31; 36, no. 27; 38, nos. 2, 7, 1865-1902.
- Gummi Zeitung. 1-21, 1888-1906.
- Haeder's Zeitschrift für Maschinenbau. 1-20, 1893-1912.
- Helios. 1-date, 1895-date.
- Houille Blanche. 1-10, 1902-1911.
- Ingenieur (The Hague). 1-24, 1886-1902.
- Institute of Marine Engineers. 1-20, 1889-1909.
- Institution of Engineers and Shipbuilders in Scotland. 1, 8, 1857, 1865.
- Iron Age, 1-23, 1855-1878.
- Jern Kontorets Annaler. 2, 4, 13, 16, 17, 19, 20. New ser. 1. pts. 1-2; 3; 5-11; 12. pt. 5-end of vol.; 14-15; 28, pts. 1-3, 6-end of vol.; 29, pts. 1-2, 4-5; 30, pt. 2; 31, pts. 2-5; 54, pt. 1; 63, pts. 3-4, 1818-1908.
- Bihang. All before 1828, 1829-1836, 1838-1867, 1869-1899.
- Register. 1817-1890.
- Journal für Gasbeleuchtung. 1-51, 1858-1908.
- Journal für Praktische chemie. 1-date, 1828-date.
- Journal of Gas Lighting. 1-22, 51-52, 56, 57, 62-63, 93, 96, 99-108, 1850-1909.
- Koninklijk Instituut von Ingenieurs (Hague). Tijdschrift, 1870-1903, 1906.
- Marine Engineer and Naval Architect. 1-25, 1879-1903.
- Metallröhren Industrie. 1-date, 1907-date.
- Métallurgie (Paris). 1-32, 37, 40, 1868-1909.
- Mining and Scientific Press. 1-9, 11-19, 24-33, 1860-1876.
- Mining Journal (London). 1-47, 59, 1835-1877, 1889.
- Mittheilungen aus Justus Perthes Geographischer Anstalt. 43-date, 1897-date.
- Motorwagen (Berlin). 1-13, 1898-1908.
- Municipal Engineering. 1-21, 24-31, 1890-1901, 1903-1906.
- Nature (Paris). 1-date, 1873-date.
- Neueste Erfindungen und Erfahrungen. 1-date, 1874-date.
- Oesterreichische Zeitschrift für Berg und Hüttenwesen. 1-26, 1853-1878.
- Omnia. 1-7, 1906-1912.
- Petroleum. 1-3, 1905-1907.
- Petroleum Review, London. 1-25, 1889-1911.
- Petroleum World. 1-8, 1903-1910.
- Prometheus. 1-19, 1879-1908.
- Revista tecnica delle Ferrovie Italiana. 1-date, 1912-date.
- Revue de la Soudure Autogène. 1-date. 1909-date.

- Revue Générale des Sciences Pure et Appliquée. 1-10, 1890-1899.
 Schiffbau. 1-9, 1899-1908.
 Schweizerische Bauzeitung. 1-56, 1874-1910.
 Société Belge des Electriciens. 1-3, 5, 8-9, 11-20, 1884-1903.
 Société Industrielle de Mulhouse. 1-71, 1828-1901.
 Sozial Technik. 1-6, 1902-1907.
 Sprechsaal. 1-41, 1868-1908.
 Teknisk Tidsskrift. 1-25, 1877-1901. (Kopenhagen.)
 Telefunkenzeitung. 1-date, 1911-date.
 Tonindustrie Zeitung. 1-31, 1877-1907.
 Verein zur Beförderung des Gewerbefleißes. Verhandlungen. 1-88, 1822-1909.
 Water and Water Engineering. 1-date, 1899-date.
 Zeitschrift des Verbandes deutscher Schiffingenieure. 1-date, 1911-date.
 Zeitschrift für Architektur u. Ingenieurwesen (Architekten u. Ing. Verein zu Hannover). 1-48, 1855-1902.
 Zeitschrift für Bauwesen. 1-date, 1851-date.
 Zeitschrift für Beleuchtungswesen. 1-7, 1895-1901.
 Zeitschrift für Dampfkessel und Maschinenbetrieb. 1-36, 1878-1913.
 Zeitschrift für Elektrotechnik und Maschinenbau. 1-5, 1898-1902.
 Zeitschrift für Heizung, Lüftung und Beleuchtung. 1-12, 1896-1908.
 Zeitschrift für Kleinbahnen. 1, 3-8, 1894, 1896-1901.
 Zeitschrift für komprimierte und flüssige Gase. 1-date, 1897-date.
 Zeitschrift für physikalische Chemie. 1-date, 1887-date.
 Zeitschrift für Sauer und Stickstoffindustrie. 1-date, 1908-date.
 Zeitschrift für Transportwesen. 1-28, 1884-1907.
 Zeitschrift für d. Gesamte Kälteindustrie. 1-15, 1894-1908.
 Zeitschrift für das ges. Schiess u. Sprengstoffwesen. 1-date, 1906-date.
 Zeitschrift für d. Gesamte Turbinenwesen. 1-3, 1904-1906.
 Zement und Beton. 1-date, 1902-date.

LIBRARY ACCESSIONS

New Books on Mining, Metallurgy, and Chemistry

- QUELQUES NOTES SUR LES FACTEURS A ENVISAGER DANS L'ETUDE D'UN GISEMENT METALLIFERE. By J. Maurice. N. p., 1914. (Gift of Author.)
 HOW TO BUILD UP FURNACE EFFICIENCY. By Joseph W. Hays. Ed. 7, 1914.
 DIE QUANTITATIVEN UNTERSUCHUNGSMETHODEN DES MOLYBDANS, VANADIUMS UND WOLFRAMS. By Hans Mennicke. Berlin, 1913.
 DIE WIRTSCHAFTLICHKEIT DES MASCHINENBETRIEBES IM BERGBAU. By Karl Schultze. (Sammlung Berg und Hüttenmännischer Abhandlungen, pt. 143.) Kattowitz, 1914.
 DIE BEDEUTUNG DER SKANDINAVISCHEN EISENERZVORKOMMEN FÜR DIE DEUTSCHE EISENINDUSTRIE. (Sammlung Berg und Hüttenmännischer Abhandlungen, pt. 141.) Kattowitz, 1914.
 WALLAROO AND MOONTA MINES. Their history, nature and methods together with an account of the concentrating and smelting operations. Adelaide, 1914. (Gift of Wallaroo & Moonta Mining & Smelting Co.)
 LEHRBUCH DER BERGBAUKUNDE. Ed. 3, vol. 1, By F. Heise and F. Herbst. Berlin, 1914.
 FIRST-AID INSTRUCTIONS FOR MINERS. Miners' Circular No. 8, U. S. Bureau of Mines. Washington, 1914.

New Books on Geology and Mineral Production

- KRISTALLBERECHNUNG UND KRISTALLZEICHNUNG. By B. Gossner. Leipzig, 1914.
 MANUAL OF PETROGRAPHIC METHODS. By Albert Johannsen. New York, 1914.
 IRON, STEEL, COAL AND METAL STATISTICS, 1913. London, 1914. (Gift of Iron and Coal Trades Review.)
 MINERAL RESOURCES OF OREGON. Vol. 1, No. 1, Jan., 1914. Portland, 1914.
 MYSORE GEOLOGICAL DEPARTMENT. Report of the Chief Inspector of Mines, 1912-13. Bangalore, 1914.
 DIE EISENERZLAGER VON KIRCHENTHUMBACH IN DER OBERPFALZ. (Sammlung Berg und Hüttenmännischer Abhandlungen, pt. 140.) Kattowitz, 1914.

New Books on Non-Metallic Minerals

- CHEMISCHE UMSETZUNGEN WAHREND DER BILDUNG DER STEINKOEHLE. (Sammlung Berg und Hüttenmännischer Abhandlungen, pt. 139.) Kattowitz, 1914.
- PRACTICAL STONE QUARRYING. By Allan Greenwell and J. V. Eldsen. New York, 1913.
- PRELIMINARY REPORT ON THE UTILIZATION OF PETROLEUM AND NATURAL GAS IN WYOMING. Technical Paper No. 57, U. S. Bureau of Mines. Washington, 1913.
- COAL MINING PRACTICE IN DISTRICT VIII (Danville, Ill.). By S. O. Andros. Bull. No. 2, Illinois Coal Mining Investigations. Urbana, 1914. (Gift of Author.)
- DIAMOND FIELDS OF SOUTHERN AFRICA. By P. A. Wagner. Johannesburg, 1914. (Gift of Author.)

[NOTE.—This is an enlarged edition of a work in German by the author, published in 1909; the author states in his preface that in the present edition he has “resolved that it should contain all that is worth knowing of an industry, small in its beginnings which as a result of the enormous increase in the world’s wealth during the two past decades has assumed astonishing proportions.” This is an ambitious project, but the author has at least succeeded in furnishing an amazing amount of information. A long bibliography and numerous footnotes indicate an exhaustive search through the literature. There is a general map of the region and numerous diagrams and photographs illustrate the text. The treatment of the subject is less popular than in Williams’ “Diamond Mines of South Africa” and the two taken together seem to cover the subject completely.—W.P.C.]

New Books on General Subjects

- GUIDE TO THE CURRENT PERIODICALS AND SERIALS OF THE UNITED STATES AND CANADA. Ed. 3, 1914. By H. O. Severance. Ann Arbor, 1914.
- MINING WORLD INDEX OF CURRENT LITERATURE. Vol. IV, 1913. By George E. Sisley. Chicago, Mining World Company, 1914. Price \$2. (Gift of Publishers.)
- [NOTE.—In this volume of about 200 pages are classified all articles appearing in periodicals, society publications, and publications of national and State governments, on mining, metallurgy, and related subjects. A mining engineer or metallurgist who wishes to keep track of the literature of these subjects will find it a time-saving tool. The publishers keep on file copies of all articles indexed, which they forward for a small payment. Thus, no matter how isolated an engineer may be, he can, with the help of this index, procure a series of articles on any subject, at small cost.—W.P.C.]

Company Reports

- NEVADA HILLS MINING Co. 4th Annual Report, 1913.
- WALLAROO & MOONTA MINING & SMELTING Co., LTD. 24th Annual Report, 1913. Adelaide, 1913. (Gift of Wallaroo & Moonta Mining & Smelting Co.)
- BUFFALO MINES, LTD. 7th Annual Report, 1913.
- CONSOLIDATED MINING & SMELTING Co. OF CANADA. Report of the Directors, for 15 months ending Sept. 30, 1913.
- GEDULD PROPRIETARY MINES, LTD. Directors’ Report for the Quarter ended Sept. 30, 1913.
- GOLDFIELD CONSOLIDATED MINES Co. 7th Annual Report, 1913.
- PORTLAND GOLD MINING Co. 20th Annual Report, 1914.
- ROCHESTER WEAVER MINING Co. Annual Report, 1913.
- VAN RYN GOLD MINES ESTATE, LTD. Directors’ Report, 1913.

Trade Catalogues

- HARRIS-STEVENS Co., Pittsburg, Pa. Stevens centrifugal mine fan. Modern mine cars.
- NEW YORK AIR BRAKE Co., New York, N. Y. Catalogue—Air-brake apparatus. Sept., 1913.
- SHARP-HUGHES TOOL Co., Houston, Texas. S-H rotary rock drill bit. 61 pp.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Apr. 10 to May 10, 1914:

Members

- ADAMS, HENRY H., JR., Merchant, Henry H. Adams & Co., and Colonial Iron Co.,
149 Broadway, New York, N.Y.
- ADAMS, LELAND D., Mine Mgr., Supt., East Canada Smelting Co.,
Weedon, Quebec, Canada.
- AGUILERA, EUGENIO, Chief Engr. Mining & Foresting Dept. of Santiago,
Santiago de Cuba, Cuba.
- ALFORD, NEWELL GILDER, Min. Engr., Asst. Chief Engr.,
St. Bernard Mining Co., Earlington, Ky.
- ALLEN, CHESTER ARTHUR, Asst. Supt., Standard Cons. Mining Co., Bodie, Cal.
- ANDERSON, PAUL SAMUEL, Met. 2235 Auburn Ave., Baker, Ore.
- BAER, GEORGE CHARLES, Min. Engr., Care Socorro M. & M. Co., Mogollon, N. M.
- BARLING, H. B., Min. Engr. 2 Marble Hill Ave., New York, N. Y.
- BARR, JAMES ALLEN, Min. Engr., Atlantic Phosphate & Oil Corp.,
Mt. Pleasant, Tenn.
- BATTEN, H. L. Asst. Min. Engr., Cons. M. & S. Co., Rossland, B. C., Canada.
- BELL, DONALD ALEXANDER SMITH, Met., Concentrator No. 6,
Ariz. Copper Co., Box 38, Morenci, Ariz.
- BIGELOW, BRAXTON, Engr., Morococha Mining Co. Morococha, Peru, S. A.
- BOISE, CHARLES WATSON, Min. Engr., Care Forminiere, 8 Montagne du Parc,
Brussels, Belgium.
- BOOTH, EVERETT L., Min. Engr., Swastika Dev. Co. Mayer, Ariz.
- BRYANT, ALBERT DALY, Asst. Chief Oxide Dept., East Plant,
New Jersey Zinc Co., Horse Head Inn, Palmerton, Pa.
- BURBRIDGE, THOMAS BARNES, Mine Owner and Operator Cripple Creek, Colo.
- COCHRAN, WILLIAM A., Min. Engr. Lee Building, Pottsville, Pa.
- COUNSELMAN, THEODORE B., Efficiency Engr., Arizona Copper Co., Ltd.,
Morenci, Ariz.
- DOERR, KUNO Genl. Mgr., El Paso Smelting Works, El Paso, Texas.
- DU MOULIN, WALTER LOUIS, Civil Engr., Supt., Morenci Water Co., Morenci, Ariz.
- EASTER, HENRY FREDERIC, Asst. Supt., El Paso Smelter El Paso, Texas.
- ERICSON, RUDOLPH, Mine Supt. Iron River, Mich.
- EVANS, JAMES PORTER, Vice-Pres. and Gen. Mgr., Colorado Iron Works Co.,
P. O. Box 1713, Denver, Colo.
- FAHRINGER, WALTER, Div. Supt., Lehigh & Wilkes-Barre Coal Co. Audenried, Pa.
- FERGUSON, CHARLES VAUGHAN, Research Chem., Research Laboratory,
Gen'l. Electric Co., Schenectady, N. Y.
- FEUCHERE, LEON, Geol., Copper Queens Cons. Min. Co. P. O. Box 1086, Bisbee, Ariz.
- FOERSTERLING, HANS, Chem., Roessler & Hasslacher Chemical Co.,
380 High St., Perth Amboy, N. J.
- FREEMAN, RICHARD RICH, JR., Min. Geol., Cleveland-Cliffs Iron Co., Box 55,
Ishpeming, Mich.
- FRIELINGHAUS, GEORGE, Min. Engr., Director of Krupp Works, Essen Ruhr, Germany.
- GAENSLER, GEORGE R., Min. Engr., Asst. Chem., Chino Copper Co., Hurley, N. M.
- GEORGE, PERCY WILLIAM, Min. Engr., Blubber Bay, Van Andam,
Texada Island, B. C., Canada.
- GLASS, FRANK A., Min. Engr. Brainerd, Minn.
- GLENDINNING, ROBERT JAMES, Mgr., R. J. Glendinning & Co., 619 First Avenue,
Salt Lake City, Utah.
- GORMAN, THOMAS C. Surveyor, Dome Mine, South Porcupine, Ont., Canada.
- GREEN, RALPH B. Asst. Supt., Hayden Plant, A. S. & R. Co., Hayden, Ariz.
- HARDING, GEORGE Mgr., Paulsen Bldg., 1105 Paulsen Bldg., Spokane, Wash.
- HARTER, ISAAC, JR. Supt., Babcock & Wilcox Co., Bayonne, N. J.
- HATCHETT, SAMUEL P., Assayer Socorro Min. & Mill. Co., Mogollon, N. M.

- HEGGIE, JAMES, Supt. Hayden Plant, A. S. & R. Co., Hayden, Ariz.
 HENDERSON, JOSEPH W., Vice-Pres., Gulick-Henderson Co., 525-529 Third Ave.,
 Pittsburgh, Pa.
 HEWITT, GEORGE WALTER, Supt. Blast Furnace, 1107 Chapline St., Wheeling, W. Va.
 HIBBERT, ARTHUR, Min. Engr., Care Ptarmigan Mines, Ltd., 109 Belmont House,
 Vict., B. C., Canada.
 HOCKING, RICHARD O., Asst. Supt. Keewatin Mining Co., Keewatin, Minn.
 HOOK, JOSEPH STANLEY, Instructor of Economic Geology, 804 E. Seneca St.,
 Ithaca, N. Y.
 HURD, RALPH, Cons. Engr. Swastika, Ont., Canada.
 INGERSOLL, JULIUS CURTIS, Min. Engr. Ouray, Colo.
 JAUREGUIY RODRIGUEZ, ISIDRO G., Principal Asst. Engr.,
 Spanish-American Iron Co., Santiago, Oriente, Cuba.
 JENKINS, WILLIAM J., Vice-Pres. and Gen. Mgr., Cons. Coal Co. of St. Louis and
 Western Coal Mining Co., 1155 Railway Exchange Bldg., St. Louis, Mo.
 JOHNSTON, JESSE K., Vice-Pres. and Gen. Mgr., Pitcairn Coal Co., East Haven, Conn.
 JONES, ALEXANDER B., Min. Engr., Mgr., Pozo Gilpin Co., Box 68,
 Idaho Springs, Colo.
 JONES, CARLTON HILL, Salesman University Club, Salt Lake City, Utah.
 KATELEY, RAYMOND, Min. Engr. . Bel Cobe Camp, P. O. Naraguta, N. Nigeria, Africa.
 KELLY, JAMES ARTHUR, Chief Engr., Pocahontas Coal & Coke Co.,
 740 Henry Bldg., Seattle, Wash.
 KICHLIN, FRANK O., Chief Chem., Bethlehem Steel Co., 35 First Ave., Bethlehem, Pa.
 KNAPP, ARTHUR, Petroleum and Gas Engr. 116 Ardmore Ave., Ardmore, Pa.
 LANG, HENRY, Mfr. 99 South Fullerton Ave., Montclair, N. J.
 LAVOIE, GERMAIN ADEODAT, Min. Engr., Canadian Coal & Coke Co., Frank,
 Alberta, Canada.
 LICHTENBERG, ERNST, Cons. Min. Engr., 3 Great Winchester St., London, E. C.,
 England.
 LIDVALL, ED. ROBERT, Met. 159 Pierpont Ave., Salt Lake City, Utah.
 LIVERMORE, ROBERT, Min. Engr. Mgr., Kerr Lake Mine, Cobalt, Ont., Canada.
 LUNDY, WILSON THOMAS, Mining, Asst. Supt., McDonald Mine,
 East Canada Smelt. Co., Ltd., Weedon, Quebec, Canada.
 McMILLEN, RUSSELL H., I. and S. Met. 134 Second St., Aspinwall, Pa.
 MAGNUS, BENJAMIN, Gen. Mgr., Mount Morgan Mine, Mount Morgan,
 Queensland, Australia.
 MANNING, VAN. H., Asst. Dir. Bureau of Mines, Washington, D. C.
 MATHESIU, WALTER, Geh. Registrungsrat, Prof. Eisenhüttenkunde,
 Tech. Hochschule, Charlottenburg, Germany.
 MAZANY, MARK STEPHEN, Smelter Foreman, A. S. & R. Co., Hayden Plant,
 Hayden, Ariz.
 MEYER, ALBERT RICHARD, Electrician, Bournewood, South Street,
 Chestnut Hill, Mass.
 MILLER, HUGO W., Assaying, Engineering and Ore Buying Nogales, Ariz.
 MILLWARD, WILLIAM, Chief Engr. Blair Engineering Co., Chicago, Ill.
 MOERNE, JOHN, Mgr. National Copper Mining Co., Mullan, Idaho.
 MOELLER, FRANKLIN, Mech. Engr. 42 Chapman Ave., E. Cleveland, Ohio.
 MOENKE, WILLIAM FREDERICK, Min. Engr., Colorado Fuel & Iron Co.,
 Sunrise, Wyo.
 MOORE, BLISS, Min. Engr., Chino Copper Co., P. O. Box 26, Hurley, N. M.
 MUSSIGBROD, LUDWIG L., Mech. Engr. Garnet, Mont.
 NEALE, LAWRENCE IRVING, Gypsum Mining and Manufacture, 17 State St.,
 New York, N. Y.
 NEWTON, FRANCIS CLEMENT, Supt., American Smelting & Refining Co.,
 Perth Amboy, N. J.
 NIBECKER, KARL, Steam Engr. . . Youngstown Sheet & Tube Co., Youngstown, Ohio.
 PAGE, EDWARD, Vice-Pres., Federal Coal & Coke Co., 111 Devonshire St.,
 Boston, Mass.
 PARKER, SPENCER M., Met. Mina Babilonia, La Libertad, Nicaragua, C. A.
 PRETTYMAN, FRANK REMINGTON, Min. Geol.,
 Care F. Remington, 135 East 66th St., New York, N. Y.
 OLSEN, CHARLES OSCAR, Timberman, Surveyor's Helper, Federal M. & S. Co.,
 Wardner, Ida.
 REID, WALTER L., Mill Supt. Box 471, Telluride, Colo.
 RICE, LUTHER VINTON, Civ. and Min. Engr., 2200 Insurance Exchange Bldg.,
 Chicago, Ill.

- ROBERTS, W. F., Genl. Supt. Bethlehem Steel Co., South Bethlehem, Pa.
 RODGERS, S. M., Met. Frick Bldg., Pittsburgh, Pa.
 RODGERS, OSBORN PEABODY, Min. Engr., Kensington Mining Co.,
 Kensington, Alaska.
 ROSE, CHARLES C., Supt. Coal Dept., Delaware & Hudson Co., 601 Webster Ave.,
 Scranton, Pa.
 ROSSBERG, WILLIAM NICHOLAS, Met. and Min. Engr. Box 87, Butte, Mont.
 ROYCE, WARD, Min. Engr., Shannon Copper Co. Gleeson, Ariz.
 SCHAEFER, FREDERICK RULLMAN, District Sales Mgr.,
 Taylor-Wharton Iron & Steel Co., 30 Church St., New York, N. Y.
 SHEARMAN, WILLIAM HUGH, Engr. 540 Park Ave., New York, N. Y.
 SIBBALD, ALEXANDER, Min. Engr. Arizona Copper Co., Metcalf, Ariz.
 SPEAK, SAVANNAH JOHNSON, Cons. Engr., 3 London Wall Bldgs., London, E. C.,
 England.
 TARANTOUS, RICHARD, Min. Engr. Box 343, Ray, Ariz.
 TEAL, GEORGE W., Min. Engr., Mgr., Mines Primos Mining & Milling Co.,
 Boulder, Colo.
 VADNER, CHARLES SAMUEL, Research Chem. and Met., 2505 South 9th East.,
 Salt Lake City, Utah.
 WADDELL, GEORGE FULTON, Genl. Mgr., Ohio Copper Mining Co.,
 P. O. Box 1167, Salt Lake City, Utah.
 WEST, WILLIAM C., Min. and Met. Engr., Mgr., Peterson Lake Mining Co.,
 Cobalt, Ont., Canada.
 WETHERBEE, HERBERT E., Min. Engr., Crowell & Murray,
 407 Perry-Payne Bldg., Cleveland, Ohio.
 WILCOX, GORDON HOWARD, Min. Engr. Kyle, McDowell Co., W. Va.
 WILLIAMS, PERCY THOMAS, Min. Engr., Pacific Northwest Coal Co.,
 1318 Alaska Bldg., Seattle, Wash.
 WISER, OBA, Met. Engr. Hayden Plant, Ray Cons. Copper Co., Hayden, Ariz.
 WOLF, WILLIAM ALBERT, Foreman Cyanide Plant. Instructed to hold all mail.
 WOLVIN, AUGUSTUS B., Pres Butte-Duluth Mining Co., Butte, Mont.
 WOOD, ALEXANDER HAMILTON, Coal Operator and Cons. Min. Engr.,
 Petros, Morgan Co., Tenn.
 WOODRUFF, E. G., Geol. Producers Oil Co., Houston, Texas.
 YATSEVITCH, MICHAEL G., Engr. of Technology, 57 Reservoir St., Cambridge, Mass.

Associate Members

- CLARKE, TALCOTT HUNT, Asst. Purchasing Agent, Republic Rubber Co.,
 Youngstown, Ohio.
 SHAFFER, WILLIAM B. 164 Pitt St., Sydney, N. S. W., Aust.
 TINSLEY, RICHARD PARRAN, Treas. Standard Oil Co., 26 Broadway, New York.
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 Salt Lake City, Utah.

Junior Members

- COCHRAN, RALF S., Student 30½ Mellen St., Cambridge, Mass.
 FARNHAM, LLOYD L., Student State School of Mines, Rapid City, S. D.
 FELL, HAROLD BERTELS, Civil Engr. 325 So. River St., Wilkes-Barre, Pa.
 FINAGIN, JOSEPH COOPER, JR., Met. Campbell Hill, Ill.
 LADOO, RAYMOND B., Student 14 Thayer Hall, Harvard Univ., Cambridge, Mass.
 PLUMB, EDMUND TUTTLE, Student 310 York Street, New Haven, Conn.
 SCHINDLER, DONALD F., Student 207 Bernard Ct., Madison, Wis.
 VAN DORP, G. H., Student, Kappa Sigma House, Colo. School of Mines,
 Golden, Colo.
 WALTERS, MAURICE B., Student 711 Yale Station, New Haven, Conn.

CANDIDATES FOR MEMBERSHIP

The following persons have been proposed during the period Apr. 10 to May 10, 1914, for election as members of the Institute. Their names are published for the information of members and associates, from whom

the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

J. F. Aguilar-Revoredo, Bolivia, So. America.

Proposed by Franz Germann, A. D. R. Galloway, P. A. Seibert.

Born 1886, Lima, Peru. 1895-02, College des Sacres Cœurs, Lima, Peru. 1903-07, Escuela Especial de Ingenieros, Lima. Degree of Agrumensor Civil and Ingeniero de Minas. 1907, Engaged in various capacities at Casapalca, Yaulu, Cerro de Pasco, qualifying for my degree. 1908, 2nd Engineer on the exploration for the Ucayaly R. R., Government of Peru. 1909-10, District Engineer on survey and construction of the Lima-Huacho R. R. 1910-12, with the Ferro-obamba, Ltd., Peru, Engineer and Engineer in charge. 1912-13, Mine Manager of the Empresa Minera Huanuni, Simon I. Patino, Bolivia.

Present position: Technical adviser to Simon I. Patino (mining and railway sections).

Milton Arthur Allen, Juneau, Alaska.

Proposed by Arthur L. Pearse, R. Rawdon Johnson, G. T. Jackson.

Born 1890, Denver, Colo. 1897-05, Grammar school, Denver, Colo. 1905-08, Kings College, London Univ., London, England. 1908-12, Royal School of Mines, London, England. 1908-12, Imperial College of Science and Technology, London, England. Degrees, A. R. S. M., D. I. C., B. S. 1912-13, Chief Sampler, Alaska Gastineau Min. Co. 1912, miner, Gastineau Mining Co. 1913, Shift Boss, Gastineau Mining Co.

Present position: Shift Boss, Alaska Gastineau Mining Co., Juneau, Alaska.

Frederick C. Alsdorf, San Francisco, Cal.

Proposed by Abbot A. Hanks, A. W. Geizer, F. A. Gowing.

Born 1871, Utica, Ohio. 1892, Ohio State Univ., Columbus, Ohio; E. M. 1892-94, Chemist, Creighton Gold Min. & Mill. Co., Creighton, Ga. 1895-96, Supt., Duquesne Gold Min. & Mill. Co., Central City, Colo. 1896-98, Supt., Mt. Wilson Gold & Silver Mining Co., Telluride, Colo. 1898-1900, Consulting Engineering work. 1900-01, Supt., Commercial Min. Co., Joplin, Mo. 1901-03, Consulting Engineering. 1903-04, Supt., Stevens Copper Co., Clifton, Ariz. 1904-06, Supt., Cuprite Copper Co., Clifton, Ariz. 1906-08, Supt., Miami Copper Co., Globe, Ariz. 1908-13, Consulting Engineering.

Present position: Mining Engineer.

J. Owen Ambler, Douglas, Ariz.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1883, Tenn. Rolling Mills, Ky. 1896, Grammar schools, St. Louis, Mo. 1900, Central High School, St. Louis, Mo. 1904, Missouri School of Mines, Rolla, Mo.; B. S. in 1905. 1905-06, Instrument man and Asst. Engr., U. S. Reclamation Service, Montrose, Colo. 1906-13, Draughtsman, Night Supt. smelter, Asst. Supt. smelter (day), Cananea Cons. Copper Co., Ltd., Cananea, Son., Mexico.

Present position: Met., C. & A. Smelter, Douglas, Ariz.

Carl Nathaniel Anderson, Portland, Ore.

Proposed by H. O. Hofman, Robert H. Richards, Carle R. Hayward.

Born 1891, Red Wing, Minn. 1897-1907, General education received in the public schools of Portland, Ore. 1908-13, Oregon State Agricultural College; B. S. 1909, Summer, operating a placer mine in Union County, Ore. 1910, Summer, at Camp Carsen, Union County, Ore. 1911, Summer, Cœur d'Alène district, especially at the Morning mine, Mullan, Idaho. 1912, Summer, practical mining. 1913, Summer, Field, Asst., Oregon Bureau of Mines and Geology. 1913-14, Asst., Dept. of Mining and Metallurgy, Massachusetts Inst. of Technology.

Present position: Mining Engineer for C. A. Hafner, Kinsman, Ohio.

Frank Linden Antisell, Perth Amboy, N. J.

Proposed by J. W. Allen, W. S. Harper, B. B. Thayer.

Born 1875, San Francisco. General collegiate course in mechanical engineering. Associated with father in metallurgical work. Systematically studied electricity, chemistry, correspondence course, supplemented with special college course. 1900, Draftsman, American Smelting & Refining Co. 1902, De Lamar Copper Works Laboratory. 1904, Constructing Engineer, Raritan Copper Works.

Present position: 1908 to date; Asst. Supt., Raritan Copper Works.

Andrew Moir Balfour, Republic, Wash.

Proposed by S. H. Richardson, J. C. Haas, L. K. Armstrong.

Born 1871, Dunfermline, Scotland. 1885, Grad. high school. 1887-89, Edinburgh Univ., Chemistry. 1894, Miner, Gagnon mine, Colo. Min. & Smelt. Co., Butte, Mont. 1898, Leonard mine, Boston & Mont. Co. 1900, Foreman, Kearsarge mine, Virginia City, Mont. 1901, Foreman, Toledo mines, Bismarck Nugget Gulch Min. Co., Sheridan, Mont. 1902, Foreman, U. S. Grant mine and mill, Virginia City, Mont. 1904, Prospecting. 1905, Engineering practice in Butte, Mont., Balfour & Holley, Asst. County Surveyor. 1906, Shift boss, Southern Cross cyanide mill. 1907, Prospecting near Anaconda, Mont. 1908, Mill Foreman, cyanide mill, Tortman, Mont. 1908, Mine Foreman, Alabama mine, Tortman, Mont. 1909-12, Prospecting. 1913, Shift boss, Harper mill, Republic.

Present position: Prospecting.

Charles Smith Beck, Morenci, Ariz.

Proposed by M. H. McLean, Frank W. McLean, Frank D. Rathbun.

Born 1886, Lewiston, Minn. 1907-11, Engineer of Mines, Minn. School of Mines. 1911-12, Millman, Miami Copper Co. 1912-13, Engr., Morenci Water Co.

Present position: 1913 to date; Engineer, Detroit Copper Co., Morenci, Ariz.

Raymond A. Bingham, Bartlesville, Okla.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1888, Dorchester, Mass. 1894-1907, Grad. of Watertown, Mass., public schools. 1907-11, Missouri School of Mines, Rolla, Mo.; B. S. 1911, Mill testing and timekeeper, Vinegar Hill mine, Vinegar Hill Zinc Co., Galena, Ill. 1911-12, Timekeeper, Wilkenson mine, Vinegar Hill Zinc Co., Benton, Wis. 1912-14, Chemist, Lanyon-Starr Zinc Co., Bartlesville, Okla.

Present position: Chemist, Bartlesville Zinc Co., Bartlesville, Okla.

Guy Norman Bjorge, Globe, Ariz.

Proposed by John M. Boutwell, W. H. Emmons, L. C. Graton.

Born 1887, Underwood, Minn. 1900-04, High school at Duluth, Minn. 1908-12, School of Mines, Univ. of Minn.; E. M. 1904-08, Mercantile business with Bjorge & Sjordal, Underwood, Minn. 1912, Engineer, Pickands, Mather & Co., Gilbert, Minn. 1912-13, Geologist, Caribbean Petroleum Co., Venezuela, S. A. 1913, Asst. Chief Engineer, Pickands, Mather & Co., Gilbert, Minn.

Present position: 1913 to date; Geologist, Old Dominion Copper Mining & Smelting Co., Globe, Ariz.

True Walter Blake, Novinger, Mo.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1890, Chicago, Ill. 1907, Grad. Austin High School, Chicago, Ill. 1911, Missouri School of Mines, Rolla, Mo.; B. S. 1911-12, Levelman and later transitman, Arizona Territorial Survey. 1912, Classifier man, Ray Cons. mill, Hayden, Ariz. 1912, City surveying, Chicago, Ill.

Present position: 1912 to date; Mining Engineer, Rombauer Coal Co., Novinger, Mo.

George M. Brown, McAlester, Okla.

Proposed by C. M. Young, J. J. Rutledge, James W. Malcolmson.

Born 1889, Providence, R. I. 1894-1902, Common school, near Pleasanton, Kan. 1902-07, Grad. Pleasanton High School. 1908-12, Grad. Univ. of Kansas, Lawrence, Kan.; B. S. 1910, Summer, Engineering dept., City of McAlester, Okla. 1911, General survey and construction work, Chinook Coal Co., Ltd., Lethbridge, Alberta. 1911-12, Instructor in Mining Laboratory, Univ. of Kansas, Lawrence.

Present position: 1912 to date; Mining Engr., Osage Coal & Mining Co., Great Western Coal & Coke Co., Samples Coal & Mining Co., and the Hailey Ola Coal Co., McAlester, Okla.

R. I. Brown, San Jose, Costa Rica.

Proposed by Robert Gordon, P. G. Spilsbury, Robert S. Hancel.

Born 1889, Bucksport, Me. 1905-09, East Maine Conference Seminary, Bucksport, Me. International Correspondence Schools, Scranton, Pa. Asst. to Engineer, railroad construction. 1911, Cukra Co., Toronto, Ltd., Bluefields, Nic. 1912, United Fruit Co., Bluefields, Nic. 1913, River Plate Co., San Jose, C. R. 1913-14, Surveying, inspections, etc., for private companies in Costa Rica.

Present position: Mine Surveyor, Montezuma Mines of Costa Rica.

Murray Fred George Burrows, Leederville, W. Australia.

Proposed by H. R. Sleeman, Thomas T. Read. (One signature required.)

Born 1888, Barnawartha, Victoria, Aust. 1907, Educated privately. 1908, Matriculated Adelaide Univ., honors in Mathematics and Literature. 1907-09, Technical instruction at Perth Technical College, W. A. 1909, Gained W. A. Government Assayer's Certificate. Awarded Gov. Senior Scholarship, W. A. School of Mines. 1910, Grad., in Metallurgy, W. A. School of Mines, Kalgoorlie. Awarded Critchly Parker special prize in Metallurgy. 1910, Associated with G. O. G. Larcombe, A. S. T. C., F. G. S., in preparation of monograph "Geology of Kalgoorlie." 1911, General laboring in mill at Golden Horseshoe Estates Co., Ltd. Solutionist in same mill., Asst. Chemist same company. 1912-13, Chemist and Surveyor, Whim Well copper mines, Whim Creek. In charge of experimental work at magnetic Concentrating plant.

Present position: Chief Chemist, Whim Well Copper Mines, Ltd., Whim Creek, W. A.

Evan Fraser Campbell, Tyrone, N. M.

Proposed by Walter Douglas, Gerald F. G. Sherman, Alexander V. Dye.

Born 1884, Newport, R. I. 1897-1902, Bishops College School, Lenoxxville, P. Q., Canada. 1902-06, Harvard Min. Dept., Lawrence Scientific School; B. S. 1906-08, Sampling, mapping and assaying at Pachuca, Hidalgo, and Uruapan, Michoacan. 1908-10, Mexico Mines of El Oro, Ltd., mine sampler, shift boss, night foreman; transferred in 1909, as foreman of Somera mine, same district, El Oro Min. & Ry. Co. 1911-12, Copper Queen Cons. Min. Co., miner, timberman, machine man, asst. in engineering dept., mine sampler; transferred in latter part of 1912 to present position.

Present position: Chief Engineer, Burro Mountain Copper Co., Tyrone, N. M.

Frank Gamble Carrington, Anniston, Ala.

Proposed by Erskine Ramsay, J. W. Shook, J. M. Garvin.

Born 1892, Jasper, Ala. 1907-09, Ala. Presbyterian College, Anniston. 1909-10, Vanderbilt Univ., Nashville. 1910-11, A. P. I. Chemistry and Metallurgy, Auburn, Ala. 1912-13, Chemist, Union Foundry Co., Anniston. 1913, Standard Foundry Co., Construction Engineer.

Present position: Sec'y. and Asst. Mgr., Standard Foundry Co., Anniston, Ala.

Edward Ernest Carter, Quartzbury, Idaho.

Proposed by Etheredge Walker, Charles E. Locke, Edward E. Bugbee.

Born 1879, Lowell, Mass. Common schools, 2 yrs. Lowell Textile. 1 yr. Missouri School of Mines. 1898-01, Apprentice, Lowell & Ariz. mine, Bisbee, Ariz. 1901-02, Mo. School of Mines. 1902-06, Supt., Big Horn Mine, Shoemaker, Ariz. 1906-07, Sampling mines for F. Hanchett, Lowell, Mass. 1907-08, Sullivan Mach. Co. 1908-09, Traylor Engineering Co.

Present position: 1909-14, Mgr. Gold Hizz Mine, Quartzbury, Idaho.

Clarence Hawley Chester, Steelton, Pa.

Proposed by Edward N. Trump, E. C. Witherby, W. H. Blauvelt.

Born 1878, Minneapolis, Minn. 1895-1900, Univ. of Penn.; A. B. 1899, By-product operation Semet Solvay Co., Dunbar, Pa. 1900-03, Chemist, Semet Solvay Co., Ensley, Ala. 1903-05, Asst. Supt., Semet Solvay Co., Ensley, Ala. 1905-12, Asst. Supt., Semet Solvay Co., Lebanon, Pa. 1912-13, Asst. Supt., Semet Solvay Co., Steelton, Pa.

Present position: 1913 to date; Supt., Semet Solvay Co., Steelton, Pa.

Paul Richardson Cook, Buckhorn, Nev.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1884, Crawford Co., Iowa. 1901, Grad. Rolla High School, Rolla, Mo. 1907, Mo. School of Mines; B. S. 1903-05, Carpenter, Rolla, Mo. 1906, Construction, Camp Bird Mill, Gold Prince Mill, Colo. 1907, Prospecting, mining, Elko Co., Nev. 1908, Construction, Steptoe Valley Smelter. 1909-10, Transitman, O. S. L. R. R. 1911, Assayer, refineryman, millman, Nevada Hills mill, Fairview, Nev. 1912-13, Asst. Supt., Mexican mill, Virginia City, Nev.

Present position: Shift boss, Buckhorn Mill and Cyanide Plant, Buckhorn, Nevada.

Guy Walter Crane, Eureka, Utah.

Proposed by C. K. Leith, Alexander N. Winchell, Charles R. Van Hise.

Born 1876, Monroe, Wis. 1896, Grad. high school, Monroe, Wis. 1901-06, Student, Univ. of Wis. 1912-13, B. A., Univ. of Wis. 1902-06, Open seasons, iron ore explorations in Canada for C. R. Van Hise, and C. K. Leith. 1906-07, Geol., Missouri Geol. Survey. 1907-09, Geol., Federal Lead Co., Flat River, Mo. 1909-12, Geol., Missouri Geol. Survey. 1912-13, Geological work in Missouri and Arkansas for J. R. Finlay.

Present position: Geologist, Chief Cons. Mining Co., Eureka, Utah.

Arthur Harrison Crouk, Rosiclare, Ill.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1887, Omaha, Neb. 1908-12, Grad., Missouri School of Mines; B. S. 1906-08, Engrg. Dept., Union Pacific R. R. Co. 1908-12, Student, Missouri School of Mines. 1912, Engrg. Dept., Union Pacific R. R. Co.

Present position: 1912 to date, Engineer, Rosiclare Lead and Fluor Spar Mines, Rosiclare, Ill.

Walter H. Cunningham, Huntington, W. Va.

Proposed by James S. Cunningham, Howard N. Eavenson, E. W. Parker.

Born 1882, Allentown, Pa. 1898-99, Kiskiminetas Springs School. 1899-1900, Lehigh Univ., So. Bethlehem, Pa. 1901-04, Lehigh Univ., So. Bethlehem, Pa. 1898-1900, Summer work, Berwind-White Coal Mining Co. 1900, Special apprentice, P. R. R., Altoona, Pa. 1901-04, Summers, Berwind-White Coal Mining Co. 1905, Engineering Dept., Pocahontas Cons. Coal Co. 1906, Engrg. Dept., New River & Pocahontas Cons. C. Co. 1906-07, Mechanical Engr., Davis Coal & Coke Co. 1907, Supt., Thomas Davis Coal & Coke Co. 1907-10, Mgr. Mines West Ky. Coal Co. 1910-12, Genl. Mgr., West Ky. Coal Co. 1912, Consulting Engr., Huntington, W. Va., and Ashland, Ky.

Present position: Consulting Engr., Huntington, W. Va., and Ashland, Ky. Secy.-Treas., Kentucky Mine Owners' Assn., Ashland, Ky.

George H. Dern, Salt Lake City, Utah.

Proposed by J. C. Dick, R. C. Gemmell, W. Lee Heidenreich.

Born 1872, Dodge Co., Neb. 1893-94, Univ. of Nebr. 1895-1900, Treas., Mercur Gold Mining & Milling Co. 1900-01, Asst. Gen. Mgr., Cons. Mercur Gold Mines Co. 1901-13, Gen. Mgr., Cons. Mercur Gold Mines Co.

Present position: 1913 to date; General Mgr., Mines Operating Co.

Cecil Fitch, Eureka, Utah.

Proposed by Walter Fitch, C. F. Moore, J. R. Finlay.

Born 1885, Beacon, Mich. 1907, Mich. College of Mines; B. S. 1908, Mich. College of Mines; E. M. 1908-09, Performed miscellaneous duties consisting of machine mining, surveying, assaying, black-smithing, hoisting engineer, bossing, etc. for Little Chief Mining Co., Eureka, Utah.

Present position: 1910-14, Supt., Chief Cons. Mining Co., and Eureka City Mining Co., Eureka, Utah.

Walter Fitch, Jr., Eureka, Utah.

Proposed by Walter Fitch, C. F. Moore, J. R. Finlay.

Born 1887, Beacon, Mich. 1909, Michigan College of Mines; B. S., E. M. 1909, Machine miner, Chief Cons. mine; 1910, Machine mining and surveying. 1911-12, Machine mining, shift bossing and surveying. Afterward Assistant Supt.

Present position: Asst. Supt., Chief Cons. Mine, and Eureka City Min. Co.

Harold Percy Ford, Lake Linden, Mich.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1889, St. Joseph, Mo. 1894-1902, Common school training, Hall School, St. Joseph, Mo. 1902-06, College preparatory course, high school, St. Joseph, Mo. 1907-10, Mining Engineering course, School of Mines and Metallurgy, Rolla, Mo. 1910-11, Elective course, Mont. State School of Mines, Butte City, Mont. 1911-12, Mining Engineering course, School of Mines and Metallurgy, Rolla, Mo.; B. S. 1906-07, Clerk, First National Bank of St. Joseph, Mo. 1910, Miner, Leonard mine, Butte City, Mont., Boston & Mont. Min. Co. 1911, Asst. Geologist, Missouri Bureau of Geology and Mines. 1911-12, Student Asst. in Physics to Prof. Austin Lee McRae, Rolla, Mo. 1912, Miner, at Wilkinson Mine, Benton, Wis. Employed by Vinegar Hill Zinc Co.,

Present position: 1912 to date; Calumet & Hecla Stamp Mills, Lake Linden, Mich. Asst. in experimental work, Hydrometallurgy of Copper.

John Kavanaugh Forman, Miami, Ariz.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1884, Winchester, Ky. 1903-04, Northwest Missouri College, Albany, Mo. 1905-06, Central College, Fayette, Mo. 1907-10, Missouri School of Mines; B. S. 1910-12, Lanyon Starr Smelting Co., Bartlesville, Okla. 1912, Ore Agent at Kansas City, Tri Bullion Smelt. & Dev. Co., Wellington Mines Co., Butte & Superior Copper Co. 1912-13, Chemist, Butte & Superior Copper Co., Butte, Mont. 1913, Flotation millman, Butte & Superior Copper Co.

Present position: Flotation millman, Inspiration Cons. Copper Co., Miami, Ariz.

E. L. Fortney, Cle Elum, Wash.

Proposed by John N. Pott, Raymond P. Tarr, Harry Boyle.

Born 1875, Clarksburg, W. Va. 1893, High school. 1898, Normal school, Fairmont, W. Va. Home study, and correspondence course. 1902-06, Transitman for Lehigh Valley Coal Co., Centralia, Pa. 1906-08, Div. Engr., New River Collieries Co., Rush Run, W. Va.

Present position: Resident Engr., Northwestern Improvement Co.

Francis James Gallagher, New York, N. Y.

Proposed by Robert Peele, C. Edwin Nighman, Charles P. Berkey.

Born 1885, Ireland. 1903-05, Brookline High School, Brookline, Mass. 1913, Columbia Univ., New York. 1905-06, Civil Engr., A. P. Milne, Lexington, Mass. 1906-08, C. E., W. B. Foster, Hingham, Mass. 1908-09, So. Pac. R. R. Co., San Francisco, Cal. 1909-10, C. E. with City Engr., Los Angeles, Cal. 1910-11, Chief Engr., Palo Verde Land & Water Co., Blythe, Cal. 1911-13, Mining Engr., Ely Central Copper Co., Ely (Perth), Nevada.

Present position: Special student in School of Mines, Columbia Univ., New York.

Charles Clifton Griggs, Eureka, Utah.

Proposed by J. C. Dick, W. Lee Heidenreich, R. C. Gemmell.

Born 1875, Nebraska. 1897, Univ. of Nebraska, Lincoln, Neb.; B. S. 1894-97, Surveying, Burlington R. R., Lincoln. 1897-1900, Assayer and Chemist, Dexter Gold Mining Co., Tuscarora, Nev. 1900-01, Electrician R. R. construction, Kilpatrick Bros., Wyoming. 1900-02, Mill Supt., Wasp No. 2 Min. Co., Deadwood, S. D.

Present position: 1904 to date; Supt., Uncle Sam Cons. Min. Co., and May Day Mining & Milling Co., Eureka.

Herman Carl Hase, Miami, Ariz.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1881, Dresden, Germany. 1887-1896, Graded schools, Kansas City, Mo. 1896-1900, Manual Training High School, Kansas City, Mo. 1903-08, Mo. School of Mines, Rolla, Mo.; B. S. 1906-07, Engineering Corps, Greene Cananea Copper Co., Cananea, Mexico. 1908-10, Concentrator and assay office, Detroit Copper Mining Co., Morenci, Ariz. 1910-11, Concentrator, Tomboy Gold Mines Co., Telluride, Colo. 1911-12, Concentrator and crusher, Old Dominion Copper Mining & Smelting Co., Globe, Ariz.

Present position: Inspiration Cons. Copper Mining Co., engineering and flotation.

Lewis H. Haupt, Palmerton, Pa.

Proposed by Harrison Souder, B. E. McKechnie, Francis P. Sinn.

Born 1889, Philadelphia, Pa. 1895-1902, Graded schools in Philadelphia. 1902-05, Central Manual Training School of Phila. 1905-09, Univ. of Penn. in Mechanical Engineering; B. S. 1905, Three months with Franklin & Clarke, Phila., civil engineers. 1906-07, Three months each summer with Cornwall Ore Bank Co., Cornwall, Pa. 1908, Three months transitman for Tidewater Oil Co. 1909, Two months Baldwin Locomotive Works in erecting shop, through Arthur L. Church. 1909-10, Instructor in Mech. Engrg., Univ. of Pa. 1910-13, Engineer for Cornwall Ore Bank Co.

Present position: Employed in Service and Maintenance Dept. of New Jersey Zinc Co. of Pa.

John Ambrose, Heskett, Vict., Australia.

Proposed by E. G. Banks, A. Jarman, R. E. Williams.

Born 1889, Hawthorn, Victoria. 1895-1905, Schools in Queensland, N. S. W., and Victoria. 1905-08, Under father's tuition, metallurgy of iron and steel. 1911, Course on metallurgy and manufacture of iron and steel with International Correspondence Schools. 1913, Went through Dr. Pritchard's course on iron and steel at Melbourne School of Mines. 1905, Coal mining, boring for coal and coke making patent process, Bluff colliery, Queensland. 1907-11, Installing and working crucible steel plants in various portions of Queensland. N. S. Wales, and Victoria. 1912, With Mt. Lyell Co., on research work, also erecting for them a crucible steel plant. 1912, Also at Mt. Bischoff, on same work.

Present position: Experimenting with iron sands of New Zealand, at Waihi Co. workers.

Henry Claude Heise, Weedon, Quebec, Canada.

Proposed by William Hague, A. D. Foote, Arthur B. Foote.

Born 1886, Gardnerville, Nev. 1903-07, High school; Reno, Nevada. 1907-11, Univ. of Nevada; B. S. 1911, Underground work, surveyor's asst., timekeeper, North Star Mines Co., Grass Valley. 1912-14, Specimen Boss, North Star Mines Co., Grass Valley.

Present position: Assayer, East Canada Smelting Co.

Ott Fleming Heizer, Idaho Springs, Colo.

Proposed by Richard A. Parker, W. E. Renshaw, W. M. H. Woodward.

Born 1881, Staunton, Va. Public and high schools, Washington, D. C. 1900-04, Univ. of Nevada, School of Mines, pursued special work during last year instead of regular course. 1905-06, U. S. Reclamation Service, Hazen, Nev. 1907-08, Assayer, surveyor, Nevada Hills mine, Fairview, Nev. 1909-10, Supt., Nevada Hills mine, Fairview, Nev. 1911-12, Supt. Seven Troughs Coalition Min. Co., Seven Troughs, Nev. 1913, Supt., United Mines Corp., Tuolumne, Cal.

Present position: Gen. Mgr., Argo Reduction & Ore Purchasing Co., Idaho Springs, Colo.

August F. Hilleke, Ensley, Ala.

Proposed by E. C. Witherby, J. D. Pennock, W. H. Blauvelt.

Born 1869, Ft. Wayne, Ind. Common school and practical experience. Past fourteen years by-product coke business. Prior to that time gas business. Supt., Semet Solvay Co.; Supt., Augusta Gas Co., Augusta, Ga.

Present position: Dist. Supt., Semet Solvay Co.

Theodore P. Holt, Park City, Utah.

Proposed by J. C. Dick, R. C. Gemmell, W. Lee Heidenreich.

Born 1882, Talent, Ore. 1907, Univ. of Ore.; B. S. 1908, Univ. of Ore.; E. M. 1909, Univ. of Utah; M. S. 1909, Metallurgist for the Cons. Mercur Gold Mines Co. 1910, Utah Leasing Co. 1910-12, Instructor in milling at Utah State School of Mines.

Present position: Supt. of Mines, operating company.

Louis Orrin Howard, Salt Lake City, Utah.

Proposed by J. C. Dick, Ernest Gayford, E. R. Lidvall.

Born 1884, Thompson's, Conn. 1903-06, Harvard College. 1906-07, Lawrence Scientific School. 1907, A. B., Harvard College. 1907-09, Grad., School of Applied Science of Harvard Univ.; M. E. 1907-09, Austin Teaching Fellow, Mining and Metallurgy, Harvard Univ. 1909-11, Instructor in Metallography and Ore Treatment, Case School of Applied Science, Cleveland, Ohio. 1912 (six months), Supt., Camp Vanura, Green River, Utah, for Angus Cameron, Gwynedd, Pa., agent for Philadelphia syndicate, now Radium Company of America.

Present position: 1912 to date, Associate Editor, *Salt Lake Mining Review*.

Dibrell Pryor Hynes, Chicago, Ill.

Proposed by H. L. Hollis, W. H. Emmons, G. H. Cox.

Born 1884, Arkansas. 1902, Ft. Smith, Ark., High School. 1905-08, Missouri School of Mines; B. S. 1910-11, Univ. of Chicago. 1908, Engr., with L. S. Griswold. 1908-09, Instructor in Chemistry, Missouri School of Mines. 1909-10, Engineer, Socorro Mines Co., Mogollon, N. M. 1910, Experimental Engr. with C. C. C. Co., Cananea, Mexico. 1910, Junior Geologist, U. S. Geological Survey.

Present position: Asst. to H. L. Hollis.

John Frederick Johnson, Eureka, Utah.

Proposed by Walter Fitch, Lewis A. Jeffs, J. R. Finlay.

Born 1887, Quincy, Mass. 1900-04, Adams Academy. 1904-07, Harvard Univ. 1907-10, Machine mining, timbering, sampling, surveying and mapping, Centennial-Eureka mine.

Present position: Shift boss, Chief Cons. mine, Eureka, Utah.

George Kingdon, Cananea, Mexico.

Proposed by James Douglas, T. F. Cole, J. S. Douglas.

Born 1862, Cornwall, England. Practical experience. 1870, Miner, Dolcoath Mine. 1878, Calumet & Hecla, Butte, Mont. 1881, Tombstone, Globe, Ariz. Bisbee, Silver City, N. M. 1904, Nacozari, Sonora, Mine Foreman. 1907, O. D., Globe, Mine Foreman.

Present position: Genl. Supt., Cananea Cons. Copper Co., Cananea, Sonora, Mexico.

Frank Eugene Lathe, Grand Forks, B. C., Canada.

Proposed by E. J. Carlyle, F. F. Frick, Louis V. Bender.

Born 1883, Lacolle, Quebec. 1904, McGill Univ., Chemistry course; B. A. 1907, McGill Univ., Metallurgy course; B. S. 1908-14, Laboratory, Granby Cons. Min., Smelt. & Power Co., Grand Forks, B. C., working in all depts.

Present position: Chief Assayer, Granby Cons. Min., Smelt. & Power Co.

Julcs A. Lefebvre, Bruxelles, Belgium.

Proposed by A. DeDeken, Axel Sahlin. (One more signature required.)

Born 1874, Verviers, Belgium. 1891, College, certificate, 1898, Certificate of Mining Engineer, Université de Liège, Belgium. 1898-1900, Engineer, Mines d'or du Katchkar, Siberia. 1900-06, Chief Engineer, Société Russe de fabrication de tubes, Ekaterinoslow, Russia. 1906-07, Elected to prospect Madagascar. 1907-10, Prospecting Spain, Portugal, Russia, Hungary, Belgian Congo, etc., for various groups. 1910-12, Gen. Mgr., Minière Congolaise (Bangu Nagelmackers), Brussels.

Present position: Managing director, Minière Congolaise; director of La Productora Coto San Antonio, Spain; director of La Société Minière et Industrielle du Tamega, Portugal; director of Mines d'Etain d'Avelleda, Portugal. Member of La Société Géologique de Belgique; member of L'association des Ingenieurs de Liège.

William Rufus Lindsay, Treadwell, Alaska.

Proposed by E. P. Kennedy, R. G. Wayland, Robert A. Kinzie.

Born 1885, Farmingdale, S. D. 1905, Chainman and computer, Seattle Interurban Ry. 1906, Transitman, location Alaska Pac. Ry. and Copper River & N. W. Ry., Katalla, Alaska. 1907, Placer mining, Nizina, Alaska. 1908, Mach. helper and machine man, A. T. G. M. Co., Treadwell, Alaska. 1909, Univ. of Washington, Seattle, Wash.; B. S. 1909-10, Mills and assay office, Treadwell, Alaska. 1910-12, Surveyor and engineer, Treadwell, Alaska. 1912-14, In charge of reconnaissance and construction, hydro-electric plant, Treadwell, Alaska.

Present position: Engineer, Alaska Treadwell Gold Mining Co., Treadwell, Alaska.

Charles W. McMeekin, Nevada City, Cal.

Proposed by Charles C. Derby, H. Foster Bain, F. Lynwood Garrison.

Born 1862, Newport, Ky. 1885, Completed special course in Engineering, Univ. of Iowa. 1885-87, Instrumentman and Division Engineer in Missouri and Kansas for Chicago, Rock Island & Pacific Ry. 1887-88, Estimating Engineer, Chicago, Santa Fe & California Ry., in Chicago. 1888-92, Chief Engineer for the Eureka Coal Min. Co., in Des Moines, Iowa. 1892-93, Engineer in charge of construction masonry dam for Butte City Water Works, Butte, Mont., and bridge construction on Gouverneur & Oswegatchie Ry., New York. 1893-95, Chief Engineer of Des Moines Northern & Western Ry. 1895-98, Chief Engineer, Iowa Central Ry. 1898-99, Principal Asst. Engineer, Anaconda Min. Co. (Waterworks Dept.), Anaconda, Mont. 1899-1900, Consulting Engineer, Snow Shoe Mining & Milling Co., Libby, Mont. 1900-07, Engineer and Vice President, Britannia Copper Syndicate, Ltd., Vancouver, B. C. 1909-13, Gen. Mgr., Croesus Gold Min. & Mill. Co., Alleghany, Cal.

Present position: 1913 to date; Private practice, in Nevada City, Cal.

Miles S. Milward, Mocorito, Sin., Mexico.

Proposed by William D. Blackmer, Frank A. Keith, Seeley W. Mudd.

Born 1883, Cape Town, So. Africa. 1888-99, Private schools in England and various parts of the U. S. General and classical education. 1899-1902, Three years private individual instruction—chemistry, chemical research, physics, surveying, etc., in San Francisco, Cal., and vicinity. 1905-08, Public Assayer with Gorrill, Milward & Gorrill, Rhyolite, Nev. 1908, Millman, refining etc., Montgomery Shoshone Mining Co. 1908-09, Assayer and surveyor, Homestake King Cons. Min. Co. 1909, Assayer, sampler, surveyor, Mayflower Bullfrog Cons. Mining Co. 1910, Asst. Mgr., Eclipse Development Co.

Present position: 1911¹/₂ to date; General Supt., El Palmarito Leasing Co.

Ben Alfred Mizen, Crosby Minn.

Proposed by William Wearne, O. B. Warren, A. P. Silliman.

Born 1879, London, England. Kensington High Grade School, London, England. 1899-01, Asst. Chemist, Oliver Iron Mining Co. 1902-04, Asst. Engr., Oliver Iron Mining Co. 1904-05, Constructing Escanaba Crusher Plant. 1905-08, Supt. Explorations in New York State, U. S. Steel Co., Engineer, Ore. 1908-10, Concentrating Dept. Oliver Iron Mining Co., Canisteo, Minn. 1910-14, Supt., Crosby Exploration Co., Cuyuna Range, Minn.

Present position: Consulting Engineer.

Alfred C. North, Galt, Cal.

Proposed by Robert Nye, Arthur B. Foote, Thomas V. Reeves.

Born 1884, Riverside, Cal. General preparatory education received in public schools of Riverside Co., Cal. 1904-10, Univ. of Cal.; B. S. 1902-04, Rodman to transitman, Pacific Light & Power Co., Los Angeles, Cal. Summers, 1904-07. Field Asst., U. S. Geol. Survey. 1907-08, Mucker to timberman, Empire Mines, Grass Valley, Cal. 1909-10, Timberman. 1910, Summer, Surveyor, McNamara. West End, Midway Mines, Tonopah, Nev. 1910-12, Sampler, shift boss, surveyor, etc., Butters Divisadero Co., El Salvador, C. A.

Present position: Secy., Mines, Murcielago Mining Co., Galeras, Honduras.

Frederick J. Pack, Salt Lake City, Utah.

Proposed by J. F. Kemp, Charles P. Berkey, Alfred J. Moses.

Born 1875, Bountiful, Utah. 1904, Univ. of Utah; B. S., M. E. 1905, Columbia Univ.; A. M. 1906, Columbia Univ.; Ph. D. Since 1906 I have done a considerable amount of geological expert work, chiefly in the mines of Utah; all of my time outside of duties at the university has been devoted to this work.

Present position: Deseret Professor of Geology, Univ. of Utah.

Arthur B. Parsons, Candor, N. C.

Proposed by H. A. Morrison, Benjamin F. Tibby, Benjamin H. Case.

Born 1887, Salt Lake City, Utah. 1893-1905, Public schools of Salt Lake City including high school. 1905-09, Utah School of Mines; B. S. 1909-10, Goldfield Cons. Mining Co.

Present position: 1911-14; Supt., Cyanide Plant, Candor Mines Co.

Elmer Pfouts, Butte, Mont.

Proposed by F. F. Sharpless, George E. Farish, C. W. Goodale.

Born 1876, Virginia City, Mont. Common school education. 1899-1904, Boston & Montana Smelter, Great Falls, Mont. 1904-07, M. L. McDonald, Tonopah, Nevada. 1907-08, McQuatters & Shepherd, Chihuahua, Mexico. 1911-12, Dobie Mines, Ltd., Porcupine, Ont. 1912-13, Mother Lode Mine, Sheep Creek, B. C. 1913, Elko Prince, Midas, Nev.

Present position: Supt. Black Bear Mine, Telluride, Colo.

Evan E. Price, Salt Lake City, Utah.

Proposed by J. C. Dick, R. C. Gemmell, L. S. Austin.

Born 1876, Wales, G. B. 1904, Grad., Missouri School of Mines; E. M. 1904-06, Asst. Engr., Utah Copper Co. 1906-07, Gen. engrg., Nev. 1907-09, Foreman, Yampa mine, Bingham, Utah. 1909-10, General practice. 1910-12, Field Engr., Gum Thompson Co.

Present position: 1912 to date; Supt. Lower Mammoth Mining Co., Mammoth, Utah.

Donald Hewson Radcliffe, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1889, Kansas City, Mo. 1905-07, Graduated Kirkwood Military Academy, Kirkwood, Mo. 1907-13, Missouri School of Mines; B. S. 1908, Underground diamond driller, Federal Lead Co., Flat River, Mo. 1909, Rodman, Union Pacific R. R., Eastern Div., Kansas City, Mo. 1910, Miner, Chino Copper Co., Santa Rita, N. M. 1910, Mine sampler, Detroit Copper Co., Morenci, Ariz. 1912, Miner, Cobalt Lake Mg. Co., Cobalt, Ont.; Assayer, Campbell & Deyell's, Cobalt, Ont. 1913-14, Asst. in Geol. Dept., Mo. School of Mines, Rolla, Mo.

Present position: Instructor in Geology and Mineralogy, Missouri School of Mines, Rolla, Mo.

Samuel Coleman Redd, Morenci, Ariz.

Proposed by M. H. McLean, Frank W. McLean, Frank D. Rathbun.

Born 1880, Verdon, Va. 1898-1910, Three years at Randolph Macon College, Ashland, Va. 1905-06, Galveston, Harrisburg & San Antonio Ry., El Paso, Texas. 1906-07, Rio Grande, Sierra Madre & Pacific Ry., Temosachic, Chih., Mexico. 1909, Cananea Cons. Copper Co., Cananea, Son. 1910-12, Mexico Northwestern Ry., Chih., Mexico. 1912, Madera Lumber Co., Madera, Chih., Mexico.

Present position; 1913 to date: Engineer with Detroit Copper Co.

Miles Sedivy, Hurley, N. M.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1884, Cleveland Ohio. 1900-03, South High School, Cleveland, Ohio. 1905-08, Missouri School of Mines; B. S. 1911, Foreman, Concentrating Dept. 1912, Ray Cons. Copper Co., Hayden, Ariz. 1912, Foreman, Fine Crushing Dept. 1914, Repairs, Chino Copper Co., Hurley, N. M.

Present position: Foreman, Fine Crushing Dept., Chino Copper Co.

Jacob David Shilling, Jr., Bingham Canyon, Utah.

Proposed by R. C. Gemmell, C. W. Whitley, Thomas S. Carnahan.

Born 1882, Hilltown, Pa. 1902-06, Lafayette College, Easton, Pa. 1906-09, Asst. Mine Engr., Utah Copper Co. 1910-12, Gen'l Foreman, Utah Copper Co.

Present position: 1913 to date; Asst. Supt. of Mines, Utah Copper Co.

Alvin Clark Stahl, Wilkes-Barre, Pa.

Proposed by F. M. Chase, J. M. Humphrey, Charles Enzian.

Born 1867, Wilkes-Barre, Pa. 1884, Public school, Wilkes-Barre, Pa. 1886, Business course, Wilkes-Barre Business College. 1890, Mathematics during winter at Wyoming Seminary, Kingston, Pa. 1885, One year in car shop of Lehigh & Wilkes-Barre Coal Co., Wilkes-Barre, Pa. 1887, Mining engineer corps, Lehigh Valley Coal Co., Wilkes-Barre, Pa. 1891, About one year as draftsman with James Pollock, mechanical engr. and breaker designer. 1892, Two years transitman, with Simpson & Watkins, Scranton, Pa., individual coal operator. 1894, Transitman and draftsman, L. V. C. Co., Wilkes-Barre, Pa. 1902, Division Mining Engineer, L. V. C. Co., Centralia, Pa.

Present position: 1910 to date, Asst. Mining engineer, Lehigh Valley Coal Co.

William Boyce-Thompson, New York, N. Y.

Proposed by Walter H. Aldridge, B. B. Thayer, Henry Krumb.

Born 1869, Virginia City, Mont. Phillips Exeter academy, Columbia School of Mines. Senior member, New York Stock Exchange firm of Thompson, Towle & Co. Director First Natl. Bank of Yonkers. Director, Hendee Mfg. Co. Director, Utah Copper Co. Director, Nevada Cons. Copper Co. Director and Pres., Inspiration Cons. Copper Co.

Present position: Director, Utah Copper Co., Nevada Cons. Copper Co., Pres., Inspiration Copper Co.

Thomas Norton Turner, Nikolaievsk-on-Amur, Siberia.

Proposed by J. C. Branner, C. F. Tollman, Jr., G. H. Clevenger.

Born 1888, Port Carbon, Pa. 1906, Grad. of Bloomsburg State Normal School. 1906-07, Rodman and levelman, Los Angeles Aqu. Survey (Credit in C.E. of Stanford given for this work). 1907-11, Student, Stanford Univ., summers spent in general mining work. 1912, Instructor, private, mineralogy and geology. 1912-13, Asst. to Supt. of Orsk Goldfields in East Siberia. 1914, Stanford Univ.; A.B. 1906-07, Rodman and levelman, Los Angeles Aqueduct Survey. 1910, Lead mines, Death Valley, Cal. 1911-12, Asst. in metallurgy, Stanford Univ., private instructor in mineralogy and geology.

Present position: 1912 to date, Asst. to Supt. of Orsk Goldfields in East Siberia.

Charles M. Warner, New York, N. Y.

Proposed by H. J. Stehli, A. S. Dwight, F. F. Colcord.

Born 1889, Memphis, Tenn. 1904-08, Rutgers Scientific School; B.S. 1910-12, Columbia University. 1908-10, Mammoth Copper Co., plant of A.S.S.M. & R. Co., one year and a half underground and half year in various capacities. 1912-13, Midvale plant of A.S.S.M. & R. Co. in charge of construction of roasting equipment and then roaster foreman.

Present position: 1913 to date; Field Engr. for the Dwight & Lloyd Sintering Co.

Arthur J. Wehr, Morenci, Ariz.

Proposed by Frank W. McLean, M. H. McLean, Frank D. Rathbun.

Born 1890, St. Paul, Minn. 1898-1901, Assumption School, private. 1901-07, College of St. Thomas, St. Paul. 1907-11, Univ. of Minn.; E. M. 1911, Inspector of Roads and Bridges, Ramsey Co., Minn. 1912, Asst. to J. C. Carnow, Mine Supt., Ariz. Copper Co., Morenci, Ariz. 1912, Mill operative and experimental sampler, A.C. mill. 1912-13, Feed Floor Foreman, D. C. M. Co. smelter, Morenci, Ariz.

Present position: 1913 to date, Asst. Engineer and Draftsman, D. C. M. Co., Rocky Gulch Division.

Joseph L. White, Mayer, Ariz.

Proposed by G. M. Colvocoresses, R. T. Walker, D. J. Evans.

Born 1883, Albany, N. Y. 1892-1900, General school, Denver, Colo. 1900-01, High school, Denver, Colo. 1903-07, Colo. School of Mines; E.M.; M.E. 1907-09, El Tiro Copper Co., El Tiro, Ariz. 1909-11, Edua Mines Co., Edua, Idaho. 1911, Compania Minera Cusi, Cusihiuriachic, Chih., Mexico. 1912, Atlas Min. & Mill. Co., Sneffels, Colo.

Present position: Cons. Arizona Smelting Co., Blue Bell Mine.

Arno S. Winther, Bingham, Utah.

Proposed by Fred Cowans, R. C. Gemmell, C. W. Whitley.

Born 1882, Fergus Falls, Minn. 1899, Fergus Falls High School. 1903, Univ. of Minn.; E.M. 1903-04, Silver King Min. Co., Park City, Utah. 1904-10, Cerro de Pasco Min. Co., Cerro de Pasco, Peru, S.A.

Present position: 1910 to date, Mine Supt., Utah Cons. Mining Co.

Clark Watson Wright, Kincaid, Ill.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1883, St. Louis, Mo. 1890-1900, public schools. 1900-01, St. Louis Univ. 1907-08, Private tutor, W. A. Godbey, Principal of the Pope School of St. Louis. 1908-12, Missouri School of Mines and Metallurgy, Rolla, Mo.; B.S. 1906-08. Rodman and instrument man, Terminal Railroad Assn., St. Louis, Mo. 1912-13, Asst. Engr., Madison Coal Corp., St. Louis, Mo. 1913, Mining Engr., Compania Estanifera de Llallagua, Llallagua, Bolivia, S.A. 1913, Field Engr., Allen & Garcia. Cons. Engrs., McCormick Bldg., Chicago, Ill.

Present position: 1913 to date; Division Min. Engr., Peabody Coal Co., McCormick Bldg., Chicago, Ill.

Associates

William Edward Colby, San Francisco, Cal.

Proposed by S. B. Christy, H. Foster Bain, W. S. Morley.

Born 1875, Benicia, Cal. Have specialized in the practice of mining law for fifteen years. Lectures on law of mines, Univ. of Cal. Associated with Hon. Curtis H. Lindley in the practice of law and assisted in preparation of 3d edition of "Lindley on Mines." Prepared article on law of mines for "Modern American Law."

Present position: Preparing historical chapter on American mining law for Carnegie Institution, Washington, D. C.

William Tucker Dean, Chicago, Ill.

Proposed by Rex R. Seeber, C. F. Malcolmson, Edwin C. Reeder.

Born 1876, Iowa. 1900, Armour Inst. of Tech.; B.S., E.E. 1905, Armour Inst. of Tech.; M.S., E.E. 1900-01, Gen. Electric Co., Schenectady. 1901-06, Ill. Steel Co., So. Chicago, Supt. Electric Dept.

Present position: District Mgr., Power and Mining Dept., General Electric Co., Chicago, Ill.

Charles C. Goodrich, New York, N. Y.

Proposed by C. M. Weld, H. P. Henderson, R. P. Bowler.

Born 1871, Akron, Ohio. 1893, A. B. Howard.

Present position: Pres., Pozo Gilpin Co., Pres., Millcreek Gas Co.

Clyde A. Heller, Philadelphia, Pa.

Proposed by F. Lynwood Garrison, Eli T. Conner, H. M. Chance.

Born 1870, Pennsylvania.

Present position: Pres., Tonopah Belmont Development Co., Philadelphia, Pa.

Junior Members

Elwin B. Hall, Stanford Univ., Cal

Proposed by D. M. Folsom, J. C. Branner, H. W. Young.

Born 1892, Ventura. 1910-14, Geology, Stanford Univ.

Present position: Student, Geology Dept.

Wallace Hansen Hayden, Golden, Colo.

Proposed by F. W. Traphagen, William R. Chedsey, Harry P. Wolf.

Born 1886, New York City. 1914, Grad., Colo. School of Mines. 1910-11,

Worked underground, Homestake Mine. Worked, Magna Plant, Garfield, Utah. 1909, Bowdoin: A.B.

Present position: Student.

William Coryell Hogoboom, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1889, Wahoo, Neb. 1906, Grad. Los Angeles High School. 1909-12, Univ. of Southern Cal. 1912-14, Member of present senior class, Missouri School of Mines.

Present position: Student.

Holman Thompson Marshall, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, V. H. McNutt.

Born 1891, Sioux City, Ia. 1905-09, Grad., Sioux City High School. 1909-14, Mining Engineering course. 1913, Steam shovel repair work with Utah Copper Co., Bingham, Utah.

Present position: Student.

Herbert Nelson Witt, Cambridge, Mass.

Proposed by Raymond B. Ladoo, Charles H. White, L. C. Graton.

Born 1889, Boston, Mass. 1908-10, Harvard College. 1910-13, Aid, U. S. Coast and Geodetic Survey in the Philippine Ids. (1910-12) and Alaska (1913). 1913, Geologist with International Cons. Oil Co., at Spring Valley, Wyo. 1913-14, Harvard College.

Present position: Student.

Change of Status—Associate to Member

Emory M. Marshall, Yerington, Nev.

Since joining the Institute in 1912, I have graduated from the Massachusetts Inst. of Tech., M.S. July 5, 1912, Nevada Douglas Copper Co., Ludwig, Nev., first as timberman and later as asst. engr. August, 1912, McConnell Mines Co., Mason, Nev., as engr. and assayer and bookkeeper.

Present position: 1913 to date, Empire Nevada property, Yerington, Nev., as assayer and engineer.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Apr. 10 to May 10, 1914. This list, together with the list published in *Bulletin* Nos. 88 and 89, April and May, 1914, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1914, and brings it up to the date of May 10, 1914.

ABRAHAMS, A. I.	1391 Madison Ave., New York, N. Y.
ADAMS, HUNTINGTON	115 East 80th St., New York, N. Y.
ADDICKS, LAWRENCE	Douglas, Ariz.
ANDERSON, W. G.	Northbrook, Frontenac Co., Ont., Canada.
ALLEN, ROY H.	Jamesburg, N. J.
BADLAM, STEPHEN	134 Beaver Street, Beaver, Pa.
BAKER, HUGH C.	316 Blackburn Ave., Ottawa, Ont., Canada.
BARBOUR, THOMAS J.	The Pacific-Union Club, San Francisco, Cal.
BATCHELLOR, STILLMAN	Real del Monto, Hgo., Mexico.
BECK, EDWIN L.	1555 Alice St., Oakland, Cal.
BELLINGER, H. C.	Spokane, Wash.
BERG, P. T.	63 Strandvagen, Stockholm, Sweden.
BODDINGTON, HENRY D.	Ruatan, Bay Islands, Spanish Honduras, C. A.
BRANTLY, J. E.	Geological Dept., Atlanta, Ga.
BRAYTON, C. C.	433 California St., San Francisco, Cal.
BUDROW, L. R.	Esqueda, Son., Mexico, via Douglas, Ariz.
BURDICK, CHARLES A.	Socorro, N. M.
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The deaths of the following members were reported to the Secretary's office during the period Apr. 10 to May 10, 1914:

Date of Election.	Name	Date of Decease.
1886	*Bailey, Charles.	April 20, 1914.
1893	*Crawford, J. J.	April 8, 1914.
1911	*Dull, A. J.	July 9, 1913.
1893	*Frasch, Herman.	April 3, 1914.
1903	*Maiz, Joaquin.	
1912	**Tatton, Edwin H.	
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

The Placer Law as Applied to Petroleum

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(New York Meeting, February, 1914)

AN intelligent discussion of the oil situation and its needs, whether from the standpoint of the prospector, the operator, the engineer, or the public administrative officer, must be founded upon a knowledge of the law under which oil lands are obtained. A consideration of new legislation must be prefaced by an understanding of the present law and its results. This paper undertakes to present no new remedy for the ills of the present situation, no new solution of the problems the oil business is facing. It attempts merely to summarize the present law, its provisions, its origin, its development, and its effects. The discussion does not pretend to be exhaustive from a legal standpoint. It aims to avoid legal intricacies which tend to confuse rather than clarify. It seeks but to present the essentials of the existing situation.

The title to government lands known to contain valuable deposits of petroleum may be acquired only under the terms of the placer law as embodied in Section 2329 *et seq.* of the Revised Statutes. As a preliminary to considering the desirability of this situation it is profitable to examine into its origin.

ORIGIN OF THE LAW

Congress has left no room for doubt as to the legal applicability of the placer law to oil lands. The Act of Feb. 11, 1897 (29 Stat., 526), provides:

"that any person authorized to enter lands under the mining laws of the United States may enter and obtain patent to land containing petroleum or other mineral oils, and chiefly valuable therefor, under the provisions of the laws relating to placer mineral claims."

The immediate cause for the passage of this Act was stated in a brief report in the House of Representatives (there was no debate in the Senate) to be a decision of the Department rendered Aug. 27, 1896 (Union Oil Co., 23 L. D., 223), directing the cancellation of an oil placer entry on the ground that land containing petroleum cannot be located and entered as a placer mine. The report of the Public Lands Committee of the House contains the following interesting statements (*Congressional Record*, vol. xxix, p. 1409):

* Chairman Oil Board, U. S. Geological Survey. Non-member.

"The Committee on the Public Lands, having had under consideration House bill 9983, report the same back with the recommendation that it do pass.

"Public lands containing petroleum and other mineral oils have been held and patented under the placer mining acts of the United States for many years past. . .

"Under these rulings and in view of the uniform practice of the Land Department for many years, considerable quantities of lands have been filed upon, held, and improved for the purpose of developing oils in various of the public-land States, and in many instances large sums of money have been expended in such development.

"Under a decision of the Secretary of the Interior, dated August 27, 1896, in the case of mineral entry No. 140, Los Angeles district, California, the Secretary held that lands containing oils could not be patented under the placer-mining acts, for the reason that the acts did not originally contemplate the extension of their provisions to lands of this character. It has never been held by any authority that the provisions of the placer acts were not adequate to meet the conditions surrounding the development of mineral oils. The effect of the above-cited decision of August 27, 1896, is to preclude the possibility of obtaining title to lands containing oils, and as there are large areas of such lands in various of the public-land States, and as upon these lands in many instances large expenditures have already been made, and as it is manifestly for the public good that there should be some provision whereby these lands may be held and patented, the committee believe this bill should pass."

Secretary Hoke Smith, in the Union Oil Co. decision of Aug. 27, 1896, to which the committee refers, is by no means so convincing as is the report just quoted as to the previous practice of the Department. He states, with regard to the extension of the placer Act to cover petroleum (23 L. D., 228):

"But the subject has been, indirectly at least, before the Department several times, and while it has never been definitely decided, so far as I can ascertain, yet there seems to have grown up the idea that the rule prevails. An examination of the cases, however, will demonstrate the fact, I think, that there is no precedent for such belief."

After considering the various cases theretofore decided bearing more or less directly upon the question, he concludes:

"I am unable to agree that it falls within the contemplation of the mineral laws, and that it may be located and entered as a placer mine."

A reading of the cases cited by Secretary Smith shows that the specific question of whether or not petroleum-bearing lands are properly enterable under the placer Act was in no case definitely and squarely decided, although some of the cases, and particularly that of the Piru Oil Co. (16 L. D., 117), imply that such entry might with propriety be made. A letter by Secretary Bliss to the Commissioner of the Land Office under date of Nov. 6, 1897, reviewing the Union Coal Co. case (25 L. D., 351), explains this dearth of specific decisions on the point as follows (p. 354):

"From an examination of the records of your office, which I have caused to be made, it is ascertained that ever since the circular of July 15, 1873, until the date of the decision complained of, the practice of allowing entry and patent for lands chiefly valuable for their deposits of petroleum, under the law and regulations relating to placer claims, has been continuous and uniform. Under this practice a large number of patents have

been issued, and very large and valuable property interests have been acquired. Until the decision in this case, the correctness of the practice does not appear to have been questioned, and for that reason no case distinctly presenting such a question ever reached this Department."

It is interesting to note that as early as Mar. 17, 1865, the Commissioner of the General Land Office wrote, in response to an inquiry by the local land officers at Humboldt, Cal.:

"It is not the policy of the Government to deal with petroleum tracts as ordinary public lands, any more than with auriferous or other mineral or saline lands. Hence, you will report the exact description of any and all tracts strictly of the character you mention, and will withhold the same from disposal by the Government, unless otherwise specially instructed."

The history of the matter in reality begins with the discovery of gold in California in 1848. Prior to that time, and for many years thereafter, there was no law providing for the acquisition of title to mineral lands on the public domain, with the exception of special Acts providing for the sale of lands in the middle West valuable for ores of the baser metals. The development in California and adjacent parts of Nevada of a mining industry involving a large and heterogeneous population necessitated some code of rules to govern and guide the vast and intense mining activities. The Federal government failing to provide such a code, and there being no State or Territorial government, the miners of the various districts formulated and enforced, with greater or less definiteness and rigidity, local rules founded on the customs which had grown up by common consent in the interest of justice and fair play. These local regulations obviously could not provide for acquiring title from the United States, but they did prescribe the size of the claim, the method of locating and marking it, and the amount of assessment work necessary to hold it. The common underlying theory of these local rules in all the districts was that discovery of mineral is the one fundamental prerequisite to the establishment of a right or title. After the State of California was admitted to the Union both the legislature and Congress recognized the local codes as governing when not in conflict with the statutes or laws of the State. Recognition of the district rules was also accorded by the Federal courts and by the Land Department (Lindley on Mines, pp. 72 and 73, cases cited in footnotes).

It was not until July 26, 1866, that a Federal law was passed governing the mining industry of the public domain. This Act (14 Stat., 251) recognized and continued the local codes so far as not in conflict with the laws of the United States, and provided for the granting of title in fee to veins or lodes of quartz, or other rock in place, bearing gold, silver, cinnabar, or copper. Strangely enough, the Act of 1866 made no provision for the acquisition of title to placer claims, and to remedy this

omission Congress passed the so-called placer Act of July 9, 1870 (16 Stat., 217), amending the Act of 1866 by providing:

"That claims usually called 'placers' including all forms of deposit excepting veins of quartz, or other rock in place, shall be subject to entry and patent under this act, under like circumstances and conditions, and upon similar proceedings as are provided for vein or lode claims; . . . "

The lode law of 1866 was replaced by the general mining Act of May 10, 1872 (17 Stat., 91), which modified the procedure necessary to obtain patent to lode claims, emphasized the necessity of discovery as a prerequisite to location, and provided that the Act of July 9, 1870, should remain in force except as modified by the changes made in the lode law. Section 10 of the Act is in part as follows:

"That the act entitled 'An act to amend an act granting the right of way to ditch and canal owners over the public lands, and for other purposes,' approved July ninth, eighteen hundred and seventy, shall be and remain in full force, except as to the proceedings to obtain a patent, which shall be similar to the proceedings prescribed by Sections 6 and 7 of this act for obtaining patents to vein or lode claims."

! The Acts of 1872 and of 1870 were incorporated into the Revised Statutes and are the mining law of the present day. Thus Section 2329 provides:

"Claims usually called 'placers,' including all forms of deposit, excepting veins of quartz, or other rock in place, shall be subject to entry and patent, under like circumstances and conditions, and upon similar proceedings, as are provided for vein or lode claims; but where the lands have been previously surveyed by the United States, the entry in its exterior limits shall conform to the legal subdivisions of the public land."

The language of this section and of the Act of 1870, from which it was taken, is sufficiently comprehensive to cover all the minerals known, or ever to become known, to man. There is, however, nothing in the debates in Congress to indicate that the sweeping words "all forms of deposit" were adopted with the intention of including minerals not then important but later to become so. The debates in both houses of Congress indicate that the placer Act was passed for the purpose of granting title to the gold-placer claims of the far West, and that the all-inclusive wording was adopted as the easiest way to distinguish these claims from so-called quartz claims. It was the result not of a far-sighted attempt to include other minerals which might later become of importance, but of a failure to foresee or to consider the possibility that such minerals might be discovered and be included within the all-embracing language. The speeches by Senator Stewart, of Nevada, who piloted the bill through the Senate, and by Mr. Sargent, of California, who introduced it in the House, embody the only reasons advanced on the floor of Congress for the passage of the Act, Senator Stewart thus stating its object (*Congressional Globe*, 41st Congress, 2d session, p. 3054, vol. xci):

"The act (of 1866) proposed to extend the principle of the preemption laws to quartz veins. I wish to say to the Senate that the act has been in operation now four years, and so far as quartz veins are concerned it has proved very beneficial. The purpose of this bill is to extend it to placer mines. There was a difficulty at that time in extending it to placer mines, so as to give homes to those who have worked at the diggings of the foothills of California and elsewhere where there were placer mines. They got no title, and they cannot prosper for that reason. They have got little orchards and little homes, and we want them to get title to their property. They have a placer mine where they can work a little in the winter, perhaps get a few dollars to keep along, and then they have a little orchard and they want one hundred and sixty acres of this land. Now, for the purpose of allowing them to get these homes the bill extends the principle of preemption to these worn-out placer diggings. That is the object of the bill."

The remarks of Mr. Sargent in the House are much more extensive than those of Senator Stewart in the Senate, and dwell at length upon the situation in the gold-placer districts in California, and to a less extent in Colorado. His discussion is confined to the gold placers, without reference to the possibility of other minerals, and his constant references to the manner and difficulty of finding placer gold and to the desirability of granting title to gold deposits only emphasize the completeness with which his mind was filled with the thought of his mineral, to the exclusion of all others. It is interesting to note that Mr. Sargent's principal argument is that the granting of title to placer claims would attach the migratory miner to the land, and would give a permanence similar to that of agricultural communities to the mining camps. His attitude is only feebly indicated by the following quotations from his speech of Mar. 18, 1870 (*Congressional Globe*, 41st Congress, 2d session, p. 2028, vol xc):

"In July, 1866, Congress passed a law by which the holders of quartz mining claims could purchase their claims. It extended the principle of the preemption laws and recognized the possession of miners so far as quartz mines were concerned. . . . The proposition of this bill is to extend the principles of that act. . . . We propose by this bill to allow the placer miner who owns a limited quantity of land under mining customs to come before the land office and make proof of his possession, and having so done to the satisfaction of the Government, and shown that there is no adverse claim whatever, that then he shall pay double the minimum price for his claim and have a patent for it. . . .

" . . . The effect of it will be that our mining population will be attached to the soil, that they will be changed from the nomadic, wandering character which they now have to settled communities."

Although Congress so evidently had in mind only gold placers, the Land Department seemingly had no hesitancy in applying the apparently comprehensive language to other minerals, among them being oil. On July 15, 1873, Commissioner Drummond of the General Land Office issued a circular in which he stated (*Copp's Mineral Lands*, p. 61):

"That whatever is recognized as a mineral by the standard authorities on the sub-

ject . . . should be treated by this office as coming within the purview of the mining act of May 10, 1872," and

"That lands valuable on account of borax, carbonated soda, nitrate of soda, sulphur, alum, and asphalt, as well as 'all valuable mineral deposits,' may be applied for and patented under the provisions of the mining act of May 10, 1872."

On Jan. 30, 1875, Commissioner Burdett, in response to a specific inquiry as to whether or not land yielding petroleum might be patented under the mining Acts, wrote (1 Copp's *Land Owner*, 179; Sickles *Mining Laws*, 491):

"Petroleum claims may be entered and patented under the mining act of May 10, 1872, upon full compliance with the provisions and requirements of said act."

According to the report of the House Public Lands Committee already quoted (*Congressional Record*, vol. xxix, p. 1409), the first patent to an oil claim was issued Mar. 22, 1880, for mineral entry No. 18, Los Angeles district, California. Further inquiries appear, however, to have been raised, for under date of Mar. 31, 1882, Commissioner McFarland expressed himself rather emphatically (9 Copp's *Land Owner*, 51):

"Lands containing deposits of petroleum have been entered as placers and patented as such. Your inquiries are fully answered, therefore, by stating that lands of that character are subject to entry and disposal according to the law and regulations relating to placer claims."

It will be noted that these rulings were all by the Land Office and were, therefore, subject to review by the Secretary of the Interior. Although the Department must have acquiesced in the matter, there appears to be, as already pointed out, no case in which the Secretary passed squarely upon the question until the adverse decision of Aug. 27, 1896, which resulted in the Act of Feb. 11, 1897, definitely extending the placer law to oil lands.

PROVISIONS OF THE LAW

Having thus examined into the origin of the placer law and its application to petroleum, the reasons for its various provisions become apparent.

The most important requirement is that a discovery of mineral must be made before possessory rights attach. This principle, which, as we have already seen, was fundamental in the miners' codes, was adopted into those codes from the Spanish and Mexican law. Section 2329 of the Revised Statutes provides for placer claims "under like circumstances and conditions and upon similar proceedings" to those for lode claims. Section 2320 provides that "no location of a mining claim shall be made until the discovery of a vein or lode within the limits of the claim located."

Lindley summarizes the law on the subject as follows (Lindley on *Mines*, pp. 779 and 781):

"Discovery is just as essential in case of placers as it is in lode locations. The supreme court of California at one time expressed the view that neither the federal laws nor the local rules and customs of miners required that a discovery should be made as a prerequisite to a placer location, but this is obviously a mere dictum; it is also opposed to the current of judicial authority. The land department has uniformly held that discovery is essential in the case of placers, going so far at one time as to hold that such discovery was essential in each twenty-acre tract within a location of one hundred and sixty acres located by an association of persons.

"In the case of petroleum deposits the courts in California have in recent years been confronted with some serious problems upon the subject of what constitutes a sufficient discovery which will sanction a location of a claim to oil lands under the laws applicable to placers. It is well known that the natural habitat of this class of mineral hydrocarbons is in stratified rocks some distance below the surface, and except for the occasional appearance at the surface in the form of oil seepages, springs, or other indications of the subterranean existence of petroleum, there is nothing to guide the miner in making his location. It requires more or less extensive development in the nature of well boring and prospecting to determine the nature, extent, and permanency of the deposit.

"Of course, exploitation on adjacent lands might raise a strong presumption that a given tract contained petroleum. An oil-producing well within each of four sections of land surrounding a fifth would produce a conviction that the oil deposit was underneath the fifth section. This fact might justify the land department in classifying the section in the category of mineral lands, or the government surveyor in returning it as such, but it would not dispense with the necessity of making a discovery."

The only essential to a valid location in addition to a discovery of mineral is, so far as the Federal law is concerned, that the claim be "distinctly marked on the ground so that its boundaries can be readily traced" (R. S. 2324). In many instances State laws and local regulations prescribe other requirements, such as posting a location notice on the claim and the recording of the location with a specified district or county officer. Once these requirements have been met the locator has "the exclusive right of possession and enjoyment of all the surface included within the lines" of his claim (R. S. 2322). The claim may comprise 20 acres if located by an individual or, if made by an association, 20 acres for each member thereof (R. S. 2321), but in no case may it exceed 160 acres (R. S. 2320). There is no limit upon the number of claims which may be located by a single individual or association.

Having complied with all the requirements necessary to a valid location and having thus become entitled to exclusive right of possession, the claimant, in order to continue the right, must expend not less than \$100 worth of labor or improvements upon the claim during each year (R. S. 2324). The requirement of development work as a condition of continued possession of mining property was, like the discovery requirement, incorporated into the local regulations from the mining laws of Spain and Mexico. Failure to perform the required assessment work forfeits the right to exclusive possession and the claim becomes "open to relocation in the same manner as if no location . . . had ever been

made" (R. S. 2324). The present law gives the claimant until the end of the calendar year succeeding his location in which to perform his first assessment work (21 Stat., 61), and thereafter the calendar year is the period for which such work is required.

If the claimant has made a valid discovery and has expended not less than \$500 worth of labor or improvements upon the claim (R. S. 2325), he is, upon compliance with certain requirements as to posting notices, publication, survey (R. S. 2325), and upon payment of \$2.50 an acre (R. S. 2333), entitled to a patent conveying a title in fee to the area covered by his claim. The right to patent may also be earned by holding and working a placer claim for the period prescribed by the statute of limitations of the State in which the claim is situated (R. S. 2332), but this right is rather infrequently exercised, is not of especial importance to the present discussion, and will not hereinafter be considered.

It is evident that there are three stages of development of the title which a claimant under the placer law acquires to the land included within his claim: First, the period between his physical occupation of the claim and his making location; second, the period between his location and patent; and third, the period after patent.

The third case requires no discussion. The patent is a grant of title in fee simple, good against the grantor and all adverse claimants, and subject to attack only upon the ground of fraud.

The character of title during the period between location and patent, while by no means clearly or consistently defined by the courts, is nevertheless such as to protect the claimant in the exclusive possession and enjoyment of his claim so long as he performs his annual assessment work and complies with State and local regulations. Lindley in reviewing the character of this estate says (Lindley on Mines, p. 892):

"Prior to the issuance of a patent the locator cannot be said to own the fee-simple title. The fee resides in the general government, whose tribunals, specially charged with the ultimate conveyance of the title, must pass upon the qualifications of the locator and his compliance with the law. Yet, as between the locator and every one else save the paramount proprietor, the estate acquired by a perfected mining location possesses all the attributes of a title in fee, and so long as the requirements of the law with reference to continued development are satisfied, the character of the tenure remains that of a fee. As between the locator and the government, the former is the owner of the beneficial estate, and the latter holds the fee in trust, to be conveyed to such beneficial owner upon his application in that behalf and in compliance with the terms prescribed by the paramount proprietor."

It should be borne clearly in mind and cannot be too strongly emphasized that this character of right and title is present *only after location* and that *no location can be made unless founded upon discovery*. Unless he has made discovery no amount of development work and no compliance with other Federal, State, or local requirements (except that

of possession and work for the period prescribed by the statute of limitations; R. S. 2332) will confer upon the claimant an equitable title as against the government or a tenure in the nature of a fee against others. As against the government, the mineral claimant who has made no discovery has in fact no title whatever, either legal or equitable. Neither has he any vestige of title against strangers so far as the Federal laws are concerned. He is not before the Department until he has made discovery of mineral on which to found a location. Consequently the Department cannot, even if it would, protect him against intruders upon his occupancy or violators of his possession. He has, however, been granted some measure of protection, none too clearly defined, by the courts, and the measure of this protection is a matter of great concern to the prospector for deep-seated deposits. How far will the courts safeguard his occupancy and possession prior to discovery against, first, agricultural claimants, and second, other mineral claimants?

The leading case upon the rights of a mineral occupant against an agricultural claimant is *Cosmos Exploration Co. vs. Grey Eagle Oil Co.* (112 Fed., 4). In this case, which was affirmed by the Supreme Court of the United States, although this point was not specifically mentioned (190 U. S., 310), it is stated that the defendants had made "pretended placer mining locations" upon the land in controversy but had made no discovery, that thereafter complainant selected this land in lieu of certain lands in the national forest under the Act of June 4, 1897 (36 Stat., 11, 36), which provides for the selection of "vacant land open to settlement," and that subsequently defendants discovered oil in paying quantities. The Circuit Court of Appeals of the ninth circuit held (syllabus):

"Land was not 'vacant and open to settlement,' and subject to selection under such act, where at the time of the application it was in the actual occupancy of others engaged in exploring it for oil, under oil placer mining locations previously made by them, although such locations did not appear by the records of the local land office, and although they were not valid as against the United States, because there had been no previous discovery of oil on the land, where the locators prosecuted the work of exploration with due diligence, and with the result of discovering oil in paying quantities before the selection by the applicant under the forest reserve act had been approved by the land department. Until by such approval an applicant is vested with the equitable title to the land, it remains subject to exploration for minerals under the mining laws; and, while lawfully occupied by one engaged in making such exploration, it is not 'vacant' within the meaning of the act; nor is it open to settlement where, as the result of such exploration, its mineral character is established, while the title, both legal and equitable, remains in the United States."

It should, however, be noted that while under this decision the prospector proceeding in good faith to explore for minerals is protected against adverse agricultural claimants, there is nothing of record in the Land Office to show that the land which he occupies is not "vacant and

open to settlement," and such agricultural filings are likely to be made at any time, subjecting him to the expense of maintaining contest proceedings before the Land Department or the courts. It may also be worthy of note that some importance seems to be attached to the fact that oil was discovered before the agricultural selection had been approved.

That a measure of protection is afforded against adverse mineral claimants who attempt by forcible, fraudulent, or surreptitious means to violate the possession of a mineral occupant proceeding in good faith, is held by *Miller vs. Chrisman* (140 Cal., 440), in which Mr. Justice Henshaw says:

"It is to be remembered that it is not essential to the validity of a location that the discovery shall have proceeded or shall coexist with the posting of the notice and the demarcation of boundaries. The discovery may be made subsequently, and when made operates to perfect the location against all the world, saving those whose bona fide rights have intervened. One who thus in good faith makes his location, remains in possession, and with due diligence prosecutes his work toward a discovery, is fully protected against all forms of forcible, fraudulent, surreptitious, or clandestine entries and intrusions upon his possession. Such entry must always be peaceable, open and above board, and made in good faith, or no right can be founded upon it. (*Belk vs. Meagher*, 104 U. S., 279; *Atherton vs. Fowler*, 96 U. S., 513; *Nevada Sierra Oil Co. vs. Home Oil Co.*, 98 Fed., 673.)"

And further:

"It further appears that certain valuable rights become the property of such locators even before discovery. They have the right of possession against all intruders (*Garthe vs. Hart*, 73 Cal., 541), and they may defend this possession in the courts. (*Richardson vs. McNulty*, 24 Cal., 339.) They have then this right of possession and with it the right to protect their possession against all illegal intrusions, and to work the land for the valuable minerals it is thought to contain. We cannot perceive why these rights may not in good faith be made the subject of conveyance by the associates as well before as after discovery."

Although it would thus appear that a mineral claimant prior to discovery may not be ousted from his possession by force or fraud, it has been held repeatedly that his possession is not good against that of an adverse claimant who enters upon the land in good faith and in compliance with the law and proceeds to make a discovery. The quotations from *Miller vs. Chrisman* just given confirm this by implication and cite among others the case of *Garthe vs. Hart* (73 Cal., 541). In this case the court below has instructed the jury as follows:

"As I said, there is still another way by which a miner in this state may acquire a right to the possession of a piece of mining ground. It is by taking possession of it and clearly defining the boundaries so that they may be readily traced, and holding such possession, keeping such possession."

The appellate court reversed the lower court on this and other grounds, saying *inter alia*:

"In the hurry of the trial, the learned judge evidently overlooked the distinction between the right of a party in possession as against mere intruders, and his right as against one who has complied with the mining laws. Possession is good against mere intruders (*Attwood v. Fricot*, 17 Cal. 37; S.C., 76 Am. Sec. 567; *English v. Johnson*, 17 Cal. 115; S.C., 76 Am. Dec. 574; *Hess v. Winder*, 30 Cal. 355; *Golden Fleece Co. v. Cable Con. Co.*, 12 Nev. 321, 322); but it is not good as against one who has complied with the mining laws. (*Du Prat v. James*, 65 Cal. 556, 557.)"

In *Miller vs. Chrisman*, *supra*, the Supreme Court also cited the leading case of *Belk vs. Meagher* (104 U. S., 279). In this case, which is not as clear or simple as might be wished, the facts were stated to be as follows (syllabus):

"A. entered Dec. 19, 1876, upon a claim not then in the actual possession of any one, but covered by a valid and subsisting location which did not expire until the first day of January thereafter. Between the date of his entry and Feb. 21, 1877, he made no improvements or enclosure, and did a very small amount of work, but had no other title than such as arose from his attempted location of the claim and his occasional labor upon it. On the last-mentioned date B. entered upon the property peaceably and in good faith, and did all that was required to protect his right to the exclusive possession thereof. A. brought ejectment October 25, 1877. Held, that A's entry and labor did not entitle him to a patent under sect. 2332, Rev. Stat., nor prevent B's acquisition of title to the claim. . . ."

The court through Mr. Chief Justice Waite said:

"No one contends that the defendants affected their entry and secured their relocation by force. They knew what Belk had done and what he was doing. He had no right to the possession, and was only on the land at intervals. There was no enclosure, and he had made no improvements. He apparently exercised no other acts of ownership, after January 1, than every explorer of the mineral land of the United States does when he goes on them and uses his pick to search for and examine lodes and veins. As his attempted relocation was invalid, his rights were no more than those of a simple explorer. In two months he had done, as he himself says, 'no hard work on the claim,' and he 'probably put two days' work on the ground.' This was the extent of his possession. He was not an original discoverer, but he sought to avail himself of what others had found. Relying on what he had done in December, he did not do what was necessary to affect a valid relocation after January 1. His possession might have been such as would have enabled him to bring an action of trespass against one who entered without any color of right, but it was not enough, as we think, to prevent an entry peaceably and in good faith for the purpose of securing a right under the act of Congress to the exclusive possession and enjoyment of the property. The defendants having got into possession and perfected a relocation, have secured the better right."

The court also said:

"His ultimate right to a patent depended entirely on his keeping himself in and all others out, and if he was not actually in, he was in law out. A peaceable adverse entry, coupled with the right to hold the possession which was thereby acquired, operated as an ouster. . . . He had made no such location as prevented the lands from being in law vacant. Others had the right to enter for the purpose of taking them up, if it could be done peaceably and without force. There is nothing in *Atherton v. Fowler* (96 U. S., 513) to the contrary of this. . . ."

This doctrine has been followed in many cases (Lindley on Mines, p. 375, cited in footnote No. 2). In *Thallman vs. Thomas* (111 Fed., 277), the Circuit Court of Appeals for the eighth circuit said (Lindley on Mines, p. 376):

"Every competent locator has the right to initiate a lawful claim to unappropriated public land by a peaceable adverse entry upon it while it is in the possession of those who have no superior right to acquire the title or to hold the possession. . . . Any other rule would make the wrongful occupation of the public land by a trespasser superior in right to a lawful entry of it under the acts of Congress by a competent locator."

In *Crossman vs. Pendery* (8 Fed., 693), it was held (Lindley on Mines, pp. 370 and 371):

"A prospector on the public mineral domain may protect himself in the possession of his *pedis possessionis* while he is searching for mineral. His possession so held is good as a possessory title against all the world, except the government of the United States. But if he stands by and allows others to enter upon his claim and first discover mineral in rock in place, the law gives such first discoverer a title to the mineral so first discovered, against which the mere possession of the surface cannot prevail."

Lindley summarizes the law of the matter as follows (Lindley on Mines, pp. 378 and 379):

"(1) Actual possession of a tract of public mineral land is valid as against a mere intruder, or one having no higher or better right than the prior occupant;

"(2) No mining right or title can be initiated by a violent or forcible invasion of another's actual occupancy;

"(3) If a party goes upon the mineral land of the United States and either establishes a settlement or works thereon without complying with the requirements of the mining laws, and relies exclusively upon his possession or work, a second party who locates peaceably a mining claim covering any portion of the same ground, and in all respects complies with the requirements of the mining laws, is entitled to the possession of such mineral ground to the extent of his location as against the prior occupant, who is, from the time said second party has perfected his location and complied with the law, a trespasser."

The principal question which this leaves unsettled is, what means may a prior occupant take and to what extent may he go in preventing the peaceable and open entry which will, if it ripen into a valid location, dispossess him. The answer to this question does not appear to be found in the court decisions nor does the forcible prevention of entry, in some cases by the threatened or actual use of fire-arms, seem a satisfactory answer.

APPLICATION TO PETROLEUM LANDS

What are the effects of these various provisions, statutory and judicial, upon the development of deposits of petroleum on the public domain? Is the placer law as thus interpreted best from the standpoint of the oil prospector and oil operator, and is it wisest from the stand-

point of the government? What is the net result of applying a law designed for stable, stationary minerals occurring in easily accessible deposits to a fluid mineral occurring at great depth below the surface and requiring long-continued and expensive development work to discover? In discussing these questions, frequent reference will be made to the hearings held May 13 and 17, 1910, before the Committee on the Public Lands of the House of Representatives, upon H. R. 24070, "A bill to authorize the President of the United States to make withdrawals of public lands in certain cases." The record of these hearings will hereinafter be referred to as "H. R. 24070."

Inadequate Protection Prior to Discovery

The first weakness which presents itself when the placer law is applied to petroleum is the inadequate protection afforded prior to discovery. Oil in large quantities is seldom, if ever, found in nature on the surface of the ground. The natural laws under which it accumulates require a superincumbent rock covering of considerable thickness and density. With the exception of negligible quantities gathered from seepages and springs, all the oil produced in the United States comes from wells of greater or less depth, and it is only by means of such wells that discovery can be made. In *Miller vs. Chrisman* (140 Calif., 440) the court says on this subject:

"To constitute a discovery, the law requires something more than conjecture, hope or even indications. The geological formation of the country may be such as scientific research and practical experience have shown to be likely to yield oil in paying quantities. Taken with this, there may be other surface indications, such as seepage of oil. All these things combined may be sufficient to justify the expectation and hope that, upon driving a well to sufficient depth, oil may be discovered, but one and all they do not in and of themselves amount to a discovery. This view finds support in the *Nevada Sierra Oil Co. vs. Home Oil Co.*, 98 Fed. 673, where the circuit court was dealing with this precise question, in regard to this precise piece of land, under these identical circumstances. While perhaps it would be stating it too broadly to say that no case can be imagined where a surface discovery may be made of oil sufficient to fill the requirements of the statute, yet it is certainly true that no such case has ever been presented to our attention, and that in the nature of things such a case will seldom, if ever, occur."

Thomas A. O'Donnell, of Los Angeles, was introduced to the Public Lands Committee of the House of Representatives as "one of the oldest operators in the field—that is, not in point of years but in point of experience. He is one of our pioneers there, and is thoroughly familiar with all the facts." Mr. O'Donnell said (H. R. 24070, p. 5):

"The placer miner, looking for gold, could go along with a shovel and turn over a little gravel and get a color of gold, and he had then made the necessary discovery. But our petroleum in California is in many instances 4,000 ft. under the earth, and it is being developed successfully to-day from that depth.

"You gentlemen can readily see the absolute necessity for the oil miner to go upon this land and proceed to make that discovery. In many instances, gentlemen, with the difficulties that have arisen in this development it has taken years to find out whether a piece of land was really oil land or not and to make the discovery which is necessary before you can make application to the Land Department and acquire title to the land.

"In the case of nine-tenths of the government land in California on which men are operating to-day there is absolutely no record in the Land Office of their operations. We cannot get near the Land Office until we have made a discovery.

These discovery wells not only require long periods, some of them months or even years to drill, but they cost large sums of money. It has been estimated that the average cost of each successful well in the State of California within the last year or two has exceeded \$15,000. It is no uncommon thing for a well to cost over \$50,000. Thus the oil operator must expend large sums of money during the months required to make a discovery, and until such discovery is made he may be dispossessed and his expenditure lost if another operator enters upon the land (by other than forcible, fraudulent, or surreptitious means) and makes prior discovery. The following colloquy between Hon. Sylvester C. Smith, Representative from California, and A. I. Weil, of San Francisco, before the Public Lands Committee of the House illustrates the situation (H. R. 24070, pp. 7, 8):

"Mr. Smith: I think, however, you will admit that this is the state of affairs in California: If A goes on a quarter section, having monumented the corners as a mining claim, and begins to drill, and B comes onto the same quarter section and wants to begin to drill, the courts will not put B off.

"Mr. Weil: Oh, no; they will not put him off.

"Mr. Smith: He has the right to go on and begin to drill side by side with A.

"The Chairman: Yes, but what will the court say to B if B makes the discovery first?

"Mr. Weil: That has not been decided.

"Mr. Smith: My opinion is that they are bound to hold in his favor.

"The Chairman: That is my curbstone opinion.

"Mr. Smith: Because he is the man who first unites a discovery with a location.

* * * * *

"Mr. Smith: But the difficulty is this: A person makes a location. He has no security whatever in his tenure.

"The Chairman: Unless he works.

"Mr. Smith: Even if he works he has not, because B may come and sit down beside him and commence drilling. Therefore A, who makes the first location, cannot go out and enlist capital, or does not always feel like bringing his own capital into the enterprise, fearing that B may come along and beat him to the oil.

"The law ought to be such that when a man in some proper manner obtains possession of a piece of government land for a purpose he can hold it against the world until he has worked out his purpose. It is not so under this situation. A man may go in there with ever so good faith and stake out a quarter section and put on his rig and begin to drill; and yet another man or 20 men can come on the same land and sit down beside him and begin to drill, too."

That such large amounts have been and are being invested in oil prospecting with no greater security of title than the placer law affords would seem to be due solely to the large percentage which the investment returns if it is successful. Certainly the law, even under the most favorable court interpretations, affords meager protection for such sums, and permits interference with and annoyance of the legitimate prospector which amount to little short of blackmail. Mr. O'Donnell, already quoted, refers to some of the more common practices. On p. 7 of the record he says (H. R. 24070, p. 7):

"It is sometimes very hard to tell exactly where your lines are; and it has been quite a trick on the part of some unscrupulous persons to erect a derrick on their own property and slip it across the line."

And on p. 10 (H. R. 24070, p. 10):

"In many instances in the little towns on midnight of January 1, almost all of the saloon men, and the men that spend a great deal of their time in these towns, go out and locate the whole country. Then they come and ask for a bonus from the operator, from me, from the driller that tries to organize a little company among his associates and go out to drill on that land."

Since it is clear that the oil prospector is subject to the loss of his possession if an adverse claimant makes peaceable and open entry in good faith and discovers oil, it is to his interest to prevent such peaceable and open entry. The extent to which such entry is prevented and the means taken thereto seem to vary with the field. As stated by Mr. O'Donnell with regard to California operators (H. R. 24070, p. 9) "when they were in legitimate pursuit of discovery they were seldom disturbed." Nevertheless instances are not wanting where gunmen have been hired to prevent attempts to enter and occupy lands already being prospected. Neither is it uncommon to see dozens of location notices put up by different locators on the same claim, each in the hope that his location will be validated by a discovery or that the courts will accord him priority of possession. Such conditions inevitably tend to violence or litigation or both.

In some of the fields, particularly in the Rocky Mountain States, almost any kind of pretended development work appears sufficient to hold a claim for an indefinite period. Here many subterfuges have been adopted by those who wish to hold possession of valuable petroleum lands without the expenditure of the large sums necessary to make discovery. Rigs capable of drilling only to a few hundred feet have been erected for the purpose of doing assessment work on lands beneath which the oil sands were known to lie at depths of thousands of feet. Tunnels have been dug in hillsides to tap oil springs or seeps in the attempt to obtain inexpensively enough oil to constitute a discovery. In at least one Western State large areas have been claimed for years under assess-

ment work in the shape of roads, some of which make transportation to the fields easier, and some of which have only the merit of entailing the expenditure of the required amount of money. One large area is being held under building-stone placer claims, although much of the area is covered by dune sand and no building stone of especial value is known in the region. In some of the fields there are deposits of gypsum of fair quality upon the surface of oil lands of great prospective value. Many attempts have been made to obtain title to these lands by locating and perfecting gypsum placers. The West Side fields of the San Joaquin valley, California, are ornamented in places by Italian gardens carved in the gypsum by humorous assessment workers, and by graceful stairways which, as has aptly been said, lead nowhere but to a title to valuable oil lands.

A consideration of these various attempts, more or less successful, to hold oil lands without making or attempting to make discovery, and the instances of entry upon oil lands already being prospected for the purpose of making discovery in advance of the prior locator, would lead to the conclusion that in neither situation is the placer law satisfactory. If occupancy without discovery is respected, large areas are withheld from exploration and development, in some instances by a single individual or group of individuals. If, on the other hand, such occupancy is not respected, the oil prospector must assume undue risk of the loss of his investment prior to discovery.

Draining of Oil from Adjacent Lands

However, this is by no means the only objection, and in reality is not the greatest objection, to the placer law as applied to petroleum. Oil is a fluid which occurs saturating relatively porous beds beneath the earth's surface. Like any other liquid, it is subject to the laws of gravity, hydrostatic pressure, and capillarity, and it is in nearly every instance under greater or less gas pressure. When a well taps an oil sand and the oil immediately around the well is drawn off other oil from greater distances flows in and is also drawn off. The amount of oil under a given tract of land is thus not a constant, as would be the amount of some solid mineral. It is subject to change through the opening of wells on adjoining tracts, or through the flooding of the oil sands with water from mishandled wells. It is illogical to dispose of oil by the acre, for the very apparent reason that it may not stay under a given acre but may be drawn off by wells on neighboring acres. It is so well established in the law of real property as to need no discussion that fluid substances are the property of the person upon whose lands they are brought to the surface, regardless of whose lands they originally underlay. When oil or gas "escape and go into other land, or come under another's control, the title of the former

owner is gone. If an adjoining owner drills his own land and taps a deposit of oil or gas, extending under his neighbor's field, so that it comes into his well, it becomes his property." (Brown vs. Spilman, 155 U. S., 665, 670, citing Brown vs. Vandergrift, 80 Penn. St., 142, 147; Westmoreland Nat. Gas Co.'s Appeal, 25 Weekly Notes of Cases (Penn.), 103.)

Since oil thus belongs to the producer thereof, it becomes advantageous for the owner of a given tract to so drill his wells that they will draw the maximum quantity of oil from adjacent lands. The radius which will be drained by a given well is not definitely known and is, of course, variable. Enough data are at hand, however, to show that considerable distances may be affected. W. W. Orcutt, of Los Angeles, in charge of the geological department of the Union Oil Co., testified before the Public Lands Committee (H. R. 24070, pp. 48, 50):

"Sometimes they will reach 1,000 feet; sometimes 2,000 feet, so that it is possible to take all, or nearly all, of the oil off of a very large area of land from one well. Usually the man who has the first well down in an area will get the bulk of the oil, because after the first well is drilled the gas pressure is loosened up, and it is all coming in that direction. The line of least resistance is toward the first well, and a well drilled subsequent to the first well will not get very much oil.

"I think a gentleman this morning made the statement that some wells he was interested in had produced 400 per cent. more than some wells his neighbor had drilled just across the line subsequent to the time he drilled his wells. That is a well-known fact. The man who gets the first well down in an area will get the bulk of the oil.

"I have an instance in mind in the Santa Maria field, where practically all of the oil has been taken off of 80 acres from one well. That was a very strong well. It had a big gas pressure. When other wells were drilled on that 80-acre piece they got only little 'strippers' that did not amount to much."

Such information as has been obtained since the date of Mr. Orcutt's statement (1910) indicates that the instances are rare where a well drains a territory so large as that given in his statement. But the following statement by Mr. O'Donnell is significant (H. R. 24070, p. 14):

"The reason for putting them close together in the richer parts of the field is principally to get them along the line. If you will note, you will see that the closest driving is usually along the line, in order to get as much of the other fellow's oil as you can. In many instances we have drilling arrangements, spacing them, and making blue prints of each piece where we have an agreement, showing just how far a fellow shall put his well from his other well. That shows that the other fellow believes that if your wells are thicker on your particular piece of land they will affect his lands in time, even although they are half a mile away. After the discoveries are made, of course the industry is very profitable; and the development will proceed in the nature of a race, perhaps, along the line."

At any rate, the distances affected are known to be such that the practice of drilling as close as may be to the property lines is almost universal. The owner of tract A attempts to draw off the oil from his neighbor's

tract B and to forestall the neighbor in his attempt to drain the oil from tract A. Thus it becomes a race for the oil along the boundaries and each operator is forced to drill by the drilling of his neighbor. Whether or not he desires to drill, even although he is unable to dispose of more oil than he is already producing, or even if for financial or other reasons it would be to his advantage to defer the bringing in of new wells, the operator must nevertheless drill and operate wells along his boundaries or his neighbors will enrich themselves at his expense. Numerous statements by oil operators might be quoted showing the universality of the practice of line drilling and the necessity for it under the placer law, but an examination of the well map of any of the high-production public-land fields will show the present situation more conclusively than any statement. It is common practice to adopt so-called gentlemen's agreements as to the spacing of wells on neighboring areas, and by this means to attempt to prevent line racing and forced production. But while these agreements are in many instances advantageous they by no means solve the whole problem. Mr. O'Donnell says (H.R. 24070, p. 117):

"The general custom in the field is to have an agreement between adjoining operators to drill not nearer than 150 ft. from the line; and after you get your agreement, the custom is to drill your well at 150 ft. just as quick as you can."

It is noteworthy that the Office of Indian Affairs, in regulating oil and gas operations on Indian lands, has recognized the evils of line drilling. Many, if not all, of the oil leases stipulate that no well shall be drilled within 200 ft. of a division line unless necessary to offset wells on adjoining tracts. The following provisions seem to be typical (Regulations of Apr. 20, 1908, revised to May 1, 1912, governing leasing of lands of the Five Civilized Tribes):

"The lessee may be required to drill and operate wells to offset paying wells on adjoining tracts and within 300 feet of the dividing line.

"Lessees shall not be allowed to drill within 200 feet of the division lines between lands covered by their leases and adjoining lands, except in cases where wells on adjoining tracts are drilled at a less distance, in which case lessees may offset such wells by drilling at an equal distance from the line."

Forced Production

A third serious objection to the placer law, growing out of the two already discussed, is the economic loss through overproduction. The operator who has not made actual discovery of oil is insecure in his title. Furthermore, his neighbors may resort to line drilling and rob him of deposits of great value. He is forced, therefore, in fields where the competition is keen, to drill for and produce oil whether he has a profitable market or not. The result of this condition is seen in California, where,

in spite of a producer's association which tends to prevent the price of oil from falling so low as to ruin the industry, the oil stock on hand at the end of 1911 was over half the production of the State for that year. Thus is created a situation which is of great disadvantage to the producer of oil and of almost equal disadvantage to the public, since a large part of the surplus production is stored in open reservoirs where tests have shown that in some cases the volatilization loss is as great as 50 per cent. in five months. This loss is of the most valuable part of the oil and represents a tremendous and irredeemable waste.

CONCLUSION

It must be concluded that the placer law as applied to petroleum is unsatisfactory from almost every angle. It fails to adequately protect the prospector during the long and expensive exploration period. It permits in some cases the holding of a country side without real development or prospecting. It permits the extraction of oil from the territory of another and fosters and necessitates line drilling. It forces production in advance of demand and results in serious waste of a valuable resource. It alienates the fee from the government and endangers and renders difficult future government control.

It is not the purpose of the present discussion to endeavor to suggest new legislation. The objections to the placer law, however, force certain features of a satisfactory law upon the attention. The prospector should be given the exclusive right of possession of a restricted area for a limited period sufficient to prove or disprove its oil-bearing character. Thus the heavy investment required for oil exploration would be protected from loss by adverse claim. The operator should pay for his oil as a commodity in terms of barrels instead of as real property in terms of acres. If oil is the property of the producer thereof, the producer should pay for it, not the owner of the property from which it is drawn. The oil only should be sold and the title to the land should be retained in the government.

ADDENDUM

It might be well to add to this discussion a brief statement of the effect upon the *bona fide* oil operator of withdrawals under the act of June 25, 1910 (36 Stat., 847). This act provides:

"That the President may, at any time in his discretion temporarily withdraw from settlement, location, sale, or entry any of the public lands of the United States including the District of Alaska and reserve the same for water-power sites, irrigation, classification of lands, or other public purposes to be specified in the orders of withdrawals, and such withdrawals or reservations shall remain in force until revoked by him or by an act of Congress.

"Provided, That the rights of any person who, at the date of any order of withdrawal heretofore or hereafter made, is a bona fide occupant or claimant of oil or gas bearing lands, and who, at such date, is in diligent prosecution of work leading to the discovery of oil or gas, shall not be affected or impaired by such order so long as such occupant or claimant shall continue in diligent prosecution of said work."

The operator who has made a valid location supported by a discovery is of course not affected by a withdrawal under this Act. Neither is the operator who, although he has not as yet made discovery, is nevertheless engaged in the diligent prosecution of work leading thereto. A withdrawal therefore operates only against those claimants who are not diligently seeking a discovery. The effect of the withdrawal is to prevent the initiation of new rights upon the lands withdrawn by those not prosecuting work at the date of withdrawal. What is the net effect then upon the operator who was diligently at work at the withdrawal date, and who has continued diligently at work, but has not as yet made discovery? If there were no withdrawals it is entirely within the bounds of possibility, as already pointed out, for an adverse claimant to enter upon the land (if such entry be not forcible, fraudulent, or clandestine), drill upon it, and by first discovering oil, to obtain title to it. With a withdrawal order covering the land no such intrusion is possible. No new rights can attach subsequent to the withdrawal, and the operator who in good faith was diligently prosecuting work leading to discovery and who has continued to do so is protected by the withdrawal against all possibility of the initiation of adverse claims.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

The Origin of the "Garnet Zones" and Associated Ore Deposits.*

BY WALDEMAR LINDGREN, BOSTON, MASS.

DURING the last 15 years much attention has been given to the "contact-metamorphic" ore deposits which mainly occur in limestone close to intrusive contacts. In general, these deposits are characterized by the association of magnetite and simple sulphides with the so-called "contact-metamorphic" silicates. These silicates comprise garnet, epidote, vesuvianite, tremolite, wollastonite, diopside, hedenbergite, ilvaite, and many other rarer forms. The contact-metamorphic zones contain also in places minerals with boron, fluorine, and chlorine, like axinite, tourmaline, danburite, fluorite, and scapolite. There is also more or less quartz and coarsely recrystallized calcite. The simple sulphides embrace pyrite, pyrrhotite, chalcopyrite, galena, zincblende, more rarely arsenopyrite. On the whole, sulphantimonides and sulpharsenides are rare. Deposits of magnetite or chalcopyrite or of both are most common.

The deposits have an irregular or rudely tabular form, following main contacts, dike contacts, or stratification planes; they rarely extend for more than 1,500 or 2,000 ft. from the contact; far more commonly they only reach a distance of a few hundred feet from that surface. The deposits are usually formed by replacement of limestone, or allied calcareous rock. Apparently they are found both in relatively pure and in impure carbonate rock.

The deposits are geological bodies of a chemical and mineralogical character differing greatly from the adjoining intrusive rock and from the unaltered limestone. They are rich in silica and iron, with more or less of lime, magnesia, alumina, sulphur, and rarer metals like copper. The so-called contact-metamorphic silicates which form the gangue usually are coextensive with the metallic minerals, though of course not necessarily with the merchantable ore. All opinions agree in holding the deposits of epigenetic origin, later than the inclosing limestone.

Regarding the origin of the metals and the genesis of the ores there are several opinions.

1. There are those who see in these deposits the result of concentration of material contained in the intrusive rocks by means of hot circulating

* Introduction to a discussion at the New York meeting of American Institute of Mining Engineers, February, 1914.

waters of atmospheric origin; the circulation established being upward on the periphery of the hot mass.

"The heated ground water would be quite competent to do all that is ascribed to magmatic waters, including the formation of lime-silicate zones."¹

Among the minority who still cling to this view I find Professor Lawson. The latter, writing with his customary well-sharpened pen, describes the magmatic theory of these deposits as "one of the fashionable vagaries of our time, not entitled to serious respect as a scientifically established theory."² I am inclined to hope that Professor Lawson has already changed this view. If he has not I am sure he will before long, especially if he should undertake a somewhat careful study of a considerable number of the contact-metamorphic deposits in the Southwest and in Mexico.

In the first place, these deposits are "high temperature" deposits. The character of the gangue minerals shows that they were formed at considerable depth—some well within the zone of fracture, perhaps only 4,000 ft. below the surface—others deep down in the region where fracturing must be considered impossible. In both cases the essential characteristics are the same. According to our best information the lowest temperature at which garnets, diopside, etc., may form is about 400° or 500° C. In view of the well-grounded attacks lately made upon the theory of the deep circulation of atmospheric waters it devolves upon the supporters of the view set forth under (1) to explain the possibility of such a circulation upward along the contact at temperatures of from 300° to 1,000° C. The Daubrée experiment will really not suffice any more as an explanation.³

I shall try to make the real position of the leaching theory a little clearer: The metals, etc., were extracted from the intrusive rock—that is, from a hot body having a temperature of from 500° to 1,500°—and deposited in the adjacent, cooler sediments, say at temperatures from 300° to 1,000° C. Consequently the circulation of the atmospheric waters would really take place in a lateral direction and not simply *along* the contact as maintained by Lawson. The silicates present in the contact zone prohibit the assumption of concentration at lower temperatures.

No one who has observed many deposits of this kind can fail to be impressed with the fact that the ores are generally found at limestone points projecting into the intrusive mass, or in slabs of limestone swimming in the igneous rock. It is difficult to see why the circulating atmospheric waters should seek such places by preference. In many

¹ Lawson, A. C.: *Mining and Scientific Press*, vol. civ, No. 5, p. 201 (Feb. 3, 1912).

² *Loc. cit.*

³ Johnson, John, and Adams, L. H.: Observations on the Daubrée Experiment and Capillarity in Relation to Certain Geologic Speculations, *Journal of Geology*, vol. xxii, pp. 1 to 15 (1913).

deposits there is no evidence of fractures or paths which could have been followed by the water.

Inevitably, such a hydrothermal circulation as is suggested by Professor Lawson would result in a strong alteration of the intrusive. In many instances there is no such evidence of replacing solutions. Fresh granitic rock often borders the ore.

Mineralized later fractures may, however, cross these contact deposits and then the intrusive rock may be hydrothermally altered, but this does not prove that this is caused by atmospheric waters. It is clear, of course, that thermal metamorphism is not denied by Lawson; and he also admits a so-called "reactionary metamorphism wherein the changes are due to reaction between the encasing rocks and the materials emanating from the intruding mass."⁴ Thus for instance he would class the hornfels of Clifton and that from the Kristiania region as the result of thermal metamorphism. But the overlying limestone when it contained andradite and magnetite and sulphides would be a product of circulating atmospheric waters. Truly an odd position to defend!

It has been asserted that these deposits are exceptional cases, because they do not occur along all contacts or along the same contact. The answer to this is that the economically valuable *deposits* are exceptional but the *mineralization* is not, as one often can ascertain by carefully following one of these contacts.

2. There is another school, of which C. K. Leith is perhaps the most prominent exponent. These men do not by any means deny the actuality of emanations from the magma, but hold that the silicate gangue minerals are mainly the result of a recrystallization of constituents originally contained in the limestone, like calcium, magnesium, iron, aluminum, and silica. This involves of course a great reduction of volume since the latter three constituents are usually subordinate in limestones, the removal of a great deal of material, and a compression obliterating structures and textures. The additions from the magma are laid mainly to a late stage during which the silicates containing mineralizers and the iron ores were introduced. "The metallic minerals may be either direct contributions from the magma, or hot water deposits resulting from the working over of the hot intrusives."⁵ No statement is made as to the derivation of this water but atmospheric waters are probably meant.

Much effort has been expended in attempting to prove the first part of the proposition: namely, the development of the silicates by a reduction of volume of the limestone amounting to 40 to 80 per cent. The last paper with this end in view has been contributed by W. L. Uglow,⁶ who endeavors to show by comparison of analyses of fresh and altered lime-

⁴ *Loc. cit.*

⁵ *Economic Geology*, vol. viii, No. 1, p. 26 (Jan., 1913).

⁶ *Idem*, vol. viii, Nos. 1 and 3 (1913).

stones that the ratio of silica to alumina and ferric oxide is constant. The analytical material is evidently insufficient, especially as several of his analyses of "fresh" limestones already contain silicates. That no such constancy is shown has been clearly pointed out by C. A. Stewart.⁷

Few of the analyses have separate determinations of Fe and Al in the unaltered limestone. Were such determinations available I suspect that the non-constancy in the ratio Al:Fe would be still more apparent.⁸

Mr. Uglow also attempts a general *critique* of the literature and theory of contact-metamorphism deposits, but in this he is evidently handicapped by lack of field experience.

The main argument of the supposition of the "residuary crystallization" theory is that the impure limestones with much silica are the ones which yield the contact-metamorphic silicates. This is of course largely a question of facts. Evidence that cannot easily be refuted has been introduced to show that the contact-metamorphic silicates and the ores often form in very pure limestone. But it is conceded that they may also develop in impure limestone. According to the supporters of the residuary crystallization theory, a very large reduction of volume has been effected in either case, which of course would make the stratification a jumble near the contact. The expulsion of so much calcium and magnesium carbonate involves first a development of porosity and then a compression by the stresses from the intruding magmas. The reason why most silicate zones are composed of hard, compact rocks is therefore that the porous rock has been strongly compressed. (In some cases one is tempted to add that the rocks, upon this theory, must have been principally pores.) Later, upon cooling, contraction fissures appeared and received some emanations from the magma or material concentrated by hot water from the intrusive rock.

I must here refer to the Washington Camp in Arizona, described by Professor Crosby. According to his statement the silicates and ore in the Pride of the West mine have been formed from an impure limestone. I investigated this occurrence carefully in 1909 with F. C. Schrader, of the U. S. Geological Survey, and was reluctantly compelled to adopt a different view. It seemed to me that a stratum of particularly pure limestone had here been replaced, lying between beds of markedly impure and siliceous limestone which have suffered little alteration.

That part of the pure bed which is converted into ore and silicates presents the unusual feature that the altered material contains some rather large druses, which are coated by fine quartz crystals. In this case, then,

⁷ *Economic Geology*, vol. viii, No. 5, p. 501 (Aug., 1913).

⁸ I find that the analysis given by Mr. Uglow for the "Modoc limestone" at Clifton has been obtained by taking the average of the two analyses from Modoc mountain, which of course is correct, and one analysis, materially different, from Shannon mountain, 3 miles distant. This is, to say the least, a curious proceeding.

the pressure of the magma was not great enough to compress a drusy material into a compact mass.

It should be added that many of these silicate rocks as exposed at the surface appear porous and drusy, but this is usually caused by the solution of interstitial and residuary calcite.

Thus far no conclusive field evidence has been introduced showing the compression demanded by the "residuary" theory. We owe probably the best and most careful examination of a metamorphic zone to Professor Barrell.⁹ He concludes that in the outer metamorphic zone there must have been effected a considerable compression, for which he, however, found no convincing field evidence; while the inner zone, in which additions from the magma seemed clearly demonstrated, showed no evidence of reduction in volume. Theoretically, according to Barrell there should be a slight expansion in this zone.

It is obviously impossible that structures and textures could be retained on the residuary crystallization theory. That such structures are in fact often retained has often been shown and I have particularly called attention to the preservation of fossils in greatly garnetized rocks at Tres Hermanas, N. M.

F. C. Calkins and B. S. Butler have recently completed careful investigations of contact zones at Philipsburg, Mont., and San Francisco, Utah.¹⁰ In both cases evidences of porosity or compression were sought for but not found.

It seems as if a careful microscopic study of limestones in all stages of recrystallization would convince any one that no such wholesale reduction of volume has taken place. Delicate needles and prisms of amphibole or pyroxene or skeletal forms of garnets appear which are not disturbed or crushed by the metamorphism in more advanced stages. A part of the limestone may be replaced by quartz, and both calcite and quartz are without disturbance traversed by a system of amphibole prisms.

The tendency of crystallization in skeleton forms leads to the so frequent development of residual calcite included in the silicates. I can find little to contradict the conclusion that the metasomatism takes place by equal volumes and that there is practically no change in volume in the altered rock compared to the original rock.

The advocates of the residual crystallization theory reluctantly admit that boron, fluorine, etc., may have been introduced. Are they sure that these were the only elements introduced? It seems to me absolutely impossible to draw a line between the ordinary silicates and those containing mineralizers. Both are commonly intergrown, showing practically contemporaneous origin. Take for instance the occurrence

⁹ *Professional Paper No. 57, U. S. Geological Survey* (1907).

¹⁰ *Professional Papers Nos. 78 and 80, U. S. Geological Survey* (1913).

of axinite, which so frequently has been overlooked, the brown massive mineral looking much like garnet.¹¹ Sulphur has equal right to be regarded as a mineralizer and from Day's and Shepherd's work at Kilauea¹² we know that even the surface lava gives off much more sulphur than fluorine and boron. Even if nothing more is admitted in the way of emanations it is clear that sulphides may result in the adjoining rocks. Has the old experiment been forgotten showing the development of magnetite in a limestone fragment immersed in molten basalt?

I do not intend to deny for an instant that the impurities in a limestone may be utilized in its subsequent recrystallization and I have noted the wollastonite developing at the contact of chert nodules and limestone as described by Crosby from Washington Camp, but I do insist that if it is admitted that a massive andradite garnet may form from a practically pure limestone no amount of diagrams can prove that the constituents of the garnet can be accounted for by residuary crystallization.

3. The view that accounts for the silicate minerals, and the minerals containing mineralizers, and the iron ores, and the sulphides by assuming that they are derived in large part from the emanations from the magma seems to me so plausible, so convincing and so fitting when the subject is regarded from a broad standpoint that I cannot help joining Kemp, Spurr, Goldschmidt and many others in advocating it. I believe thoroughly that iron, silica, various rare metals, the mineralizers, to a smaller degree also alumina and the alkalies, are thus contributed by the intrusive. The limestone and the intrusive emanations form a chemical system in which reactions of great intensity are proceeding. Sometimes there is also a vigorous interaction between the limestone and the intrusive, as shown by the occasional development of epidote and garnet in the latter.

The emanations consisting of gases, water, and volatile compounds in which the metals are probably carried as chlorides or fluorides, penetrate the limestone easily. In connection with this I wish to call your attention to some diagrams and mathematical calculations presented by Messrs. Leith and Uglow in which it is shown how excessively slow the diffusion of heat is from an intrusive contact. No doubt the calculations are correct, but their presentation is an excellent instance of the danger of applying mathematics to geology. The diffused heat is as a matter of fact a feature of minor importance. The penetration of the emanations into the solid rocks is the main cause of the contact metamorphism. These emanations penetrate the suitable strata of limestone as if the latter were a sponge and they also travel rapidly on fissures and joints. Convection thus far outdistances diffusion. Every geologist who has studied contact

¹¹ Schrader, F. C.: *Bulletin No. 497, U. S. Geological Survey* (1912).

¹² *Bulletin of the Geological Society of America*, vol. xxiv, No. 4, pp. 573 to 606 (Dec., 1913).

metamorphism is well aware that certain beds, not unlike those above and below, show extraordinary selective power of recrystallization and mineralization. It is well known that some limestones are easily penetrable by oil, so that they may be rendered strongly translucent by such impregnation. If this can be effected at lower temperatures what would be the effect of emanations under tremendous pressure and temperature?

The principal differences among those who believe in the magmatic theory relate to the order of introduction of the various minerals. Some, like J. E. Spurr, hold that the development of silicates preceded the metallization as a more or less distinct stage and make of the latter a hydrothermal phenomenon taking place after the consolidation of the intrusive magma. Others see a rather continuous process in the metallization, holding that it began at the moment of irruption.

It has always seemed to me most difficult in this case to separate metamorphism and metasomatism. I am inclined to minimize the purely thermal metamorphism, and this opinion is strengthened by the extremely slow rate of diffusion of heat. On the other hand, it is well known, for instance, through the work of F. C. Calkins,¹³ that the emanations resulting in tourmaline and scapolite spread long distances from the contacts. It is therefore probable that the metamorphism which so often extends for many thousand feet from the contact is primarily induced by the transmission through the pores, fractures, and bedding planes of solutions emanating from the magma but mingled on their way with interstitial rock moisture. In fact, the slowness of the diffusion of heat positively compels such a view. The replacement will ordinarily proceed by equal volumes, but where the movement of the solutions was extremely rapid drusy texture might result. All this is on supposition that the locality is in the zone of fracture. If it is definitely in the zone of flowage, the temperatures could normally be much higher and the difficulty of escape of the carbon dioxide would naturally be much greater. Hence the development of silicates would proceed at a very much slower rate, so slow indeed that porosity would become improbable and any replacement would proceed on the basis of equal volumes; any material dissolved would immediately be replaced by new minerals.

In the diffusion of emanations from the magma certain constituents would penetrate much further than others. Carbon dioxide and other gases would certainly penetrate much further than silica, for instance.

There is unquestionably a certain sequence in the development of minerals in contact zones. Many observations seem to prove that emanation of sulphides and magnetite continues after the contact action, resulting in silicate rock, is completed, and finally hydrothermal action results.

¹³ *Professional Paper No. 78, U. S. Geological Survey (1913).*

The strong advocates of this view, among whom J. E. Spurr is prominent, seem, however, to minimize unduly the earlier development of sulphides and magnetite. I have often had occasion to emphasize these early accessions which I think cannot be denied. The recent literature contains little to make a change of opinion necessary. B. S. Butler,¹⁴ for instance, says of the Frisco district, Utah, that "in general the minerals of the contact zones appear to have formed at essentially the same time, though in some places metallic minerals have formed later than the silicates as well as contemporaneously with them." T. Kato,¹⁵ describing the deposits at the Okufo mine, Japan, states that wollastonite is the oldest mineral and that it was formed by accessions of silica. He considers the earliest solutions to have been very siliceous. Next follows andradite with traces only of alumina; this is intergrown with vesuvianite. The sulphides were formed contemporaneously with the garnet or at the closing epoch of its deposition. Veinlets of chalcopyrite form the latest manifestation of metallization.

In conclusion, I would say that the most convincing argument in favor of the derivation of these deposits from emanations from the magma is not this or that single fact, but the great chain of circumstantial evidence which connects the contact-metamorphic deposits with the high-temperature veins on one hand (such as the cassiterite veins, for instance) and the pegmatitic veins and dikes and concentrations in molten magmas on the other hand. In some cases evidence of this kind, with grading to ordinary veins, indicating lower temperatures, is offered in the vicinity of a single intrusion. If to this is added the direct evidence of emanations from the surface lavas, for instance of sulphur and fluorine from the Hawaiian volcanoes, the whole forms a well-supported theory which should require respectful consideration even from those who would attribute a paramount rôle to the atmospheric waters.

¹⁴ *Professional Paper No. 80, U. S. Geological Survey*, p. 84 (1913).

¹⁵ *Journal of the Geological Society of Tokio*, vol. xx, pp. 13 to 32 (1913).

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

Some Recent Developments in Electric Coal-Mining Machines

BY S. B. KING,* CHICAGO, ILL.

(New York Meeting, February, 1914)

It is the purpose of this article to give those not familiar with the subject an idea of the extent to which coal is mined by machine in the United States; to describe some types of electrically operated coal cutters; and in particular, some of the latest improvements and developments in these machines. Generally speaking, the article deals with the mining of bituminous coal in room-and-pillar mines, as the use of undercutting machines in the anthracite region of Pennsylvania is in its infancy, and when machine mining is referred to bituminous coal is taken for granted. The number of mines in the United States that are operating on the long-wall principle is very limited, and the number of long-wall machines in use is a very small percentage of the total.

The advantages of mining by machine may be summed up briefly as follows:

A. Increased safety to property and life, due to the large decrease in the amount of powder required when the coal is properly undermined.

B. Reduction in the cost of mining.

C. Improvement in the quality of the product; and, consequently, increase in market value, due to the greater proportion of lump and screen sizes secured by machine mining.

D. Increase in the capacity of the mine, and more rapid development of new mines.

E. Ability to mine veins in which the height of the coal or the character of the roof has prevented mining by hand, on a commercial basis.

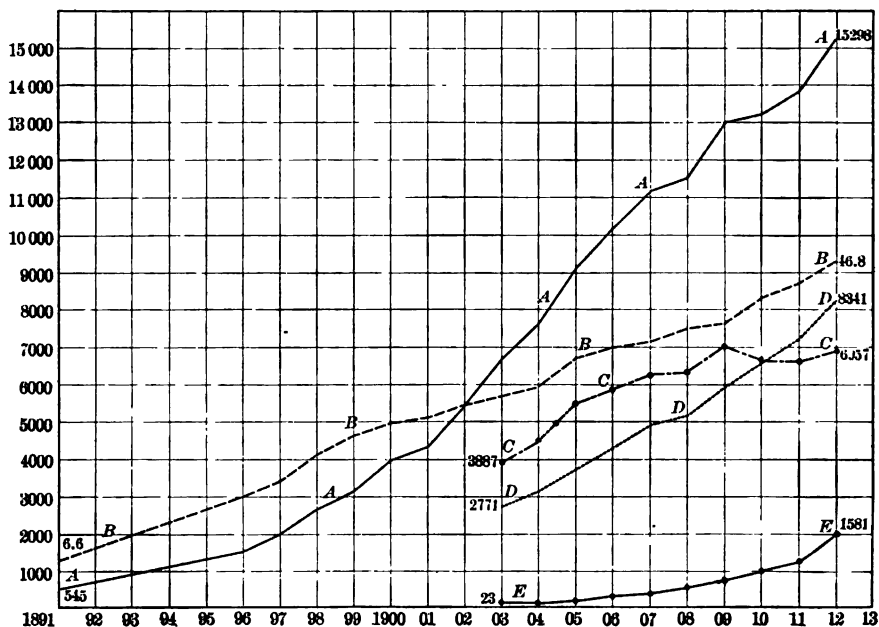
Growth of Machine Mining

Machine mining of coal in the United States is confined commercially to the past 20 or 22 years. In 1891, the first year for which records are available, there were 545 mining machines in use. The total production in that year was 93,177,978 tons, of which but 6 $\frac{1}{2}$ per cent. was mined by machine. In 1912, the latest year reported, the total tonnage had increased to 450,104,982 tons, and the number of machines to 15,298. The tonnage mined by machine was 210,538,822, or 46.8 per cent. of the

* Non-member.

total. The growth in the use of machines is indicated by the curves in Fig. 1. In this chart, curve *A* shows the number of machines in use from year to year; curve *B* shows the percentage of the total product mined by machines.

Both compressed air and electricity were employed at an early date as a motive power for mining machines. These machines are of two general patterns: reciprocating or pick machines, operated by air power; and chain or cutter-bar machines, operated by electricity. Some compressed-air chain machines are in use, but, as yet, they form an almost negligible proportion of the total. Referring again to Fig. 1, curve *C*, beginning at



A = Total number of coal-mining machines in use.
B = Percentage of total of coal production mined by machines.
C = Compressed-air pick machines.
D = Electric machines—all types.
E = Continuous-cutting electric machines.

FIG. 1.—CHART SHOWING NUMBER OF COAL-MINING MACHINES IN USE AND PERCENTAGE OF COAL PRODUCTION MINED BY MACHINES.

1903, shows the progress in the use of compressed-air pick machines, with which are combined in the later years a few post punchers or column coal cutters. Curve *D* shows the growth in the use of electric machines of all classes. It will be noted from these two curves that while the growth in the use of electric machines forms a curve ascending uniformly, the compressed-air curve is more irregular, and since 1909 has fallen off rather sharply, with a partial recovery in 1912. This relative increase in the use of electric machines may be credited to two things:

A Increasing popularity of electric current for other purposes about the mine than operating machines, with especial emphasis on electric haulage motors.

B The improvement in and development of types of electric coal cutters of increasing adaptability and capacity.

There are three principal types of electric chain machines in commercial use: the breast machine, which is the oldest pattern and the one most widely known; long-wall machines of various types; and side-cutting, continuous-cutting, or short-wall chain machines, as they are variously known. This last-named type is of most recent introduction, and is at present the machine which offers the greatest possibilities in capacity and adaptability to working conditions; and in the past two or three years its use has made remarkable strides. Curve *E* on Fig. 1 shows the growth (estimated from private sources) in the use of this particular type of machine, dating from 1903. It is with this continuous-cutting machine that this article will deal in chief.

The particular machine described and illustrated is that manufactured by the Sullivan Machinery Co., of Chicago. This was the pioneer in the side-cutting machine field for room-and-pillar work and is, further, the machine with which the writer is most familiar.

Before passing to a description of this type, it may be noted that there is a discrepancy between the production of coal per year per machine and the known increase in productive efficiency, secured by side-cutting machines over chain breast cutters and pick machines. In 1912, the production per machine was about 13,000 tons; no more, as a matter of fact, than the average production per machine per year ten or a dozen years previously. There were various factors affecting this condition. One of them is that improvements in the machines have enabled them to be used in low coal and in mines which could not be worked by hand or by other machines, and from these, of course, the tonnage is relatively low.

The Sullivan Ironclad continuous coal cutter, the latest model of the side-cutting machine, consists of a motor, either direct current or alternating current, operating through shafts and gearing a continuous chain, armed with cutter bits, which runs around a cutting frame or bar 21 in. in width and ranging in length from 5 to 10 ft. The motor and gearing and chain-drive mechanism are inclosed in an armor-plate casing, and these, with the cutter bar, are mounted for transportation on a skeleton frame or pan. The whole outfit is carried from place to place on a self-propelling truck, operated through sprockets and chains.

Motor

The alternating-current motor will be described later. The direct-current motor, which has been developed through a dozen years' service,

is of the multi-polar shunt or compound-wound type, set horizontally. Early motors were set vertically. The horizontal arrangement makes the machine lower and more compact. The armature is drum-wound, with formed coils imbedded in slots, and is built on a separate sleeve, so that the shaft is removable. The motor is tested at the factory at 30 h.p. for one hour continuously, totally inclosed, or three hours continuously at this load with open construction. The average power consumed when cutting coal is 14 to 22 h.p. for an average period of 30 min. cutting, depending on the hardness of the coal and the speed of the machine, so that if eight rooms are cut in 10 hr. time the machine would deliver from 40 to 66 per cent. of the rated capacity, for a total of 4 hr. For the remainder of the time, while loading and moving the machine from place to place, about 4 h.p. are consumed. Insulation tests are made at the factory, under a pressure of 5,000 volts alternating current. The motor and gearing are fully protected by a tight casing. Access is had to the commutator and brush holders through a hand-hole cover in the top of the frame, fitted with a protected glass window, so that the action of the commutator and brushes may be observed while the machine is at work. The controller is inside the housing and is equipped with a magnetic blow-out coil.

The electric connections are mounted on a slate base, inclosed in a tight box. The reverse switch is interlocking, and cannot be thrown out when the machine is at work. The rheostat used is made of laminated iron punchings, insulated between with mica, so as to be practically indestructible from burn-outs and grounds.

The cutter bar is of rigid built-up construction, with the top and bottom smooth, so that it cannot be blocked by falling coal or caught by rough bottom. The cutter bits are set in blocks in a strap link chain, and the construction of the guides is such as to practically eliminate side play. The bits are staggered, in from five to nine positions, depending on the character of the coal to be cut. Either pick or chisel point bits, or both, are employed, depending on local conditions. When the coal cores, or when the cutting is in thin sheets of sulphur, the larger number of positions is used to prevent the links from being cut out.

Method of Feeding

The machine is moved from place to place, and handled in all of its operations, by a heavy chain of standard link patterns, which engages sprockets, operated by feed gearing, this being protected by an adjustable friction clutch. This friction clutch is set by a dynamometer to slip at a certain pull, slightly greater than that required to cut average coal, and well within the limit of strength of the feed gearing. This constitutes an absolute safeguard against strain or breakage. The machine weighs 5,000 lb. and stands 23 in. in height, so that it may be used in any coal that

may be developed on the room-and-pillar plan. It is at present being used in mines in which the coal is only 16 in. thick, and in which roof and bottom must be taken up to permit the passage of the machine. In this case, the cutting is done in the fire clay below the coal.

Method of Operation

To describe the method of operation of the continuous-cutting machine, the reader is asked to refer to Figs. 2 and 3. Fig. 2 shows the plan used in cutting mine rooms with the familiar chain breast cutter. As

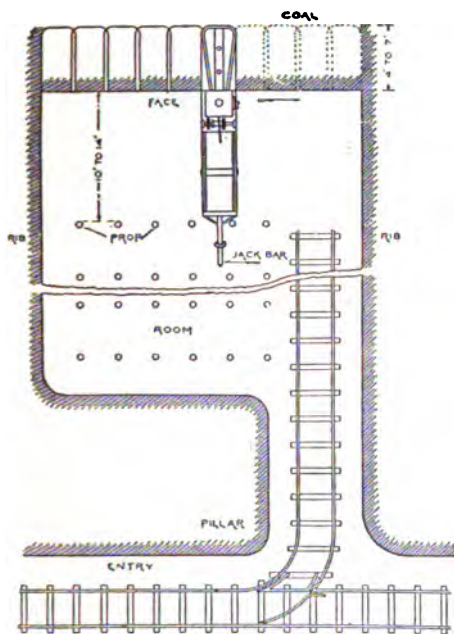


FIG. 2.—ROOM-AND-PILLAR MINING WITH BREAST MACHINE.

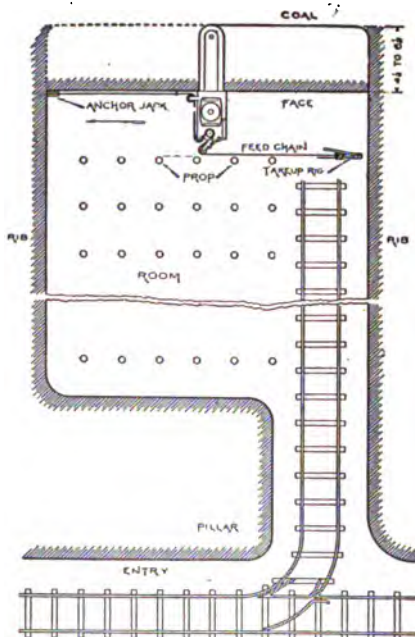


FIG. 3.—ROOM-AND-PILLAR MINING WITH SULLIVAN CONTINUOUS COAL CUTTER

will be noted, the machine cuts the coal in a series of runs, or bords, each about the width of the cutter bar. It is held to its work for each run, or cut, by a rear jack—sometimes by an additional front jack or prop. When the cutter bar has been run to its full depth under the coal, the motor is reversed and the bar backed out. The props or jacks are then taken down, and the operator and his helper move the machine along the face until fresh coal is encountered, for a new bord, or run. This work is done with crowbars and is very laborious. The jacks are then reset and the operation is repeated.

It will be seen that the necessity for backing out the machine for each

fresh cut makes necessary a space between the face and props sufficient for this purpose, which ranges from 11 to 14 ft. with this type of machine. In case the roof is so poor that this cannot be done, then the props must be



FIG. 4.—MACHINE ENTERING THE ROOM.

moved as the machine is moved across the room. In weak top, the continued bracing of the rear jack bar against the roof, and the consequent weakening of the latter, are an additional cause of danger.

Fig. 3 shows the method of operation of the continuous coal cutter.



FIG. 5.—UNLOADING MACHINE FROM THE TRUCK.

After the sumping cut has been made at one of the ribs, usually the right, in the same manner as with the breast machine, the machine leaves its frame, or pan, the chain is attached to an anchor jack at the left-hand cor-

ner of the room, and behind the machine to a ratchet or take-up rig. It is then started feeding along the face under its own power, cutting as it goes, and is not withdrawn until the further rib is reached. The series of



FIG. 6.—BEGINNING THE SUMPING CUT.

photographs, Figs. 4 to 12, show the cycle of operations of the Ironclad machine. They were taken especially for this purpose at the Furnace Run mine of the Allegheny River Mining Co., near Kittanning, Pa.



FIG. 7.—END OF SUMPING CUT.

Fig. 4 shows the machine on its self-propelling truck, moving from the heading into the room. The coal in these rooms was from 36 to 42 in. high. This picture shows the customary method of transporting all

equipment used with the machine, and sometimes one or both of the operators, upon it while moving.

In Fig. 5, the truck wheels have been blocked and the machine is unloading from its truck on to the floor of the room, traveling on the feed chain, which is anchored by a jack, shown at the right-hand side.

In Fig. 6, the machine has been squared with the right-hand rib and is ready to make the sumping, or rib, cut—in this case on an angle iron or sumping bar, used instead of the regular pan or frame.

Fig. 7 shows the machine at the end of the sumping cut. The take-up rig and front jack have been set in position, to permit the continuous undercut across the face. The cutter bar is under the coal and the machine has already begun to cross the face.

The cutter bar is ordinarily under the coal, from one side of the room to the other, cutting to the full depth of the bar and making guards for the



FIG. 8.—CUTTING OUT SULPHUR BALLS.—FIRST POSITION OF MACHINE.

cutter chain unnecessary. This is not the case, however, when impurities such as sulphur balls, spar, or clay veins are encountered. The way in which such difficulties are overcome deserves explanation. It is, of course, undesirable to leave these in the coal, as they hinder proper shooting and rolling out of the coal, after the cut is completed. It is also undesirable to back the machine out and re-sump it on the other side of the obstruction. The procedure with this machine is as follows: The feed chain, at the take-up rig behind the machine, is slackened up, allowing the body of the machine to cut ahead at an angle, as shown in Fig. 8. This cuts the coal away in front of the obstruction. When this has been done, the machine is drawn back on its feed chain, to the right, and the point of the cutter bar forced to feed ahead, behind the obstruction. When it is thus freed on three sides, it is usually easy for the machine to

jerk out the sulphur ball, and cutting is then resumed in the usual way. Fig. 9 shows the machine in the second position referred to above.



9.—CUTTING OUT SULPHUR BALLS.—SECOND POSITION OF MACHINE.

Fig. 10 the machine is shown at the left rib, ready to back out. The point of the cutter bar has been thrown to the left by manipulating the feed chain, so that the point of the bar will follow the line of the feed chain,



FIG. 10.—MACHINE READY TO BACK OUT.

as shown at the left of the picture, on its way out, leaving a perfectly straight, square rib.

Fig. 11 shows the machine reloading itself on to the self-propelling truck.

The room in which this machine has just been seen at work is shown in Fig. 12 after having been shot down, the center or break-down shot having



FIG. 11.—RELOADING THE MACHINE ON THE TRUCK.

been fired. The rib shots have not yet been discharged. This shows the freedom with which the coal is thrown out, after the machine undercuts, and the large proportion of lump coal which it contains.



FIG 12.—VIEW OF ROOM AFTER BREAK-DOWN SHOT.

Referring to Fig. 8, an element of interest is the nature of the cuttings made by the machine. This picture and some of the others show how coarse these cuttings are, being, not dust or slack, but often of a grade suitable for marketing as pea coal, or even as chestnut. The height of the

mining, or kerf made by the machine, is $5\frac{1}{4}$ in., and in the case shown in the pictures was $6\frac{1}{2}$ ft. deep under the coal. As intimated above, three shots are required, the holes being bored one in the middle and one at each rib, $6\frac{1}{2}$ ft. deep, straight in from the face. The rib shots consisted of cartridges 14 in. long and $1\frac{3}{4}$ in. in diameter; and for the center shot, a cartridge 18 in. long by $1\frac{1}{4}$ in. in diameter.

As will be seen from the description and photographs, the method of operation of the machine secures for it a much higher cutting speed than is possible with the breast machine or with the puncher. It is safer to operate, because it is so compact that props may be set permanently within 6 ft. of the face, the machine passing between them while cutting across the room. In tender coal, which has a tendency to fall while being cut, the narrow, solid cutter bar and the method of operation permit the machine to proceed without delay, cutting itself free in all cases. The continuous cutter moves uniformly along the floor, leaving an open, free



FIG. 13.—ALTERNATING-CURRENT IRONCLAD CONTINUOUS CUTTING MACHINE.

cut with no ribs or sprangs at the back to hinder shooting. It follows rolls or unevennesses in the floor readily. The small space which it occupies in front of the face means less work in clearing away the débris after shooting, and also enables cars to be brought closer to the face of the room, facilitating loading. All operations of the machine are made under its own power. No hand barring or heavy lifting is necessary. This is an important feature, particularly in low coal, as it permits the use of a heavy, substantial machine of adequate strength for the severe duty which coal-mining work imposes upon it.

In flexibility and adaptability to working conditions, the Ironclad stands second only to the puncher or pick machine. When first introduced, it was thought that its field would be confined to wide places or rooms or long-wall faces. The speed with which the machine can be handled, and with which it cuts, however, have disproved this idea, and it is now regularly used for cutting entries and break-throughs as narrow as

7 or 8 ft. Perfectly straight entries are driven, by using the pan or starting frame at the sumping rib, and at the other rib, by cramping the cutter bar, as illustrated in Fig. 11. In an Illinois mine, a new runner has cut seven 8-ft. entries in less than 5 hr. In the same mine, another machine cut thirteen 9-ft. entries in $7\frac{1}{2}$ hr., including all movements of the machine, with two moves of over 1,000 ft. One entry was cut in 13 min., including unloading and setting the machine. In drawing pillars, this machine is more valuable than any other, because of its ability to cut in either direction, so that it can always cut away from an unstable roof. The machine lends itself particularly to pitching coal. It has been operated successfully and without extra labor—that is, with the usual runner and helper—in rooms driven across a pitch of 25° and in rooms and entries driven up a pitch of 20° . An element in the successful operation of the machine which is of particular interest is the method of protecting the feed gearing by means of the friction. The speed of the continuous cutter along the face of the coal is regulated by feed gears, supplied for speeds of 12 to 36 in. per minute. When cutting harder than the average is encountered, or when impurities occur, the friction slips; as the cutter chain is positively driven, it maintains its speed, thus allowing the machine to keep in motion at a reduced speed, and to cut much harder material than if the feed gears were positively driven, as well as preventing strain and breakage. If the runner should try to continue cutting after the bits are too dull the friction will indicate the need of fresh bits, by slipping. The protection which this friction affords to all parts of the machine may be indicated by one or two concrete examples. In the No. 5 coal seam at Bicknell, Ind., a band of sulphur reduces the mining space to a strip only 3 to 8 in. high, and in this restricted area, sulphur balls and black-jacks frequently occur. When the top or bottom of this small space is struck, or when sulphur balls are encountered, the friction slips, thus preventing damage to the gearing and other parts of the machine. It is frequently necessary in a room in this seam to replace the dull cutter bits four or five times, and yet, over a period of years, during which the machines have been at work, the cost of repairs on the machines themselves has been relatively light.

At a New Mexico coal mine, operating in anthracite, the seam is 32 in. high, and pitches 25° . It carries sulphur balls and seams near the floor. There is also a seam of draw slate, from 1 to $2\frac{1}{2}$ ft. thick, overlying the coal. This comes down with it, and must be drawn. It is often necessary to have the gob within 5 ft. of the face. Sullivan machines were installed here, and men who had never run them before were employed to operate them. Owing to the restricted space in front of the coal, it is impossible for them to back out the machine and sump it ahead of an obstruction. They are unable to tell what they are cutting, or whether the bits are dull or sharp, by the sound of the machine, and therefore

trust to the machine itself as long as it will feed. When the friction ceases to feed the machine is stopped, the bits are changed and work begun again. In this instance, if the friction arrangement were not present, it would be impossible to cut coal, without tearing the machine to pieces. It is not unusual to put in fresh bits four or five times in one room 40 ft. wide.

In a Colorado mine, where these machines were installed to make it possible to work a seam with an especially poor roof, unlooked-for advantages were secured. It is not only possible to prevent the roof from falling, but the proportion of lump coal has been increased 20 per cent. over the output when mining by hand, and the cost of production has been decreased 18c. per ton.

Recent Improvements

Reversibility.—A number of improvements and new developments have recently been made in the machine. The first and perhaps the most interesting of these is the fact that, as now manufactured, these machines are reversible; that is, by reversing the cutter bits in the chain the machine may be operated from right to left in the room, or from left to right, as required. It is necessary in some mines to cut some rooms from left to right and others from right to left, and this ability to reverse the direction of cutting enables one machine to take the place of two, as previously constructed, and to cover a territory in which both right-hand and left-hand rooms are found. The operation of reversing occupies only a few moments.

Increased Depth of Undermining.—It used to be accepted as axiomatic in coal-mining practice that coal could not be undermined to a depth greater than the thickness of the seam. Recent practice has shown that this rule does not hold good under certain conditions, and that the depth to which a given seam of coal may be mined depends on other circumstances than the height of the face. Among these may be mentioned the character of the roof and bottom, the presence or absence of impurities in the coal or of partings, and the character of the coal itself; that is, whether it is solid and tight, requiring hard shooting, or whether it breaks out freely and rolls forward with the expenditure of little powder.

There are a number of districts in which the coal is being regularly undercut to a greater depth than the height of the seam.

In this connection, the Ironclad machine promises to be of increasing value to coal operators, due to its ability to cut to greater depth than has been considered possible until recently. The longest cutter bar employed up to a year or two ago on a continuous-cutting chain machine was $6\frac{1}{2}$ ft., although some breast machines have been fitted with bars as long as 7 ft. 6 in. At Gary, W. Va., in the mines of the United States Coal & Coke Co.,

it was determined by experiment that the coal could be undermined safely and economically to a greater depth than the height of the seam. Cutter bars 8 ft. long were satisfactorily handled by the Sullivan machines, due to their substantial and powerful construction. Later on, bars 10 ft. 4 in. in length were installed, and run without damage to the machines, and without developing other undesirable factors. This increase in the length of the cutter bar, of course, increases the efficiency of the machines materially. It has been found at Gary, and at other points where the longer bars have been adopted, that the time and power required to cut to the greater depth are only slightly in excess of the amounts required to cut to a depth of 6 or $6\frac{1}{2}$ ft. This, of course, depends on local conditions, which at Gary are favorable for this excellent performance.

At Gary, the coal in the different mines is of different heights, ranging from $4\frac{1}{2}$ to 8 or 9 ft. By using short cutter bars in the high coal and longer cutter bars in the low coal, it is possible to arrange the cutting so that the loaders are able to load the coal cut in each room in a certain stated time, and so as to give each loader a full day's work at all times, no matter what the height of the coal.

Self-Winding Reel.—An improvement of minor importance is the self-winding reel for the self-propelling truck. This device enables the machine to pick up its own cable, after it has been extended, in entering a room or heading.

Inclosure of Motors.—The motors of the machine as at present designed meet the requirements of the new mining laws of the United States, and in various foreign countries as well. The motor may now be changed readily from the open type to the totally inclosed, or to the flame-proof pattern, by alterations in the field. Frequently, electric machines are installed in mines where there is no indication of gas. As development advances gas may be encountered, necessitating a gas-proof machine. In earlier designs a new motor of larger dimensions would have been required, to permit running totally inclosed. The Ironclad is designed to operate totally inclosed, without danger of overheating, and the motor frame is arranged to permit the changes mentioned, at small expense.

Alternating-Current Motors

Perhaps the greatest advance in the construction of electric mining machinery is indicated by the fact that alternating-current motors are now available for this class of machinery. The rapid increase in recent years in the distribution of electrical power from central stations has brought with it increased demand for machinery operated by alternating current, thus obviating the cost of a motor-generator set to convert the alternating current carried by transmission lines to direct current. Fig. 13 shows the Sullivan alternating-current machine.

The motor employed in this new pattern is of the squirrel-cage induction type, and may be wound for 220, 440, or 500 volts, and for 25 or 60 cycles, as desired. It is rated at 30 h.p., and is completely inclosed so as to be gas-proof. All joints in the heavy steel-plate motor casing are carefully machined, to insure protection against the entry of gas. This type possesses several distinct advantages over other forms of polyphase motors. The controller has three positions—neutral, starting, and full load. When starting, the lever is thrown over to the right on a star, or "Y," connection. This admits 50 per cent. of the current. The lever is then thrown quickly to the left, on to the full-load position, indicated by a Delta. This method of control by varying the electro-motive forces applied to the motor leads to two important advantages—one mechanical, the other electrical. With this form of control there are no regulating appliances on the motor, which is consequently of a very simple form. The electrical advantage consists of low magnetic leakage, absence of brushes, slip-rings, etc. A further advantage lies in the high pulling-out torque. If a heavy overload or one having great inertia be suddenly thrown on a motor that has a high torque curve, the point of maximum torque may be passed for an instant, and the motor will be stopped unless the load is quickly removed. A squirrel-cage motor of good design may have its speed pulled down, but this reduction in speed gives an increased torque, thus enabling the motor to carry the overload. This motor has a starting torque larger than the full-load running torque, and is thus able to start almost any load within its rated capacity. The power factors are high and the motor efficiencies excellent over a wide range of load. The squirrel-cage motor will stand all kinds of abuse, and is practically indestructible from burn-outs due to overload. It is equipped with ball-bearings.

In regard to power consumption, it may be stated that, due to the fact that this motor is very fast in starting, although it occasions a high load to be thrown on the power line temporarily, there is very little difference in the actual wattage consumed in starting either this type of motor or the slip-ring motor, because the latter takes much longer to get up speed. The machines equipped with this alternating-current motor have been found to operate very successfully in hard cutting. If the load thrown on the machine is too great the motor will stop, and this provides an added element of protection to the mechanical parts of the machine.

One of the fields in which these machines have been installed with notable success is in Colorado. Many of the mines in this field are within a short distance from the transmission line of some one of the large central electric power companies. For a mine under development, the alternating-current machine is a valuable adjunct. The high cutting speed of the machine and its adaptability to mine working conditions result in rapid opening up of working places and in economy of time and labor, while a

transformer constitutes all the power plant necessary. In the Colorado field, observations on the alternating-current machines that have been installed there indicate that they possess a capacity and ability to overcome difficult conditions, even more noteworthy than the same machines equipped with direct-current motors. In point of power consumption, the alternating-current machine has an advantage over the direct-current type in nearly all cases. The efficiency of the motors is practically the same. The fact that the alternating-current machine maintains constant speed, and therefore has a constant cutter-chain speed, makes its power consumption less per unit of cutting on the average per day than that of the direct-current type, as the latter slows down when overloaded or up against hard conditions, which increases its power consumption per unit of cutting.

When to Install Alternating-Current Machines

In deciding whether to install machines for alternating or direct current a number of factors are to be considered. When power is purchased from a power company, it simply is run through a transformer to bring it to proper voltage for the alternating-current machine; while if the same power is to be applied to a direct-current machine it must first pass through the same transformer, and then through a rotary converter or a motor-generator set. The rotary converter will operate at from 90 to 93 per cent. efficiency, and the motor-generator set at from 75 to 85 per cent. efficiency, so that the over-all efficiency of the direct-current plant in this case is considerably less than that of the alternating-current plant. Transformer losses are small, and are the same in both cases.

This same situation exists in the case of a large coal company, having a group of mines equipped with central alternating-current plant, and distributing to the various mines. If direct-current machines are installed in these mines, a mile or more away from the power house, a motor-generator set or a rotary converter must be installed; whereas if the alternating-current machines are adopted transformers are sufficient. Motor-generator sets or rotary converters cost about three times as much as transformers, installed in the same size units, and transformer stations can be increased or decreased more easily than motor-generator sets, provided that some comparatively small sized transformer is adopted as the unit. Transformers are also much more easily handled and moved from place to place in the mine, as the mine is developed, and for this reason a better voltage is likely to be had, with smaller wires and smaller copper cost, than where the direct-current machines and motor-generator sets are used. The motor-generator sets cannot be moved ahead as frequently as transformers.

In a mine of sufficient size to use haulage motors direct current must be available to operate them, as the alternating-current haulage motor is

not yet a commercial success. While it may be difficult for mine owners to see the advantage of running the chain machines separate from the haulage circuit, it is in general advisable to adopt this practice for best results. The development of mines frequently causes the sizes of wire employed to become too small, although they may be of proper size when installed. This results in low voltage for the chain machines when the haulage motor is in their territory, causing delays for both the machines and the haulage motors. If the mine is a large one, covering, say, a section of ground, the trolley wire cannot be carried large enough to carry the chain-machine load at the boundary of the property, without feeders of large size to help it out. It would seem practicable to run these feeders separate, and use them as a chain-machine circuit, even on direct-current machines; and if alternating-current machines are adopted the copper cost will be very much less in the end, provided the State laws do not prohibit the use of voltages as high as 2,300 or 440 to transformer stations, even though 250 volts may be used on the machines themselves.

Another condition frequently encountered is that of a mine located along the power line of some large central power company. There are a number of such mines in southern Colorado, having an output of from 100 to 400 tons per day. The power company charges a flat sub-station charge of \$50 per month for connecting up to a mine, and \$1 per kilowatt for each kilowatt of transformer capacity per month. In addition to this, the sliding rate per kilowatt-hour of the actual power consumed will range from 1.6c. per kilowatt-hour to 0.8c. per kilowatt-hour, depending upon the quantity consumed. For a plant using one chain machine, they figure on a 30 kw. transformer capacity. For a two-machine plant, they figure on installing 45 kw. transformer capacity, and, from their experience, they estimate that an Ironclad alternating-current machine uses on an average 600 kw-hr. per month. The power company installs all wires, poles, transformers, meters, switches, lightning arresters and connections, up to and including the switch on the secondary side of the transformer. A company at Walsenburg is connected on this basis, and is producing 125 to 130 tons of coal per day, cut by one alternating-current machine. A fan, pump, shaker screen, and pulley are all operated by alternating-current motors and their power bill now runs from \$105 to \$110 per month. This would not pay the wages of the fireman necessary to operate a power plant, and the original investment of boilers, generator, and engine is, of course, eliminated. The coal company estimates that about \$30 of this amount is a proper proportion to be charged against the chain machine. Twenty-five hundred tons per month has been a recent average of machine-mined coal at this mine. If the alternating-current machines had not been available for this property a motor-generator set and a direct-current machine must have been installed. The motor-generator set would have cost from \$900 to \$1,000 installed, which shows considerable

difference in favor of the alternating-current machine in the first cost. The power bill and the expense in upkeep of the motor-generator set would also run their maintenance charges up considerably.

Still another case for consideration is that of a mine owning a few direct-current machines, which has reached the limit of generator capacity. The question arises whether or not it would be advisable to put in an alternating-current generator, to take care of five or six more machines of the alternating-current type. If the mine is permitted to use only 250 volts on the ground, it should undoubtedly continue to install direct-current machines. If possible to run in with a 440-volt, 660-volt or 2,300-volt line, properly protected, stepped down to 220-volt machines, it would be advisable to put in alternating-current generators and alternating-current machines, rearrange the circuits underground so that the direct-current machines could be kept close to the power house, and carry the alternating-current lines out for machines operating toward the boundaries of the property. This applies where the alternating-current machines will be located 2,500 or 3,000 ft. or more from the power house.

From the above, it would appear that the field for the direct-current machine is in the small isolated mine, with individual power plant, or in mines located in States where voltages are limited to 275 volts, and the mine is already equipped with direct-current machines. Where only one or two machines are to be used this leaves an extremely large field for the alternating-current machine, and it is probably safe to say that on new installations the alternating-current chain machine can be applied in seven cases out of ten, and the outlook for the increased use of this type of machine is very bright. The adoption of alternating-current chain machines and the development of a motor of this type have been handicapped hitherto by the backwardness of the electrical manufacturers in bringing out a commercially successful haulage motor, of the alternating-current type, but from the above discussion it will be seen that there are many cases in which alternating-current coal cutters may be used to advantage, leaving the factor of haulage out of the question.

To give an idea of the actual cutting capacity of the Ironclad continuous coal cutters and of the actual power and time consumed in the various operations of the machine, the following paragraphs are quoted from an article by G. W. Thomas.¹ It may be said, roughly speaking, that the capacity of a breast machine, under favorable conditions, is twice that of a puncher, and that of an Ironclad, twice that of a breast machine.

"The test reported below is of particular value, because in it especial efforts were made to secure accurate observations and to make the test run conform to actual working conditions.

¹ *Mine and Quarry*, vol. vi, No. 4, p. 628 (June, 1912).

"It was conducted between December 13, 1911, and January 13, 1912, in Mine No. 2 of the Lumaghi Coal Company at Collinsville, Madison County, Illinois, upon a Sullivan 'Continuous Cutter.'

"The test was to determine the cutting capacity and power consumption of the machine, as shown by measurements, time observations, and readings from a recording wattmeter."

"The number 2 Lumaghi Mine [at which these tests were run] . . . works the No. 6 seam, which is 8 ft. high and practically level, although rolls or horsebacks occur in some localities. The coal varies in hardness in different parts of the mine, but is free from sulphur or other impurities. There is a good slate roof, so that close timbering is not required, and mining is done in the coal, as close as possible to the fire-clay bottom. The mine was formerly worked on the double entry plan, but the panel system is now in vogue.

"Two areas were assigned for the test, in one of which the coal was rated as 'hard' in the other as 'soft,' and a territory of 24 rooms in each was set aside. The work was done in these rooms and in cross-cuts between the rooms. The places ranged from 18 to 45 ft. in width, and averaged 31 ft. The Sullivan machine first cut in the 'hard' territory, working three alternate days, single shift. The same procedure was followed in the 'soft' territory, so that the test covered six shifts. Cutting in this way, over the territory described, enabled the loaders to keep enough rooms cleaned up so that a full day's cutting was ready for the machine every other day.

"The readings and measurements were made by two observers, who checked each other's work and who made affidavit at the conclusion of the test as to the fairness of its conditions and the accuracy of the results contained in their reports.

Taking the Readings

"The observations taken in each working place included the time, at the beginning and end of each part of the operation; the power consumed, shown on a recording wattmeter of standard make; the average voltage, shown by a voltmeter in the room; the width of the room or break through and the depth of the undercut. After each room was completed, the wattmeter was reset at zero.

"No readings were taken while the machine was moving on its truck, as the test had to do only with actual mining, and, since the length of the moves varied, observations began in each room with the unloading of the machine from its truck and ended when it was reloaded and ready to move to the next place. The following is a sample set of readings.

No 2. East Stub Entry Off No. 9 South

Operation:	Time	Watt Hr.
Unloading.....	2.00	0
Finished.....	2.06	150
Sumping.....	2.06	150
Finished.....	2.11	600
Cutting Face.....	2.15	600
Finished.....	2.35	4,300
Loading.....	2.50	4,500

Measurements and Readings:

Average voltage.....	225
Average depth cuts.....	73 in.
Width of room.....	24 ft.
Watts per sq. ft.....	30.8

"Unloading" includes moving the machine into place against the right rib.

"The table . . . is a summary of the six reports covering the daily performance of the machine.

"The machine had 24 loaders assigned to it, and 'kept up the cutting' for all these men."

Summary of Readings

Date	No. of Pieces	Face Feet	Average Depth, Inches	Square Feet	Actual Cutting Time, Hr. Min.	Total Watt Hours	Wts. per Sq. Ft.	Average Voltage
Dec. 13, 1911...	6	168' 10"	70.0	984.3	6 17	43,475	44.1	191
Dec. 16, 1911...	6	186' 8"	72.1	1121.5	6 26	48,575	43.3	193
Dec. 19, 1911...	5	145' 2"	72.1	872.2	5 16	35,600	40.8	191
Jan. 4, 1912. ...	5	149' 5"	73.1	910.1	6 22	38,800	42.6	194
Jan. 5, 1912. ...	5	168' 9"	72.8	1023.7	5 42	45,600	44.5	185
Jan. 8, 1912. ...	5	181' 11"	70.2	1064.2	6 52	51,900	48.7	193
Total.....	32	1,000' 9"	71.7	5976.0	36 45	263,950	44.0	191.1

Table of Totals and Averages

Machine	Days	Hours Cutting	Places Cut	Face Feet	Average Depth, Inches	Total Sq. Ft. Cut	Total Tons Mined	Total Watt Hours	Average Voltage
Sullivan...	6	36.45	32	1000.75	71.7	5,976	1770.66	263,950	191.1

Table of Rates

Machine	Face, Feet per Hour	Face Feet per Day	Sq. Ft. per Hour	Sq. Ft. per Day	Tons per Day	Watt Hours	
						Per Sq. Ft.	Per Ton
Sullivan..	27.2	166.79	162.6	962.7	295.11	44.0	147.37

The writer is indebted for information used in this article to the United States Coal Reports, issued by E. W. Parker, Statistician, U. S. Geological Survey; and to George W. Blackinton, University Club, Denver, Colo.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th St, New York, N. Y.

Recent Developments in the Design of Jeffrey Electric Locomotives and Coal-Cutting Machines

BY SANFORD B. BELDEN, COLUMBUS, OHIO.

(New York Meeting, February, 1914)

My topic, Recent Developments in the Design of Electric Mine Locomotives and Mining Machinery, does not require me to go into a general review of electricity as applied to the mining industry. Interesting as that subject always is, what I might say in regard to it, by way of introduction, would be unnecessary use of your time, since such facts are familiar to all.

First, we will consider the recent development of the electric mine locomotive. We may well pause for a moment to reflect upon its importance to the coal-mining industry of to-day. With the dawn of the electrical age its advent was inevitable. Our practical interest is aroused not so much by the first crude effort as by the great efficiency attained during the 30 or more years since the electric locomotive made its first appearance.

To judge accurately what it means to coal mining, and through coal mining to the industrial field at large, we would have to know how many of these locomotives are in use throughout the country and their aggregate daily performance, and then compare the results with what could be expected with other forms of power. Such data are lacking, but it requires no statistician to make it evident that the electric mining machine is a tremendous factor in modern progress, and that without it the work of the world could never have reached its present magnitude.

Not only has the electric mine locomotive served its own special branch of usefulness, but incidentally it has solved in the main the whole problem of electric transportation. It is but a step from the triumph of a 25-ton locomotive leading a long train of loaded coal cars from a mine to the electrification of common carriers. We may or may not be near any great change in this respect. Economical reasons may hold it back for an indefinite period. But the fact remains that in the development of our mine locomotives we have met this larger emergency, in mechanical type and efficiency, whenever it may come.

APPLICATION OF PROPER LOCOMOTIVE SPEED

One of the defects of the electric locomotive that early became apparent was that of excessive speed. Both manufacturers and the trade had to learn by experience the proper ratio of speed to horse-power. Motor equipment that gave from 9 to 10 miles an hour, not only meant unnecessary power, but acted as a detriment. Motormen were compelled to use the resistance in the locomotive to control the speed of the trip. This caused a great deal of resistance trouble. While the lesson has been generally learned, still there are cases in which the trade demands a higher speed than is necessary.

Through the joint efforts of practical mining men and manufacturers, motors were designed for the special service required. Improvement in the adjustment of speed to horse-power has gone on steadily, until now former troubles in this respect are mostly a memory.

DETERMINING HORSE-POWER CAPACITY

We are able by practical test to determine the horse-power capacity that should be used on a given weight of locomotive. The fact has been established that a locomotive for ordinary haulage and gathering work should have not less than 10 h. p. per ton, with a locomotive speed of 6 miles per hour. In the large haulage locomotives, on which depends the output of a mine, operating over hauls which are long and where the duty is more or less continuous, there should not be less than 12 h. p. per ton, at 6 miles per hour, and from 14 to 15 h. p. where operating at speeds from 8 to 10 miles per hour.

Average mining conditions throughout the country show that a speed of 6 miles per hour is a practical speed. There are a few rare cases where a speed of from 8 to 9 miles per hour is permissible, but only where the haul is straight and long, and where the track is laid with heavy rails well ballasted.

The technical points to be kept in mind when selecting a locomotive for given conditions are few. The complicated data of some engineers with regard to time, temperature, and the like, are confusing rather than helpful. The essentials briefly are as follows:

The draw-bar pull depends upon the weight of the locomotive. A locomotive equipped with steel-tired wheels, which are now becoming standard, and with proper motor capacity, will develop a draw-bar pull of 500 lb. to the ton.

On grades the effective draw-bar pull is reduced by the amount necessary to haul the locomotive up that grade. This is equal to 4 per cent. of the rated draw-bar pull of the locomotive for each per cent. of grade. As you

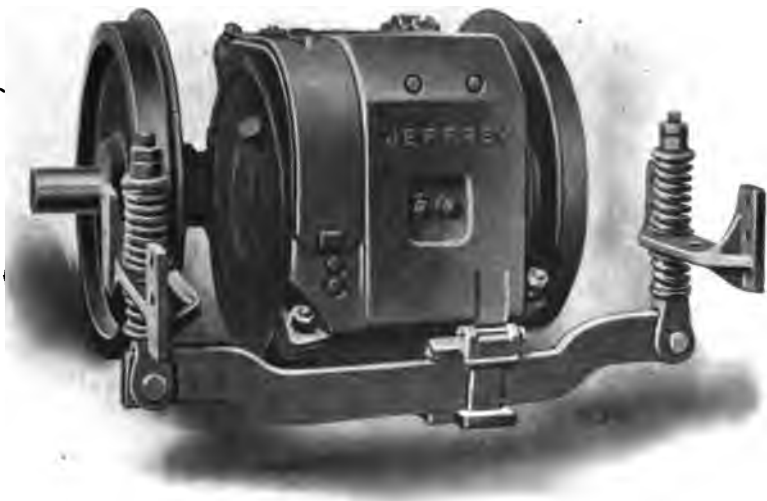


FIG. 1.—THREE-POINT MOTOR SUSPENSION USED ON LOCOMOTIVES.



FIG. 2.—40-H. P. BALL-BEARING MOTOR, SHOWING THE BALL BEARING IN PLACE ON THE ARMATURE SHAFT

understand, the effective draw-bar pull on a locomotive of any given weight is found by taking into account the per cent. of grade to be overcome. It is not necessary here to go further into figures. The same is true in regard to train loads.

When the grade to be negotiated is short, and the trip is up to speed while passing this grade, smaller locomotives can be specified. The size to be selected will be a matter of judgment on the part of some one familiar with mining conditions.

HORSE-POWER RATINGS OF MOTORS

The motors used for mine locomotive service are rated at 75° rise on all parts, including the commutator, after a continuous run at full load of 1 hr. A motor rated at 50 h. p. output should carry this load continuously for 1 hr. with a temperature rise on no part, including the commutator, of over 75° C. The reason that the commutator should be included in the 75° rise, and particularly for mine work, is to insure the purchaser that the commutator surface has not been sacrificed in order to build the motor for a narrow gauge.

We now come to the task of dissecting the electric locomotive, for the purpose of viewing separately some of the more vital improvements that have taken place in its construction. In this I desire to speak to the eye, as well as the ear. Therefore, use is made of a number of illustrations referred to in their order.

In the illustration designated as Fig. 1, the matter of motor spring suspension is emphasized. This construction greatly relieves mechanical vibrations due to rough track and the reversing of locomotive. It also reduces deterioration of the insulation on field and armature coils. There is a three-point suspension, as you will observe—two points on the axle and one on the motor-suspension bar. This allows each set of wheels and axles to readily follow the roughest track.

In Fig. 2, your attention is called to high perfection of detail with respect to the motor frame and bearings. You will see that the frames are split on the diagonal. The top half can thus be removed without disturbing any other part. A number of advantages are thus gained. The motor can be thoroughly inspected without removing the armature. Any excess of grease which might get into the split of the frame will flow out of and not into the motor casing. The armature pinion may be moved away from the gear when taking out the armature, making the equipment much more accessible than motor frames split through the axle. In the latter case the armature pinion must be rolled on the gear when moving the armature. Separate axle caps are provided, and the entire motor can be removed without taking out the wheels and axles.

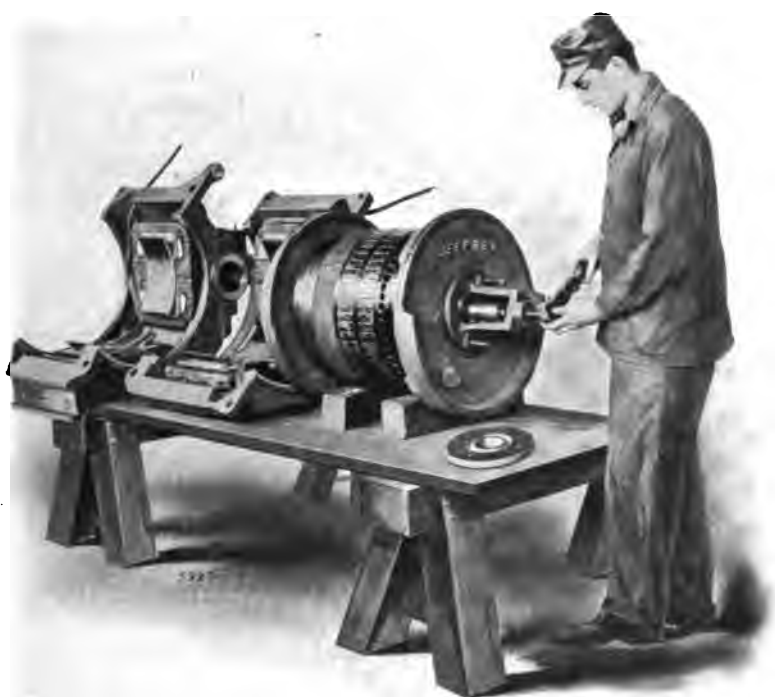


FIG. 3.—METHOD OF REMOVING BALL BEARINGS.

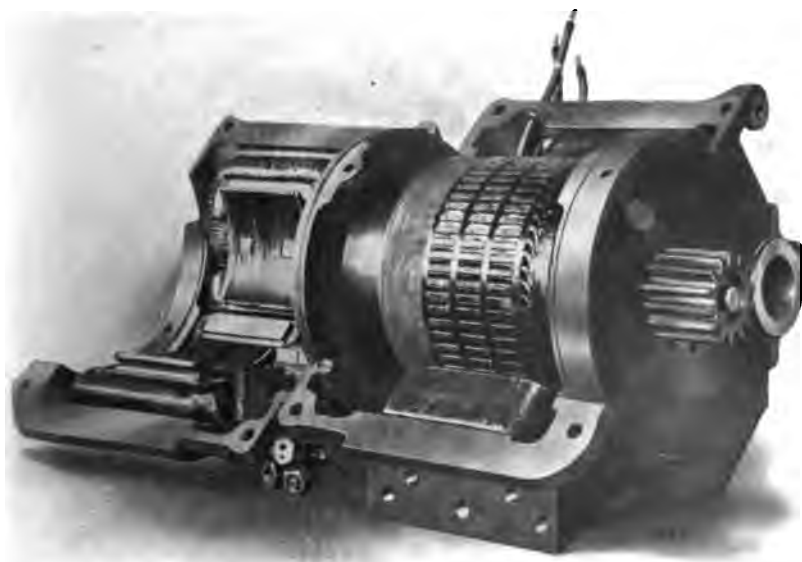


FIG. 4.—A 115-H. P. BALL-BEARING INTERPOLE MOTOR.

BALL BEARINGS APPLIED TO RAILWAY MOTORS

Of special importance is the application, as here shown, of ball bearings to railway motors. They have eliminated 90 per cent. of the ordinary mechanical troubles experienced by the use of bushed bearings. With ball bearings the armature is never on the pole piece. It has been my experience that a large per cent. of armature troubles, where bushed bearings are used, is due to the fact that, unintentionally, the bearings are allowed to wear out. After the armature rubs on the pole pieces the armature bands may be torn off and forced into the insulation of the armature coils, making repairs necessary.

But the expense in the loss of output is a more important item to the operator than that of repairs. Ball bearings allow of a more continuous operation throughout the year. Elimination of excessive grease from the inside of the motor casing reduces the deterioration of the insulation from that source. This is especially true on the commutator end, where the expense and annoyance from this cause have been the greatest. The armature revolves in a more uniform air gap during the life of the motor. It gives more perfect commutation, a better surface to the commutator, and decreases the movement of the brushes in the brush-holders, reducing the wear on the holders and giving the brushes a longer life.

The inner race of the ball bearing has a light driving fit on the armature shaft. Thus the shaft never wears out. This race is held in place by a large lock nut on the shaft, which, in turn, is secured by a spring ring fitting into a hole drilled through the lock nut into the shaft. The outer race is an oil fit in the bearing housing, and is permitted to align itself with the race inside.

Fig. 3 shows the manner in which the ball bearings may be removed. A special jack, similar to the old principle of a pinion puller, is screwed into the bearing housing, which together with the ball bearing, is pulled from the armature shaft.

INTERPOLES ON RAILWAY MOTORS

The demand for interpole motors on mine locomotives has been caused by their general use on stationary motors and interurban railway cars. The operators, however, will do well to consider the small benefit and, in most cases, the extra trouble which will result from their use.

A mine locomotive has a specified weight and it is possible to place on such a locomotive non-interpole motors that will slip the drives before normal full load is reached; therefore no overloading of the motors is possible. Non-interpole motors which have perfect commutation and large



FIG. 5.—13-TON ARMORPLATE TYPE LOCOMOTIVE.

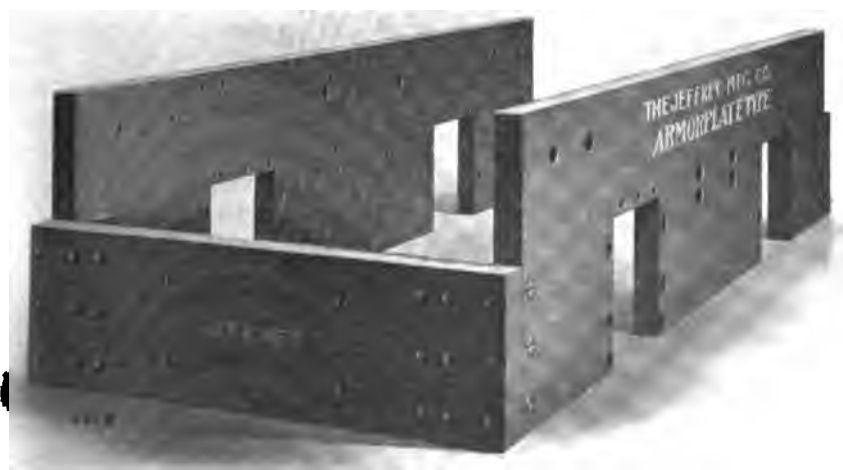


FIG. 6.—ARMORPLATE TYPE OF FRAME.

capacity can be obtained without the complication of auxiliary windings and their connections.

Auxiliary windings in a motor carcass of given dimensions do not increase the rated horse power due to the increased internal losses. Nevertheless the demand for interpole motors has been met by most manufacturers of mine locomotives, but with various results. To build a good interpole motor in the limiting dimensions met with in the design of mine locomotives, it is necessary to make as good a non-interpole motor as possible, and to this design add the auxiliary windings and interpoles. This method of procedure makes the armatures of both the non-interpole and the interpole motors absolutely interchangeable.

Fig. 4 shows the general construction of an interpole motor with three of the interpoles mounted on the bottom half of the motor frame.

ACCESSIBILITY OF MECHANISM

We have next a group of several pictures pertaining to locomotive frames. Until quite recently it was standard practice to use cast iron for this feature of construction. The breaking of castings from head-on collisions, and from leaving the rails and striking the rib, has brought about a change. Locomotives are now designed to use structural frames on sizes from 3 tons up to and including 6 tons. For the 7-ton size up to and including 20 tons, a solid rolled steel plate is used. This particular frame—shown in Fig. 5—is known as the “Armorplate” type. It is a slab which is in regular production by all the large steel mills, and can be secured in any reasonable length, with a width up to 40 in., and any thickness up to $6\frac{1}{2}$ in.

A good idea of the resisting power of this type of frame is given by Fig. 6. It is a photograph of steel slab framework to be used in a 13-ton locomotive, in process of production. With material of this kind it is unnecessary for manufacturers to give with the frame a guarantee against breakage. The “Armorplate” type permits for a given weight of locomotive the largest possible motor capacity for a given gauge. It also gives greater accessibility of parts.

Fig. 7 shows the top view of a 13-ton locomotive. It will be noted that the top half of each motor can be lifted out, if desired, without disturbing even the top half of the gear case. Neither the motor suspension nor any of the brake hangers is disturbed. The grid resistance is accessible without disturbing any other parts. The wheels and axles may be removed without disturbing the motor suspension. As indicated by Fig. 8, the journal boxes can be removed from the outside without taking out the wheels and axles.

In Fig. 9 we see how the wheels and axles may be removed from an



FIG. 7.—TOP VIEW OF LOCOMOTIVE, SHOWING MOTORS, CONTROLLETCR, ETC.



FIG. 8.—METHOD OF REMOVING JOURNAL BOX.



Fig. 10.—6-Ton Standard Cable-Reel Locomotive.

outside wheel type locomotive without disturbing the motor suspension. The locomotive is run over the pit and jacked up a sufficient height to remove the rails, allowing the wheels and axles to drop down into the pit. The motor is held in place by either a snatch or timber block, as shown in the illustration. The tie plate across the bottom of the pedestal is removed, together with the bottom half of the journal box. The journal springs are then removed and the top half of the journal box shoved up into the pedestal as high as possible. When in this position the motor frame allows the wheels and axles to be moved out of the axle linings. They can thus be removed without disturbing the motor suspension.

Having considered the general design of a mine locomotive, and the essential elements entering into it, let us turn our attention to what is necessary to haul the coal from the working places to the bottom of the shaft or to the tippie, outside.

Gathering locomotives collect the cars from the working places, and haul them to the side tracks in the mine, where they are picked up by haulage locomotives and taken to the bottom of the shaft or to the tippie. Five different types of gathering locomotives must be taken into account—the cable-reel type, the crab type, the combination crab and cable-reel type, the plain storage-battery locomotive, and the combination battery and trolley locomotive.

GATHERING LOCOMOTIVES

Fig. 10 shows a standard cable-reel locomotive, 6-ton weight, hauling two loaded cars out of a working place. This locomotive has a structural steel frame, with a wheel base of 40 in., a height of 30 in. over frame, and a length of 12 ft. over bumpers. It is equipped with two 30 h. p. motors, making a total of 60 h. p., and has a speed of 6 miles per hour. Locomotives so equipped can be coupled to as many mine cars as the mine boss desires. He can injure nothing but the wheels, which slip, and he soon learns the proper number of cars that allows the locomotive to do its work.

This locomotive is equipped on the front end with either a mechanically driven or a motor-driven cable reel. The reel holds 350 ft. of duplex or 500 ft. of single cable. The cable is automatically paid out as the locomotive runs into the working place, and automatically wound up as the locomotive comes out of the working place. The cable is wound in even layers on the drum by a spooling device and cable guide. Locomotives of this type have proven very efficient, and have gathered as high as from 120 to 180 cars per day, depending upon the conditions, and upon whether or not they must enter each working place for loaded cars.

Fig. 11 shows the standard cable-reel device, motor driven. The



FIG. 9.—METHOD OF REMOVING WHEELS AND AXLES FROM AN OUTSIDE WHEEL TYPE OF LOCOMOTIVE.

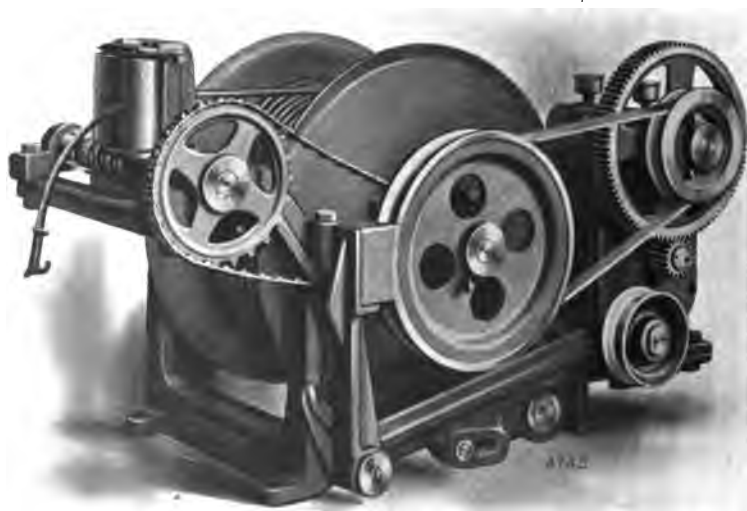


FIG. 11.—STANDARD CABLE-REEL DEVICE, MOTOR DRIVEN.



FIG. 12.—5-TON ELECTRIC CRAB LOCOMOTIVE PULLING A LOADED CAR FROM A ROOM.

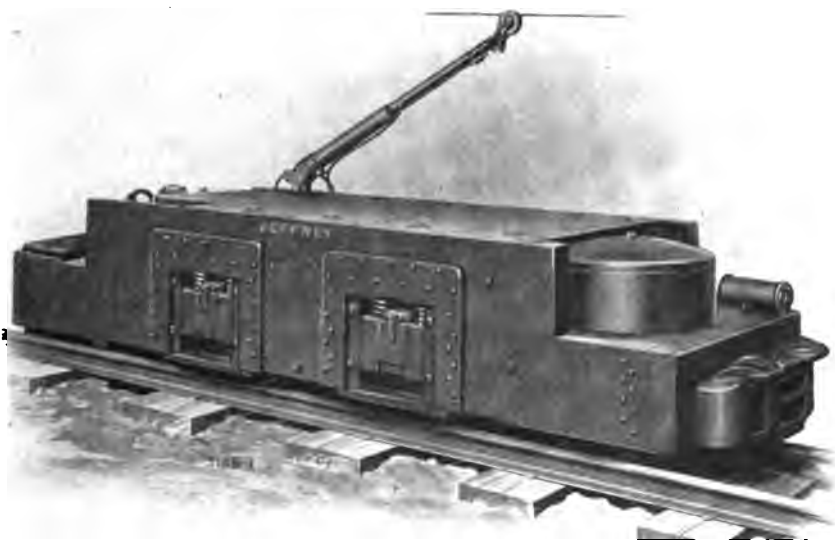


FIG. 13.—6-TON ELECTRIC CRAB LOCOMOTIVE; SHOWING A SMALL ELECTRIC HOIST ON FRONT END OF LOCOMOTIVE.

motor is thrown directly on the line when the belt is slack. A lever within convenient reach of the motorman operates a belt idler with which the tension in the belt can be adjusted. The tension in the belt determines the tension in the cable. Consequently, the latter is absolutely under the control of the motorman. The independent motor drives the reel, winding the cable faster than the locomotive travels, so that there is always a little slip in the belt drive. This keeps a constant tension in the cable while winding. A brake shoe is applied to the reel when unwinding, to keep from over-running. This device is simple and practical, and has eliminated disk clutches and all adjustments necessary to their operation.

Fig. 12 shows a crab locomotive hauling a car out of the room. The equipment consists of a small electric hoist mounted on the front end of the standard locomotive, as shown in Fig. 13. The motor on this device is rated at 8 h. p., and the device has a rated pull on the rope of 850 lb. at a rope speed of 175 ft. per minute. The drum holds 350 ft. of $\frac{3}{8}$ -in. steel rope. The gearing consists of a worm on the armature shaft and a worm wheel on the intermediate shaft, a spur pinion on the intermediate shaft meshing with a large spur gear above the drum. Between the gear and the drum is a disk clutch, as seen in Fig. 14.

To operate the crab device, the motor is thrown directly on the line without load. By means of a cam lever under the drum, the latter is raised, compressing the disk clutch between the drum and the gear, transmitting power to the drum and winding up the rope. The disk clutch acts as a safety slip should a car get off the track, or the steel rope become fastened in a tie. It is not possible to injure the gearing or overload the motor, as the disk clutch will slip. The drum is mounted upon ball bearings, making it easier to pull off the rope.

The operation of the crab locomotive is as follows: The motorman moves the locomotive on the entry in line with the room track. The trip rider takes the cable into the room and hooks it to the car, and signals the motorman to start the crab device and haul the car out of the room. Just as the car is rounding the curve out of the room the motorman starts the locomotive ahead, and about the time that the car passes the latches it is up against the bumper on the locomotive. The car is either left standing or backed up to make up a trip. This device is particularly useful where rooms run to the dip, or where the cars are light and the room track level.

Fig. 15 shows a combination crab and cable reel. It consists of a standard crab device just described, in connection with which there is a drum containing electric cable resting on a flange bolted to the large gear above the crab drum. The friction between the electric-cable drum and



FIG. 14.—STANDARD ELECTRIC CRAB DEVICE.

the flange on which it rests is adjustable by spring pressure from above. The one motor drives both devices.

STORAGE-BATTERY LOCOMOTIVES

There has been a great popular demand for the application of the storage-battery locomotive to mine

service. In order to meet this demand special designs, as shown in Fig. 16, were brought out, equipped with Edison or Exide storage batteries. For the ordinary mine locomotive the wheel base is 36 in., the over-all is 42 in., and the width is 14 in. wider than the gauge. The

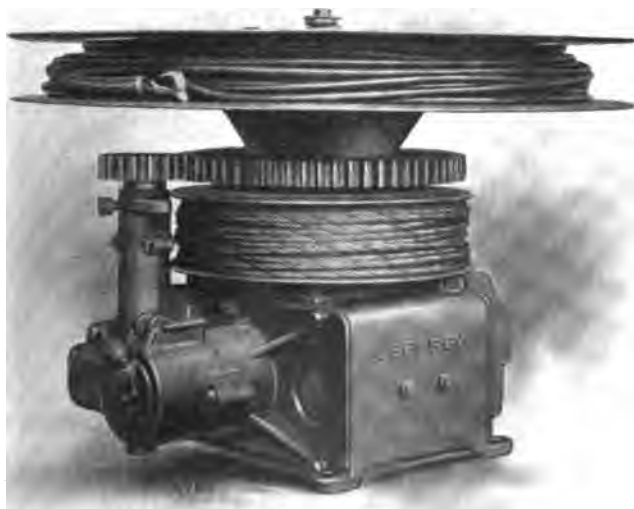


FIG. 15.—STANDARD COMBINATION CRAB AND CABLE-REEL DEVICE.

over-all length including the bumpers is 11 ft. The locomotive can be built in sizes from 3 tons up to 6 tons. On account of the fact that the storage battery is used it was necessary to design a special battery motor, shown in Fig. 17. This is designed for a minimum gauge of 18 in., with armature lengthwise of the locomotive. The design permits of a large copper capacity, reducing the internal losses of the motor to a minimum, and provides a sufficiently large capacity motor for a 6-ton battery locomotive. It also makes a very accessible equipment.

The motor is connected to the worm drive by a jaw clutch, and it is only necessary to remove four bolts from the motor frame and lift the motor out of the locomotive, without disturbing any of the worm gearing or its case. The worm and worm wheel used are of special design, having a very low internal loss, and to permit of coasting.

The work that an ordinary storage-battery locomotive can do depends upon conditions. In order to meet the conditions successfully, it is almost necessary in each case to have an engineer go over the ground and calculate the battery equipment. The greatest trouble experienced with the application of storage batteries to gathering work is to secure sufficient battery capacity in the limited space on a mine locomotive.

For estimating purposes a good value for the kilowatt-hours per ton-mile, including all losses in the battery and locomotive, is 0.125 on level track. It must also be remembered that the speed at which a battery locomotive must be kept is within the limits of $3\frac{1}{2}$ to 4 miles per hour.

The fact that the battery locomotive with sufficient capacity to do gathering work and haul the trips on the entry over heavy grades would have over-all dimensions not permissible in the average coal mine, caused the combination battery and trolley locomotive to be designed. This locomotive uses the battery only when going into the working places, but takes power from the trolley on the entry. The battery is charged from a low-voltage winding while the locomotive is operating on the trolley wire, and the low-voltage winding is used in connection with the battery when going into the working places. This locomotive, shown in Fig. 18, has a wheel base of 40 in., an over-all height of 35 in., and a length over the bumpers of 11 ft. The battery is located inside the frame and consists of 60A Edison cells or 48 Exide cells. A single motor is used, provided with a trolley winding and a low-voltage winding or battery winding. A fan is provided on the end of the armature shaft for the ventilation of the motor and the battery, as shown in Fig. 19. The armature with its ball bearings is shown in Fig. 20. The charge in the battery is continuously maintained, and the battery is ever ready for service.

Protecting devices are provided to keep the battery from short circuiting through armature winding in case the trolley circuit opens.



FIG. 16.—3-TON STORAGE-BATTERY LOCOMOTIVE.



FIG. 17.—BALL-BEARING MOTOR WITH TOP HALF OF GEAR CASE REMOVED—
USED FOR STORAGE BATTERY WORK.



FIG. 18.—COMBINATION TROLLEY AND STORAGE-BATTERY GATHERING LOCOMOTIVE.

The mechanical construction of this locomotive embodies nothing that has not already proven practical in mine service.

On the point of leaving the subject of electric locomotives and taking up that of cutting machines, mention should be made of a highly important factor in the development of mine equipment which so far I have already touched upon. It applies both to haulage and to the work of cutting the coal. It will have a growing value to manufacturers and to the trade in promoting economy and efficiency.

STANDARDIZATION OF MINING EQUIPMENTS

There is a word which we hear frequently mentioned these days in connection with all lines of endeavor. That word is standardization. There is a progressive tendency to sift out non-essentials, and to reduce the confusion of variation to standard types or systems. This is made possible largely through adjustment. To illustrate, when a device or a machine which satisfactorily performs a given service fails in a few particulars to meet the needs of some other condition of the same general nature, the problem is solved by the substituting of parts, rather than by producing an entirely new machine.

From the standpoint of progress the advantage to the manufacturer is obvious. He can concentrate his energies on the perfecting of really essential equipment instead of dissipating them over an unlimited range of departure from fundamental type. It is only fair to the manufacturer to state that he has kept in advance of the trade in this respect, seeking first refinement instead of the perhaps larger material rewards that might come to him through the multiplying of new apparatus.

To the coal producer the economical value of standardization is without question. It means less original outlay and a saving of time and expense in upkeep and operation. The confusion arising from lack of an elastic uniformity has been often experienced in the case of consolidations, where different mines of former separate ownership came under one management. No intelligent estimate can be made of the waste resulting from the purchase of special equipment for supposed distinctive needs. In the great majority of such cases slight modification of standard would have served the purpose, the change not being of so important a nature as to forbid easy restoration to normal performance, if at an time this should be desired.

COAL-CUTTING MACHINES

One of the first coal-cutting machines was known as the "cutter-bar" type. In Fig. 21 we see that the principal part was a horizontal bar

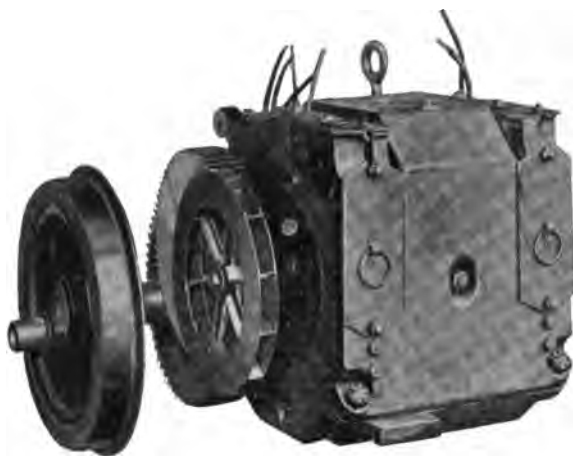


FIG. 19.—MOTOR USED ON COMBINATION TROLLEY AND STORAGE-BATTERY LOCOMOTIVE.

at the front end of the machine. Bits were held in this cutter bar by means of set screws, as seen in the illustration. The defect in this design was that each bit had to cut all of the coal which it encountered. This made the work of each so severe that it was impossible to maintain bits, especially where impurities were encountered in the coal. Further, the reduction

in the shaft at the two places shown caused no end of trouble, due to breaking where the diameter was reduced. The cutter bar, held in a moving frame on which was mounted the motor and gearing, moved forward in a stationary or outside frame.

Following the cutter-bar machine was the chain breast machine with which we are so familiar. We are so interested in the machines which are doing the important work to-day and our time and space so limited that history of the development must wait. The shortwall or continuous cutting machine has been developed so rapidly in the past few years that a description of its most important features will not be out of place.

SHORTWALL MINING MACHINES

Fig. 22 shows a shortwall machine on a truck ready to be taken to a working place. It is provided with an automatic cable reel for paying out and reeling up the cable, eliminating dragging of the cable when going from one working place to another. A cable drum on the front end, power driven, has wound on it 50 ft. of $\frac{3}{4}$ -in. steel rope. It is this cable which is



FIG. 20.—ARMATURE WITH ITS BALL BEARINGS, HAVING TWO ARMATURE WINDINGS AND TWO COMMUTATORS —USED IN MOTOR SHOWN IN FIG. 19.



FIG. 21.—CUTTER BAR USED ON THE FIRST COAL-CUTTING MACHINES.

used to pull the machine off the truck into the proper working place and back on the truck. There are two speeds provided in the gearing, a handling speed and a feed speed. The handling speed averages about 12 ft. per minute. The feed speed is determined by the nature of the coal to be cut, and averages from 15 to 30 in. per minute.

On the rear end of this machine is a cable drum equipped with a $\frac{1}{2}$ -in. steel cable and a band brake. This band brake is used in order to hold the machine to the proper angle to the coal face, so as to keep the cutter bar under the coal. The gearing of this particular shortwall machine is shown in Fig. 23, with the motor removed, and it shows a very accessible machine.

All the bearings are cast integral with the steel bed plate, which is necessary on this class of machinery, in order to obtain perfect alignment of the gears and shafting. The cutter bar is made up of cast or structural steel. The particular one shown is cast steel. Later designs are made of structural steel with tool-steel wearing strips on all wearing surfaces, so that all wearing parts can be replaced without renewing the entire cutter head.

The machine is taken to the working place on a self-propelling truck, as shown in Fig. 24, with the operator riding on the truck. On the self-propelling truck is provided a disk clutch which regulates the speed of the truck, and gives a variation in speed from zero up to $3\frac{1}{2}$ miles per hour, without using the starting box. This gives a fast-moving truck where the track is straight, and a low speed where the track is bad, or



FIG. 22.—SHORTWALL MACHINE ON TRUCK, PROVIDED WITH AN AUTOMATIC CABLE REEL.



FIG. 23.—INTERIOR OF SHORTWALL MACHINE, SHOWING GEARING WITH MOTOR REMOVED.

where there are frogs or switches to cross. The fast-moving truck saves time in moving from place to place.

The machine is unloaded from the truck, as shown in Fig. 25, by taking the steel cable from the front drum and fastening it to a jack at the coal face. The power is turned on and the machine pulls itself from the truck. The jack is then moved to the right-hand corner of the room, and the steel cable is brought around on suitable sheaves back of the machine, swinging the machine in position, as shown in Fig. 26. Without moving the jacks, the machine is pulled into the corner and, with a jack set in the slot in the cutter head, the cutter bar is forced its complete depth under the coal. After this sumping cut is made the steel cable is fastened to a jack at the left-hand rib and a cut across the face is made. After the cut



FIG. 24.—SHORTWALL MINING MACHINE ENTERING ROOM ON ITS OWN TRUCK.



FIG. 25.—UNLOADING SHORTWALL MACHINE FROM TRUCK.



FIG. 26.—BRINGING MACHINE TO FACE.

across the face is completed, as shown in Fig. 27, the steel cable is brought around the machine on suitable sheaves, and fastened to a jack, and where the props are set close to the face the machine is brought back with the cutter bar still projecting under the kerf, until opposite the truck. The cable is then fastened to a hook on the truck, and the machine loaded as shown in Fig. 28.

This particular machine has an over-all height of 24 in., and is equipped with a 35 h. p. motor, rated on the 1-hr. basis. The armature shaft is equipped with ball bearings, and the construction of the motor will permit of totally inclosing where necessary for gaseous mines.

While the shortwall machine was a step in the right direction, to eliminate manual labor, and the machines are proving very successful, and will continue to do so, there was some cutting to be done in various mines in the country which demanded still larger equipment.

TOP-CUTTING MACHINES

After the shortwall machine was developed the question naturally arose in the minds of the men at the mine, why could not the time of unloading the machine be saved? Why not leave the machine on its own truck, and in addition, if there is a dirt band in the seam, why not cut out this dirt band? This led to the development of the top-cutting machine, a shop view of which is shown in Fig. 29.

The construction of this machine is as follows: The motor with all its mechanism and the cutter bar is mounted on a circular steel casting supported by a ring on adjustable pedestals. The machine never leaves the truck. A top view of this machine is shown in Fig. 30. A ratchet feed driving a cable drum through a disk clutch winds up a $\frac{5}{8}$ -in. steel rope, one end of which is fastened to the stationary frame. The cutter bar, together with all gearing which is mounted on the circular casting, revolves in a complete circle.

Following is the operation of the machine: The machine is self-propelled into the working place until the truck strikes against the face. The cutter bar is swung around to the left for the beginning of the cut, as shown in Fig. 32. The ratchet feed is started, taking a large number of teeth, and giving a high rate of feed where the depth of cut is shallow. After the bar swings around under the coal, this feed can be reduced without stopping the machine, and again increased until the complete cut is finished, as shown in Fig. 33. In this particular photograph it will be noted that after the dirt band was cut out the coal separated from the roof, giving an impression that the kerf was not cut. The special advantage of this machine is the time and labor saved in not



FIG. 27.—HANDLING MACHINE IN LIMITED SPACE.



FIG. 28.—METHOD OF LOADING UPON TRUCK.

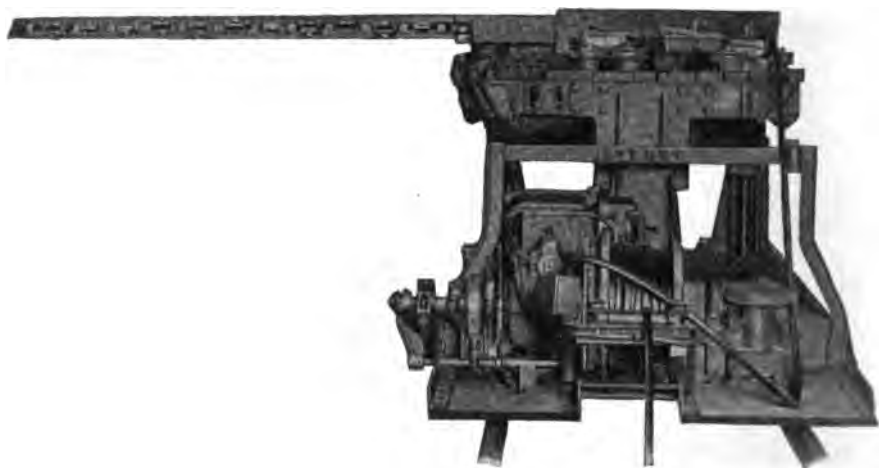


FIG. 29.—“ARCWALL” TOP-CUTTING MACHINE—POSITION FOR CUTTING.

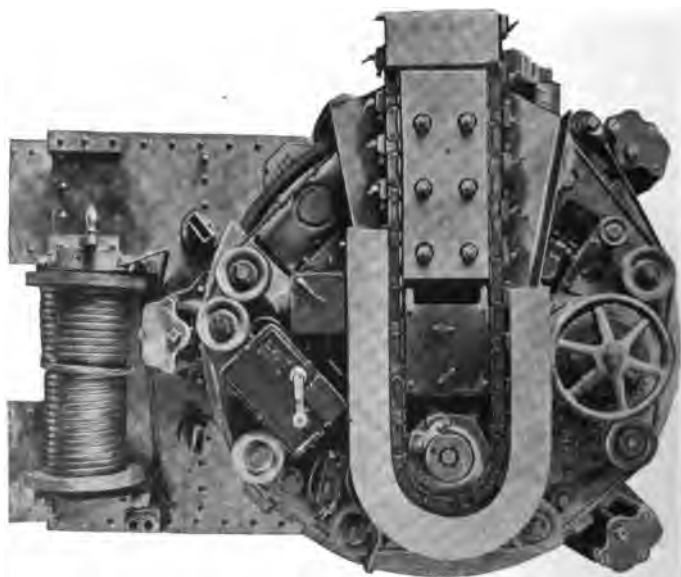


FIG. 30.—TOP VIEW OF ABOVE MACHINE.

unloading from the truck, and the fact that the cuttings do not have to be handled except where a band of dirt is cut and must be gobbed before the coal is shot. It gives an opportunity of obtaining clean coal where dirt bands can be cut and gobbed. It is a very rapid machine, either for entry driving or for room work. It is suitable for cutting rooms up to 20 ft. in width, and can be built for a minimum height of 36 in. above the rail, and has cut 20-ft. rooms in 8 min. The cutter bar is raised and lowered by power. The self-propelling speed of this machine is controlled by disk friction clutches permitting high moving speed without the use of starting box or resistance, thus saving wear and tear on the electrical parts.

BREAST MACHINES ON TURNTABLE TRUCK

The design of this top cutting machine naturally led to the placing of breast machines on turntable trucks, as shown in Fig. 34. The machine here shown is cutting a break-through. The advantages offered by a machine cutting in the middle of the seam are as follows: They are especially suited for narrow work, no slack has to be shoveled, and they afford protection to tender roof due to the fact that small shots can be used to shoot the coal. There is no labor in handling the machine, and the coal can be cut at various angles by changing the angle of the machine on the truck.

Machines of this kind shown in Fig. 34 have cut as high as forty 12-ft. places in 10 hr. by the use of gathering locomotives to haul them from place to place rapidly. Self-propelling machines of this type can cut as high as 24 to 30 places. Machines of this type can also be inverted to cut a dirt band at the top of the seam, as shown in Fig. 35.

TURRET MINING MACHINES

Another important combination is the shortwall machine mounted on a turntable truck, as shown in Fig. 36. This machine is shown just starting a run. The cable from the feed drum is fastened to an anchor drilled into the face of the coal, and the machine is fed forward until the cutter bar projects under the coal the proper depth. Another anchor is placed in the left-hand rib and the feed cable attached, swinging the machine around through the proper angle, cutting a place of the desired width. See Fig. 37.

The machine is self-raising and lowering, and can be raised or lowered 3 ft. in 25 sec. It has cut 15-ft. rooms in 10 min. and has cut as high as 25 places in 10 hr. The machine on the truck is under the absolute control of the operator, and is geared to have a high speed on



FIG. 31.—SELF-PROPELLED COAL CUTTER GOING OUT OF ROOM.



FIG. 32.—COAL CUTTER STARTING "OVER-CUTTING" IN 20-FT. ROOM.



FIG. 33.—COAL CUTTER FINISHING "OVER-CUTTING" IN 20-FT. ROOM.



FIG. 34.—BREAST TYPE COAL CUTTER ON TURNTABLE TRUCK.

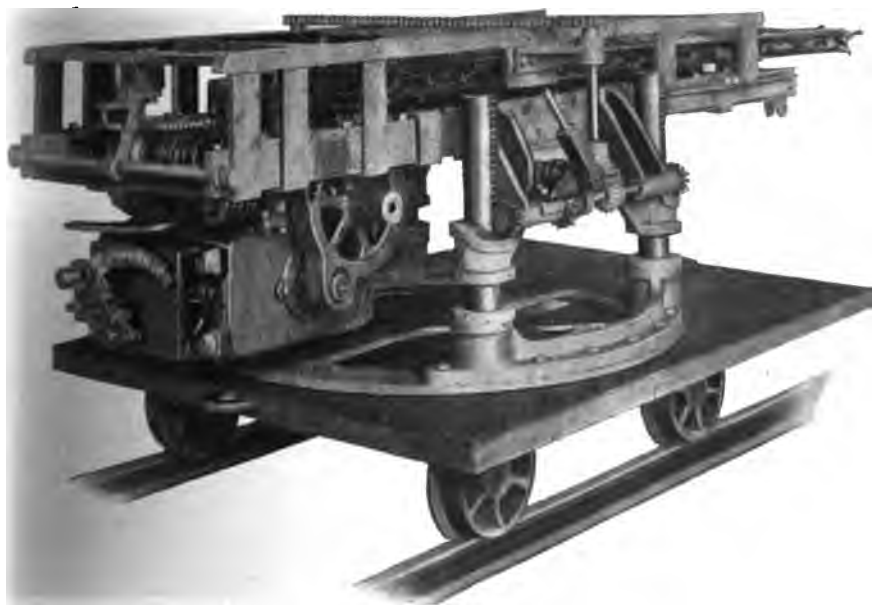


FIG. 35.—ABOVE MACHINE ADAPTED FOR TOP CUTTING.



FIG. 36.—SHORTWALL TURRET TYPE—MOUNTED ON TURNTABLE TRUCK.



FIG. 37.—MACHINE CUTTING A BREAK-THROUGH IN SHALE BAND.



FIG. 38.—“OVER-CUTTING” IN SHALE BAND.



FIG. 39.—TOP VIEW OF LONGWALL MACHINE.



FIG. 40.—LONGWALL MACHINE CUTTING ALONG THE FACE.

straight track and slow speed over switches and frogs. It is equipped with band brakes to control the speed down grades. In the particular illustrations used this machine is cutting a dirt band from the middle of the seam. This dirt is gobbed, and the coal shot down, producing clean coal.

LONGWALL MINING MACHINES

Fig. 39 is a top view of a longwall machine designed for longwall work in England and different parts of Europe. This machine is equipped with either direct-current or alternating-current motor. The power for the cutter chain is transmitted direct from the armature shaft through a worm and worm wheel. The machine is provided with a ratchet feed, so that the speed of feed can be varied by changing the number of notches with which the pawl engages. The crank operating the machine is driven through a worm and worm wheel. There are no exposed wires between the starting box and the motor, all wires being taken through metal conduits. This construction makes a very simple design and is noiseless in operation. The elimination of noise, as far as possible, is one of the essential elements of longwall work, so that machine operators can hear the working of the roof. Fig. 40 shows this machine cutting in a thin seam of coal.

The undercutting machines have been developed to such a point of perfection that coal operators and manufacturers have spent considerable time and energy in producing machinery which will reduce the cost of loading the coal after it is shot down.

To design a successful loading machine is not an easy task. The machine must have a wide range of action in order to load coal from various widths of working places and must also be compact and readily hauled from one working place to another.

MECHANICAL COAL LOADER

The design of a successful coal-loading machine can best be described by referring to the illustration. Fig. 41 shows a loading machine mounted on its truck ready to be hauled to a working place. It will be noted that a single conveyer is used.

After the loader is taken into the working place it is unloaded from the truck by its own power by means of a 1-in. rope and a winch on the side of the conveyer. The conveyer is pulled forward off of the truck, allowing the rear set of small wheels to come into contact with the rails and placing the support of the conveyer on a pivot supported by a steel casting which



FIG. 41.—MECHANICAL LOADING MACHINE ON TRUCK.

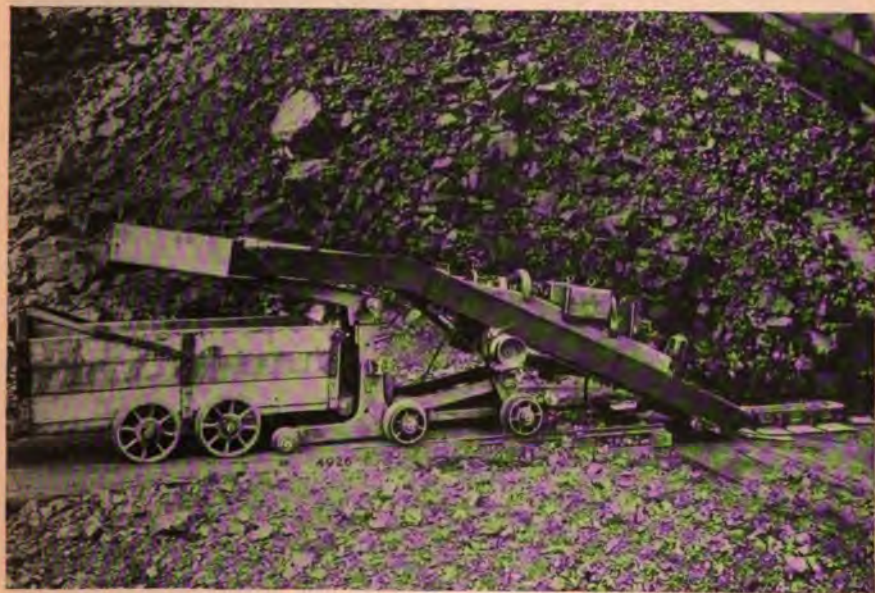


FIG. 42.—READY TO LOAD THE COAL.



FIG. 43.—PULLING LOADING MACHINE ON TRUCK WITH ITS OWN POWER.

locks into the main truck frame after the conveyer is pulled up into working position.

Fig. 42 shows the machine ready to load coal. The arrangement of the conveyer support permits the coal car to project under the conveyer, allowing the coal to be loaded into the middle of the car. The flights used on the conveyer are flat castings linked into an endless chain which is driven by an electric motor. Between the motor and the gearing is a disk friction clutch. The feed of the machine to the right and left is accomplished by means of power-driven drums on which is wound $\frac{3}{8}$ -in. steel rope. A disk friction clutch is also provided in the gearing of this feed mechanism. The disk friction clutches act as safety slips should the conveyer flights become fastened under the coal or strike on projections in the bottom.

When it is desired to load the machine the conveyer is pulled back onto the truck by its own power as shown in Fig. 43.

The capacity of this machine is 1 ton of coal per minute, but it has never been found necessary to rate it at a greater capacity than 1,000 lb. per minute.



FIG. 44.—MACHINE DRIVING ENTRY.



FIG. 45.—SHOWING POSITION OF LOADING END.

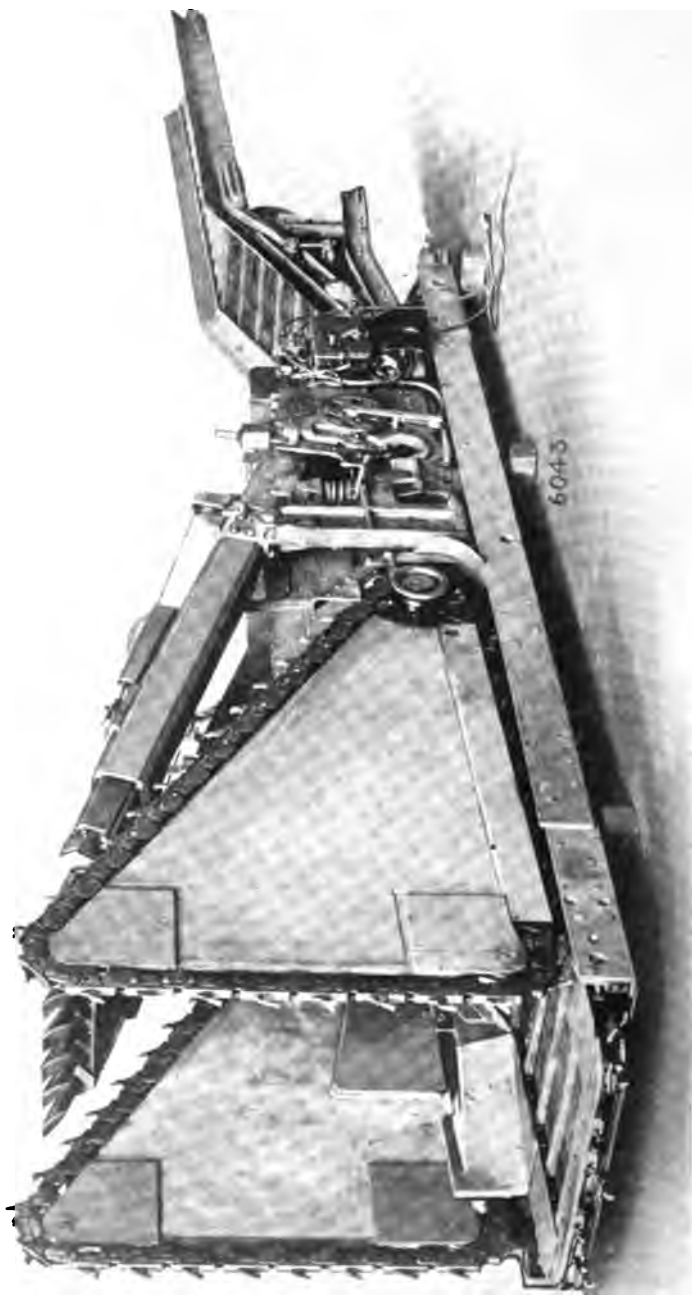


FIG. 46.—COMBINED CUTTING AND LOADING MACHINE.

COMBINED MINING AND LOADING MACHINE

The latest method of mining coal from the face is by the use of the cutting and loading machine, a view of which is shown in Fig. 46. The essential elements in this design are the stationary frame or pan in which the machine feeds itself forward or backward by its own power, the undercutting chain, similar to the breast type or cutter chain, two vertical shearing chains, the conveyer in the cutter head, the rear pivoted conveyer which loads the coal into the car and the mechanically operated pick which knocks down the coal after it is undercut and sheared. The machine is also provided with a slack conveyer.

The operation of the machine is similar to that of the breast machine. It is taken into a working place and never removed until the place is worked out. Entries, rooms, or break-throughs of any desired width can be cut with this machine. Fig. 44 shows the machine in operation driving an entry in Pocahontas coal. Fig. 45 shows the complete machine including the rear conveyer. The machine is entirely self-handling and moves into different positions by its own power.

Particular attention is called to the clean appearance of the working place. No attempt was made to clean up this place for photographing purposes. The capacity of this machine is estimated at 100 tons in 8 hr.

With this, I feel that I have gone over at enough length, however lamely, the ground embraced within my subject. Speculation as to what further improvements are to come would be idle. Credit for what has been accomplished belongs jointly to the practical mine man, always alert to the significance of anything growing out of his experience, and to the manufacturer, who is constantly striving after a higher degree of perfection. The two have worked together, though sometimes unconsciously. As it has been in the past, so it will be in the future. While standards have been reached from which, it would seem, there could be no marked departure, it is a safe conclusion that progress will always remain the guiding principle.

Valuation of Iron Mines

Further discussion of the paper of James R. Finlay, presented at the New York meeting, February, 1913 (Trans., xlv, 282), with special reference to the discussion by R. B. Brinsmade (Trans., xiv, 322.)

BY E. E. WHITE, ISHPERING, MICH.

I DISAGREE with Mr. Brinsmade on several points, and I will discuss his remarks under the same six topics which he has used.

1. *Rate of Interest Earned by Sinking-Fund.*—I believe Mr. Brinsmade is confused in his ideas as to the effect of new gold on the interest earned on money. The flood of gold which he mentions certainly should not operate to increase interest rates, but since it replaces and adds to liquid capital should reduce interest rates and counteract the destruction of liquid capital which he mentions. Furthermore, it is doubtful if the destruction of capital since 1897 by wars and catastrophes has been any greater in proportion than in similar periods before 1897. With regard to savings banks, I would say that only 3 per cent. is allowed on deposits through a large part of the West. As to bonds, they can be bought to yield any rate of interest desired up to 6 per cent., but a 5 per cent. bond is not as safe as a 4 per cent. bond, nor a 4 per cent. bond as safe as a 3 per cent. government security.

I do not understand why Mr. Brinsmade says that Mr. Denny's quoted opinion of a 3 per cent. rate is *evidently* based on English conditions before the Boer war. Certainly there is no reference to the Boer war or to English conditions in Mr. Denny's article. M. Howard Burnham, in his admirable work on mine valuation, says¹: "For instance, redemption is taken at 3 per cent., because even 4 per cent. represents a greater risk, as is indicated by the purchase of Government paper to bear only 3 per cent., the great desideratum of the latter being security." Again, Mr. Burnham says:² "The tables are calculated on the assumption that the dividends shall accumulate at 3 per cent. only, as in accordance with the general idea set out elsewhere, a rate greater than 3 per cent. involves appreciable risk with which it were inaccurate to saddle the sinking-fund if risks are to be segregated, this being the basis of all the writer's calculations."

Consols, the best-known government security in the world, are now (December, 1913) selling at about 72, yielding 3.48 per cent. on that price; French Rentes are selling to yield 3.5 per cent. and German 3 per cents to yield 3.95 per cent. Every one knows, however, that bonds and investment stocks all over the world are depressed at the present time, due to

¹ Burnham, M. Howard: *Modern Mine Valuation*, p. 19 (1912).

² *Idem*, p. 134.

straining of credit by the last few years of great prosperity, and the consequent scarcity of money. The usual return on Consols is lower than at present, and it has only averaged higher for one year in the last 75 years—in 1848, when it averaged 3.51 per cent. In 1897 it averaged 2.45 per cent., and in only five of the last 30 years has it averaged higher than 3 per cent. It therefore seems to me that Mr. Denny and Mr. Burnham are amply justified in assuming that 3 per cent. is the rate of interest that should be expected for a minimum risk.

2. *Rate of Interest Earned by Investment.*—I am not at all certain that Mr. Denny referred to gold mines with uncertain and scanty ore reserves in his statement of mining risks, as Mr. Brinsmade says. The Rand cannot be included in this class and I certainly believe it fair to assume that he included the Rand in his general statement of mining risk. A statement from Mr. Denny would be valuable in this connection, also from Mr. Hammond, whom I quoted in my discussion of Mr. Finlay's paper.

I disagree with Mr. Brinsmade in his comparison of iron mines and city real estate. As Mr. Brinsmade says, city real estate is speculative, but a decrease of 25 per cent. in the gross income only means a decrease of approximately 50 per cent. in the net income and therefore in the value, whereas in the case of most iron mines a 25 per cent. cut in ore prices would mean a decrease of 100 per cent. in the net income; that is, would wipe out all the profit and all value. For instance, assume taxes, repairs, maintenance, etc., to be 5 per cent. of the value of real estate, then the rent must be 10 per cent. to return 5 per cent. net on the investment. If the rent falls to $7\frac{1}{2}$ per cent. the net income would be $2\frac{1}{2}$ per cent. In the case of iron ore, however, the selling price is roughly \$4 per ton, and the profit rarely over \$1 per ton, usually much less. A decrease of 25 per cent. in the value per ton would therefore take all the profit. In other words, the cost of production is a so much larger proportion in the case of iron ore that the two cannot be fairly said to resemble each other. I believe the low rate of interest accepted by investors in real estate is partly due to the fact that they expect an unearned increment—i.e., an increase in the value of their property due to the growth of the city. Investors in real estate are willing to pay more, and value the property upon a 5 per cent. basis, for the same reason. There is little chance of an increase in the value of an iron mine, where the ore is already blocked out. There is almost no chance of a permanent increase in the selling price of the ore, so that any increase in the value of such a mine must be due to a decrease in the cost of production. The tendency of the latter is in the wrong direction at present.

Mr. Finlay applied his method and the same rate of interest to all Michigan mines, regardless of whether or not there was any speculative value due to possible increase in the ore reserves. Where mines are valued

on the assumption that the ore body continues for 100 ft. below the bottom level, or some other conservative arbitrary distance, there is a possibility of increase in the ore reserves and in the value, as in the case of real estate. Where the limits of the ore bodies are completely known, however, either by underground work or by diamond drilling, or where any possible extension of an ore body is cut off by the property lines, this speculative value does not exist, and the return on the investment should be correspondingly greater. By using the same rate of interest in valuing all mines by this method, an injustice is done mines which are fully developed, and particularly certain mines where the ore bodies are in flat-lying beds, and where the whole tonnage may therefore be calculated from the results of drilling. J. Parke Channing discusses this point in his remarks on "Mine Valuation." He says:³ "May it not be possible to have a mine with too much ore in sight? I ask this question because the general tenor of the discussion on mine valuation has made ore in sight the crucial test of the value of a property. If based on this test, many of our good iron mines would make a sorry showing, and yet their capacity for production is well known and year after year they come up to expectation." Speaking of certain pyrites mines, Mr. Channing goes on to say: "We have taken the ground that in mines of this character the deposits are so regular that the better test of their relative values is to be found in their capacity for production per 100 ft. of depth, rather than in the number of tons of ore available for immediate shipment; that is, in sight." In other words, Mr. Channing claims that mines should not necessarily be valued in proportion to their ore reserves, and that if they are, the mines with the most ore in sight are overvalued. This is the same as saying that if ore reserves are taken into account, as they must be, the valuations of mines with large ore reserves should be obtained by capitalizing the profits from the reserves at a higher rate of interest than in the case of steady producers with small ore reserves but considerable speculative value.

Another important difference between iron mines and city real estate is the amount of physical risk to the property. It must be admitted that the risk of fires, cave-ins, strikes, sand-runs, breakdowns of machinery, etc., in the case of mines, is greater than the risk of fires, floods, tornadoes, etc., in the case of city real estate. Furthermore, the cost of insurance against these risks in the case of real estate is comparatively small, and is included in the fixed charges. In the case of mines no insurance against the risks mentioned above is carried, as its cost would probably be prohibitive, and the damages difficult to ascertain.

Another condition that enhances the value of city real estate, and so affects the rate of interest, is the possibility of ready sale without material sacrifice. Iron mines are only sold at long intervals.

³ *Engineering and Mining Journal*, vol. lxxvi, No. 11, p. 383 (Sept. 12, 1903).

The differences between iron mines and real estate are, briefly, the greater stability of profits in the case of real estate, the lesser physical risk and possibility of covering it with insurance, the greater chance of an increase in the principal (as compared with most iron mines), and the possibility of ready sale. In view of these differences it certainly seems reasonable that the return upon an investment in iron mines should be considerably greater than upon an investment in real estate.

As regards stock-market valuations, Mr. Finlay realized that mines are much overvalued in the stock market and disregarded these valuations in his valuation of Michigan copper mines. This overvaluation is probably because stocks are a medium of speculation and most investors in mining stocks expect to make more from an increase of principal than from dividends. Because speculators are satisfied with 5 per cent. dividends from listed mining stocks is no reason that that is a fair return on money. They all expect to make more than that, whether they do or not. I would ask Mr. Brinsmade if he would invest in mining stocks expecting to net only 5 per cent. after charging off part of his dividends for a sinking fund, or if he values mining properties for his clients on a 5 per cent. basis.

Mr. Burnham says:⁴ "It is a source of constant wonder that people invest in copper or silver ventures on the same basis as gold—i.e., that they accept, say, 6 per cent. return on copper shares, and ask the same of gold. If such investors could bring themselves to abandon their attitude of self-deception for a moment they would no doubt admit that what they really want is share speculation, but in order to indulge themselves comfortably in such pastime, they seek to be told that it is investment."

In this paragraph Mr. Burnham had reference to the variable selling price of other metals than gold, and iron ore falls in the same class as copper and silver. I believe that 5 per cent. is altogether too low a return to make an investment in iron mining attractive, and therefore too low a rate to use as a basis of valuation.

It has been claimed that in a valuation such as Mr. Finlay's all risks are taken into account in the average cost of production. This might be true of certain hazards of the mining business, such as strikes, mine fires, cave-ins, sand-runs, etc., if they have happened during the period for which costs are averaged. They probably have not happened in such a short period, however, and there is always the risk that they may be worse in the future than in the past. But, aside from these risks, there are certain risks that cannot possibly be taken account of in the cost of production. These are the risk that the future average price of ore will be less than the average price assumed, the risk that the estimate of tonnage, often based largely on diamond drilling, is too great, and the risk that for one reason or another the annual production, depending upon the amount that can be sold

⁴ *Modern Mine Valuation*, p. 26.

annually, may not average as high in the future as in the past. These risks can only be taken into account by a rate of interest higher than the normal rate of 5 per cent., and necessitate a rate of at least 10 per cent.

3. *Place of Royalty Item in Valuation.*—From Mr. Brinsmade's discussion it is evident that he misread my paper. I had no idea of adding royalty to mining cost in reckoning valuation for taxation purposes and did not do so in my discussion of Mr. Finlay's figures. I considered royalty in comparing the cost of production of Lake Superior and Cuban ores, for reasons stated, but that is a very different thing.

If we were reckoning valuations from the standpoint of a purchaser it would nevertheless be entirely correct to add royalty to mining cost, and I therefore disagree with Mr. Brinsmade's last paragraph on this subject. Theoretically royalties may not affect the scale of prices, but in practice they do when an industry is firmly established on a royalty basis with long-term leases, as is the case in the Lake Superior district, or in the anthracite field in Pennsylvania. If conditions change so there is no profit to the operator on this royalty scale, I admit that the average royalty would tend to decrease, but it would be a slow process and would probably take at least 10 years to effect any considerable decrease in the royalty. A purchaser could not count on such a future reduction and must consider royalty as a mining cost. From the standpoint of the State, royalty is immaterial, however, since it makes no difference to the State whether the profit goes to the operator or to the fee owner in form of royalty. The valuation for taxation purposes is necessarily reached by including royalty as profit.

4. *Place of Taxation Item in Valuation.*—Mr. Brinsmade's suggestion is very good theoretically, but I do not believe it is practical. If the only taxes to be levied were State taxes, it might do, but with the complication of city, township, and county taxes brought in, different for almost every mine, it would be very difficult to apply. No values could be placed on the mines until the preliminary valuation of each city, township, and county and the total valuation of the State were determined and the amount of money to be raised for each. "t" would then have to be determined independently for each mine three times, for city, county, and State taxes, and the sum of the three t's used to figure the final valuation.

It seems to me simpler to apportion each mine its share of the taxes for the current year as nearly as can be estimated and divide by the average tonnage per year, adding the cost of taxes per ton to mining cost. The result should be the same as by Mr. Brinsmade's suggested method.

5. *Calculation of Future Ore Prices.*—Mr. Brinsmade treats Mr. Finlay's assumed future price of Lake Superior ore too lightly. To be sure, it is not so bad for a company to pay \$100,000 more than its fair share of taxes for one year as it would be to pay the extra \$100,000 for 10 years, but it is bad enough. Furthermore, it is just as bad to have the

erroneous assumption made every year for 10 years as it would be to have it made in the beginning to hold for 10 years. In Michigan, Mr. Finlay's valuations stood for two years, and when they were revised, I believe, the future price of ore was determined from the average of ore prices for the last five years instead of Mr. Finlay's period of seven years. If the revaluation were made every year and the same period used each year to determine the average price, I admit that the errors in determining the future price would balance in the course of time, but what security is there that the same period will be used each year, with changing tax commissions and different mining experts? After a period of low prices when mines have been valued high because of the high average price during the previous period it will be a great temptation to the tax commission, when prices rise again, to change the period used to determine the average price so as to obtain a higher average, more nearly corresponding with current prices. In this way the mines would always be taxed on the basis of high prices. In other words, the human and political element must be considered, and the future average price must be forecasted as accurately as possible, not depending on errors to balance.

6. *Valuation of Undeveloped Mining Land.*—"Self-assessment" may be practical for timber and real estate where the value is in sight and may be estimated closely. In the case of undeveloped mining property, however, the owner does not know the true value, and has no means of determining it except by expensive exploration. It surely is not right to force him to explore it, involving not only the expenditure of large sums of money but a large annual loss in interest on this expenditure until such time as the property can be developed. It would be suicidal for an operator to attempt to explore and develop all his undeveloped land at once, yet that is what "self-assessment" would lead to. Such a policy would result in over-expansion of the operator, overproduction of iron ore, and collapse in the first period of business depression. Would Mr. Brinsmade wish to force an overproduction of iron ore similar to the overproduction of bituminous steam coal, or rather the over-capacity of production, in this country? It is well known that most soft-coal mines are closed down a part of the year. Iron mining is now a prosperous industry and the miners are well housed and fed and have steady work all the year round as a rule. There is over-capacity enough in the iron business now without forcing development of all undeveloped lands at once. "Self-assessment" is diametrically opposed to conservation, to which more and more attention is being directed of late.

If "self-assessment" did not lead to over-development and over-production, it would lead to confiscation of property by the State, and we have not yet reached that stage in this country. The State has no right to force an owner to sell or to name a price at which he will sell.

Another disadvantage would be the possibility of the land becoming

more valuable before a second assessment. It would be a temptation to graft and dishonesty on the part of office holders, who might confiscate the property and sell to more favored parties. Others would probably know of the increased value long before the owners, and by conniving with State officials might confiscate it.

Summary.—In Mr. Brinsmade's summary he states that Mr. Finlay's general scheme of mine valuation has been sustained by the Michigan courts. This is true, as far as it goes, but it must be borne in mind that it was only the general scheme that was ruled upon in the Newport and Sunday Lake cases, to which Mr. Brinsmade undoubtedly refers; that the factors used by Mr. Finlay were not specifically confirmed; and that these cases have only been decided in the lower court. The Newport case has been heard in the Supreme Court but has not yet been decided. The Sunday Lake case has been appealed, but has not yet been heard in the Supreme Court.

Mr. Brinsmade must be unfamiliar with the terms of leases in the Lake Superior district, since he commiserates "any operators who . . . were unwise enough to take long leases on a fixed royalty with the proviso that the lessee should pay all taxes." As a matter of fact, *all* operators have to take leases either on a flat rate royalty or on a percentage of the selling price per ton, since those are the only royalty terms ever given in the district. A percentage basis is worse for the operator than a flat rate, since as taxes increase the selling price increases and hence the royalty increases. A percentage of net profit would be more desirable from the operator's standpoint, but I know of no such leases in the district. It is doubtful if fee owners would ever agree to such an arrangement, since it would be apt to lead to falsification of records and wrongly kept cost sheets, and with mismanagement they would receive little or no royalty.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

Fuel Oil in the Southwest

With a Bibliography of Fuel Oil Generally

BY WILLIAM B. PHILLIPS, AUSTIN, TEXAS

(New York Meeting, February, 1914)

THIS paper was prepared at the request of Capt. A. F. Lucas, Chairman of the Institute's Committee on Petroleum and Gas, as a preliminary discussion of the fuel oils which are used in the Southwest, giving their composition, uses, prices, etc. Should it afterward be found desirable, these observations may be extended over a larger area. In this connection, I suggest that samples of all kinds of fuel oils, including various tars, from all over the country, be collected and sent to some testing laboratory for examination. We would be glad to extend the facilities of the laboratory of the Bureau of Economic Geology and Technology, at the University of Texas, for this purpose, and to do the work free of all expenses. After such examinations are made and the results classified and studied, it would be possible to suggest specifications under which fuel oils should be purchased, and to arrive at some conclusions respecting the uniformity of such specifications, with due allowance for climatic and other more or less local conditions. While such specifications do not now depart from a common standard anything like as much as specifications for illuminating oils do, yet there are discrepancies which should not exist. Some of these discrepancies are of lessening importance, due to the pronounced tendency to distill every crude oil that can be distilled with profit, and to send residues into the market for fuel oil. Residues from the refineries are of a more uniform chemical composition and physical nature than many of the crude oils that go direct into the fuel market.

The lower flash-point oils are disappearing from the fuel trade and in place of them we are getting oils that do not flash below 220° F., nor burn under 320° F. In the Southwest, it has not been long since we had fuel oils that flashed at as low as 110° F., and the average flash point over a considerable period was 151° F. Now we are using oil that flashes at about 30° lower than the former burning point, which was 256° F. In other words, the flash point has risen, on the average, 100°; and the burning point about 75°.

Less than two years ago the range of flash point in fuel oils brought under our observation was from 110° to 208° , a variation of nearly 100° . These same oils had a burning point from 210° to 303° , a range of nearly 100° . Within the last months, fuel oils have shown a range of flash point from 180° to 276° , and of burning point from 260° to 345° . While the range of flash and burning points in these oils is about the same in each case, taking all kinds of oil, yet the range in residuum oil is much less marked. It is easy to see that a fuel oil flashing at 110° is a dangerous oil. The U. S. Navy regulation of 200° flash point is not unreasonable. We shall recur to this matter again.

As an aid in this investigation, I have prepared a bibliography of books and articles dealing with fuel oil. The basis of these lists is the excellent Bibliography of Petroleum prepared by W. H. Dalton, L. V. Dalton, S. L. James, and E. S. Ward for the third (1913) edition of Sir Boverton Redwood's *Treatise on Petroleum*. These gentlemen listed 8,804 titles, from 1662 to 1911. I went over this list carefully and took out the titles of all books and articles referring specifically to fuel oil and added to it many titles for the years 1911-1912, and part of 1913. All of these titles were then classified under appropriate and convenient headings, so that it is now easy to see what has been published, according to subject. It is not claimed for this bibliography that it is original or complete, but it is thought that the 104 books and the 800 articles listed comprise by far the greater number of the really important books and papers. It represents the results of the reading and selection of some 10,000 titles. I would be much gratified to have this list criticised, corrected, added to or taken from. With the exception of the Russian references, which were translated for Redwood's book, the titles appear in the language in which the books or articles were written and as they are given in the Bibliography of Petroleum above referred to.

The fuel oils that now come into market in the Southwest are of two principal kinds:

1. Crude oil that cannot, under present conditions, be profitably distilled. Such oils contain the distillates which are removed in the refineries, together with the heavy, more or less asphaltic material which remains in the stills after the lighter fractions have been removed.

2. Residuum from the stills (*ostatki*, or *astatki*, of the Russian trade). This contains no light oils and is more or less asphaltic in character.

In the early days of the Texas-Louisiana oil industry a great deal of crude oil was sold as fuel and at very low prices, 20c. a barrel and even less. An analysis of a representative crude oil of that period is as follows:

Beaumont Oil

Fractions, Degrees F.	Per Cent. by Volume	Specific Gravity	Color
84.2 to 212			
212 to 257			
257 to 302	2.9	0.7938	Colorless
302 to 347	2.4	0.8275	Colorless
347 to 392	1.1		Colorless
392 to 437	3.1	0.8387	Pale amber
437 to 482	6.2	0.8590	Pale amber
482 to 527	11.8	0.8748	Straw yellow
527 to 572	15.3	0.8897	Straw yellow
572 to 617	5.8	0.8957	Straw yellow
Above 617	36.8	0.9094	Reddish brown

The color of the crude oil was reddish brown; the flash point was 120° F.; the specific gravity was 0.9103 = 24° Baumé; it contained 2.4 per cent. of sulphur; the British thermal units per pound were 19,785.

Other samples of the early Beaumont oil had a flash point of 165° F. (open cup), a viscosity of 98 at 70° F., and did not congeal at 10° below zero F. They showed less sulphur than the above analysis. The fractions yielded up to 500° F. varied from 19 to 38 per cent., according to different samples and different analysts. A flash test of 110° and a burning test of 180° are recorded, but, so far as the writer is informed, there was no flash point higher than 165° F., nor burning point higher than 200° F. The oil had a comparatively low viscosity, flowed readily, was easily handled by ordinary pumps and did not congeal at a temperature several degrees below zero, 5 to 10.

Fuel oil of this general character was supplied for several years, even after the completion of some of the refineries, and it is still marketed.

Following the completion of the refineries and the laying of pipe lines less and less crude oil was sold for fuel and more of it was distilled. The increasing demand for gasoline, naphtha, etc., consequent upon the development of the internal-combustion engine and chemical industries, was instrumental in causing a gradual shifting of the fuel-oil trade from crude oil to the residuum oil.

In the laboratory of the Bureau of Economic Geology and Technology, University of Texas, we have recently examined 12 samples of representative fuel oils from Texas and Louisiana. These oils were distilled by the Engler method, and, in addition, there were determined the color, specific gravity, viscosity, flash point, burning point, and British thermal units per pound.

The following tables show the character of these oils (analyses by S. H. Worrell and J. E. Stullken):

Analysis No.	Specific Gravity at 60° F.	Viscosity at 70° F.	Flash Point, Deg. F.	Burning Point, Deg. F.	B.t.u. per pound.
1,152	0.898 (26° B.)	2,000	248	302	19,164
1,153	0.921 (22° B.)	228	140	195	18,996
1,156	0.915 (23° B.)	204	140	167	19,040
1,157	0.940 (19° B.)	1,200	248	302	18,880
1,158	0.910 (24° B.)	204	131	156	19,080
1,159	0.898 (26° B.)	1,008	212	302	19,160
1,160	0.940 (19° B.)	1,134	158	230	18,884
1,161	0.921 (22° B.)	106	104	140	19,000
1,167	0.915 (23° B.)	151	194	266	19,028
1,168	0.927 (21° B.)	1,764	266	356	19,312
1,169	0.927 (21° B.)	270	159	257	19,312
1,170	0.940 (19° B.)	3,600	140	185	19,232

In these analyses the specific gravity varied from 0.898 (26° B.) to 0.940 (19° B.), with an average of 0.917 (22.5° B.); the viscosity varied from 106 to 3,600; the flash point varied from 104° to 248° F., with an average, if figures so far apart may be averaged, of 178° F.; the burning point varied from 140° to 356° F., with an average of 236° F. There was not a great variation in the B.t.u. and the average was 19,090.

These samples were also distilled, with the following results:

Analysis No. 1,152. Saratoga Light Crude Oil. Color, Red-Brown

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	None	
302 to 392.....	None	
392 to 482.....	2.5	Prime white
482 to 572.....	17.5	Cream white
572 to 662.....	7.0	Extra pale
Above 662.....	12.5.....	Lemon pale
	20.0.....	Orange pale
	19.5.....	Light red
	13.5.....	Dark red
	65.5	
Residue, by weight.....	7.5	
	100.0	

Analysis No. 1,153. Saratoga Heavy Crude Oil. Color, Dark Brown

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	None	
302 to 392.....	7.0	White
392 to 482.....	10.0	White
482 to 572.....	20.0	Cream white
572 to 662.....	44.0	Lemon pale
Above 662.....	16.0	Red
Residue, by weight.....	3.0	
	100.0	

Analysis No. 1,156. Humble Crude Oil. Color, Red-Brown

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	4.0	White
302 to 392.....	2.5	White
392 to 482.....	10.0	White
482 to 572.....	19.0	Prime white
572 to 662.....	12.0	Cream white
Above 662.....	{ 21.0 19.5	Extra pale Lemon pale
	40.5	
Residue, by weight.....	12.0	
	100.0	

Analysis No. 1,157. Welsh Crude Oil. Color, Dark Brown

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	None	
302 to 392.....	None	
392 to 482.....	4.0	Cream white
482 to 572.....	22.0	Extra pale
572 to 662.....	{ 60.0 5.0	Orange pale Light red
	65.0	
Above 662.....	None	
Residue, by weight.....	9.0	
	100.0	

*Analysis No. 1,158. Humble Crude Oil. Color, Red-Brown
2,800-ft. Sand.*

Degrees F.	Per Cent.	Color
Below 212.....	6.0	White
212 to 302.....	4.5	White
302 to 392.....	17.0	White
392 to 482.....	3.0	Prime white
482 to 572.....	14.0	Cream white
572 to 662.....	9.0	Extra pale
	{ 15.0 20.0 6.0	Lemon pale Orange pale Light red
Above 662.....	40.1	
Residue, by weight.....	5.5	
	100.0	

Analysis No. 1,159. Electra Residuum Oil. Color, Red-Brown

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	None	
302 to 392.....	1.0	White
392 to 482.....	1.0	White
482 to 572.....	11.0	Prime white
572 to 662.....	5.5	Cream white
Above 662.....	18.5	Extra pale
	17.0	Lemon pale
	16.0	Orange pale
	16.0	Light red
	5.5	Dark red
	73.0	
Residue, by weight.....	8.5	
	100.0	

Analysis No. 1,160. Sour Lake Crude Oil. Color, Red-Brown

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	None	
302 to 392.....	None	
392 to 482.....	11.0	Cream white
482 to 572.....	23.0	Extra pale
572 to 662.....	20.0	Lemon pale
	19.0	Light red
	39.0	
Above 662.....	19.0	Red
Residue, by weight.....	8.0	
	100.0	

*Analysis No. 1,161. Humble Crude Oil. Color, Red-Brown
1,200-ft. Sand*

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	11.0	Cream white
302 to 392.....	10.0	Extra pale
392 to 482.....	15.0	Extra pale
482 to 572.....	38.0	Orange pale
572 to 662.....	20.0	Orange pale
Above 662.....	4.0	Red
Residue, by weight.....	2.0	
	100.0	

Analysis No. 1,167. Markham Crude Oil. Color, Light-Brown.

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	None	
302 to 392.....	5.0	White
392 to 482.....	18.0	White
482 to 572.....	{ 20.0	White
	{ 8.0	Extra pale
	28.0	
572 to 662.....	{ 19.0	Extra pale
	{ 20.0	Lemon pale
	{ 4.0	Orange pale
	43.0	
Above 662.....	5.0	Red
Residue, by weight.....	1.0	
	100.0	

Analysis No. 1,168. Caddo Heavy Crude Oil. Color, Dark Red-Brown

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	None	
302 to 392.....	None	
392 to 482.....	8.0	White
482 to 572.....	28.0	Extra pale
572 to 662.....	13.0	Lemon pale
Above 662.....	{ 23.0	Greenish tinge
	{ 14.0	Light red
	{ 6.0	Dark red
	43.0	
Residue, by weight.....	8.0	
	100.0	

Analysis No. 1,169. Vinton Crude Oil. Color, Red-Brown

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	None	
302 to 392.....	3.5	White
392 to 482.....	11.0	White
482 to 572.....	21.0	White
572 to 662.....	{ 21.0	Cream white
	{ 19.5	Extra pale
	40.5	
Above 662.....	{ 20.0	Light red
	{ 3.0	Red
	23.0	
	1.0	
Residue, by weight.....	100.0	

Analysis No. 1,170. Mexican Crude Oil. Color, Black

Degrees F.	Per Cent.	Color
Below 212.....	None	
212 to 302.....	None	
302 to 392.....	8.0	White
392 to 482.....	6.0	White
482 to 572.....	14.0	Extra pale
572 to 662.....	20.0	Light red
	26.0	Red
	46.0	
Above 662.....	8.0	
Residue, by weight.....	18.0	
	100.0	

It is to be observed from these analyses:

1. The oil of the lowest flash point, 104° F., had also the lowest viscosity, 106. Up to 482° F. (250° C.), it yielded 36 per cent. of distillates, and left but little residue in the still. The burning point of this oil was 140° F. This is not a safe oil for fuel purposes, the flash point and burning point being entirely too low.

2. The oil of the highest flash point, 266° F., had a high viscosity, 1,764. Up to 482° F., it yielded 8.0 per cent. of distillates, and left 8 per cent. of residue in the still. The burning point of this oil was 356° F. This is a very safe oil for fuel purposes; but, on account of its high viscosity, it will have to be handled through larger pipes and with special pumps.

3. The oil of a medium flash point, viz., 194° F., had a comparatively low viscosity, 151. Up to 482° F., it yielded 23 per cent. of distillates. Its burning point was 266° F. While this oil, in respect of its flash point, would not meet the requirements of the U. S. Navy, 200° F., yet it is a safe fuel.

4. Out of the 12 samples examined, there were but four that showed a flash point above 200° F. Of the remaining eight, one flashed between 190° and 200°; two between 150° and 160°; three between 140° and 150°; one between 130° and 140°; and one between 100° and 110°. If we accept the Navy specification as to flash point, two-thirds of these oils would be rejected, and yet they are representative fuel oils, sold and used in large quantities throughout the Southwest. So far as the writer is informed, no serious catastrophe has occurred from the use of any of these oils, even those of lowest flash point.

In order to elucidate this matter further, I give the results of the examination of 31 samples of fuel oil tested in our laboratory for the Purchasing Agent of the State of Texas, from Nov. 26, 1912, to Jan. 22, 1914. The State of Texas maintains a Purchasing Agent, whose duty

it is to secure supplies for the charitable institutions. All of the fuels are analyzed in our laboratory regularly, the samples being taken at the points of consumption under instructions from this official. It is but fair to say that we do not have charge of the sampling, but we do advise the State Purchasing Agent how the samples should be taken. The examination of these samples shows (analyses by S. H. Worrell and J. E. Stullken):

	From	To	Average
Specific gravity.....	0.868 (31° B.)	0.931 (21° B.)	0.898 (26° B.)
Viscosity at 70° F.....	204	1,026	548
Flash point.....	110	291	203
Burning point.....	180	350	277
B.t.u. per pound.....	18,964	19,500	19,173

There has been a marked change in the character of fuel oils examined for the State Purchasing Agent since November, 1912, with respect to flash and burning points. During 1912-1913, we examined 19 samples. These showed:

	From	To	Average
Specific gravity.....	0.868 (31° B.)	0.931 (20° B.)	0.898
Flash point.....	110	218	151
Burning point.....	180	276	221

Of 12 samples examined for flash point, only two came up to the Navy requirements of 200° F. One sample gave between 190° and 200°; one sample between 180° and 190°; one between 140° and 150°; one between 130° and 140°; five between 120° and 130°; and one between 100° and 110°.

Beginning in the fall of 1913 and continuing through the winter deliveries, 12 samples of fuel oil, under contracts for the State, showed the following:

	From	To	Average
Specific gravity.....	0.892 (27° B.)	0.904 (25° B.)	0.90 (25° B.)
Viscosity.....	204	1,026	548
Flash point.....	210	303	250
Burning point.....	260	350	324

As compared with the quality in 1912 and the early part of 1913, we observe that the flash point increased from 110° to 210° F., the average increasing from 151° to 250°; the burning point increased from 180° to 260°, the average increasing from 221° to 324°. No viscosity determinations were made on the earlier samples.

Of these 12 samples, not one of them flashed as low as 200° F., and the average of the entire lot was 250°. These samples were taken from actual deliveries at points of consumption and are not company samples at all.

Every one of these samples represents a safe fuel oil, whereas, of the former lot only two out of twelve, or 16.6 per cent., are to be considered as safe, from the standpoint of flashing.

During the last months there has been some complaint from fuel-oil consumers as to the quality of the oil. In nearly every case that has been investigated, the trouble was found to be in the methods of handling, in the pumps and in the burners. In one particular case the oil was pumped about 1½ miles from the railroad siding to the storage tanks at the boilers. It was handled with difficulty. The pipe was too small and the pumps were not designed to handle a heavy, viscid oil.

In another case, the consumer said that he was compelled to add about 300 gal. of kerosene to his fuel oil in order to get it through the pipes to the burners, by gravity. What the insurance company would have said to him is another matter! But neither the insurance companies nor the municipal authorities are supposed to know everything and there is an old, but questionable, statement, to the effect that what you don't know won't hurt you.

Water in Fuel Oil

I approach this subject with some hesitancy, for I am aware of the confusion attending it. The accurate determination of water in heavy oils is a matter of considerable difficulty, and it is doubtful if any of the methods now proposed is reliable in every case. Of course, there should not be water in fuel oil, but, equally of course, it is sometimes present. It is gratifying to be able to say that of the 50 odd commercial samples of fuel oil examined in our laboratory during the last two years, only two contained appreciable amounts of water. Of these two, however, one contained 20 per cent., and the other 5 per cent. Neither of these oils was a very heavy oil, so that we were able to determine the water with a fair degree of accuracy.

We are now trying out a number of methods for the determination of water in heavy crude oils and in residuum oils and hope to be able to report on this matter later. As far as we have progressed, however we are able to say that the gasoline method, as adopted by the U. S. Navy, does not apply to some of the very heavy oils with which we have to deal.

Residuum oils can contain water only if it is purposely added or rain water, etc., gets into the tank cars or reservoirs.

Dirt, Sand, Sediment, etc.

Practically all of the oils that we have examined were commercially free from sand, sediment, etc. One complaint that reached us was found to be due to an insufficient pipe line and to inadequate pumps and burners, not to the quality of the oil. If the well casing is not properly set, or if there is a break in the casing, opportunity is given for the intake of dirt, etc., but we have not found that the fuel oils are dirty oils. One consumer complained that his oil was dirty because he could write his name in the dirt that was on top of the oil. This complaint was not found to be well based.

Sulphur

We have not examined many fuel oils for sulphur. As a rule, this ingredient is not included in specifications, and is not ordinarily determined. In some fuel oils the sulphur may be as high as 1 per cent., but the average is probably below 0.75 per cent.

Viscosity

Considering the advent of very heavy, viscid oils into the fuel trade, the degree of viscosity should always be specified. In the U. S. Navy it is required that fuel oil shall flow freely from a round orifice of $\frac{1}{2}$ in. under a head of 2 ft., at a temperature of 40° F. This is a severe requirement and should be modified. A better plan would be to require that the oil shall flow through an orifice of given size at the rate of so many gallons, or fractions thereof, per unit of time, at a temperature of 60° F., and under a given head. Many of our best fuel oils would be excluded under the present specifications. The expression "shall flow freely and in a continuous stream" is entirely too vague. A temperature of 40° F. is too severe a requirement, considering that the flash point is placed at the minimum of 200° F.

We determined the specific gravity at 60° F. and the viscosity at 70° F., in 24 samples. The results were as follows—specific gravity, upper figures; viscosity, lower figures:

0.892	0.898	0.902	0.904	0.910	0.915	0.921	0.927	0.940
250	444	1,026	204	204	151	106	270	1,134
285	636		384		204	226	1,764	1,200
	840		560					3,600
	1,008		564					
	2,000		618					
			768					

According to these results, there is no connection between the specific gravity and the viscosity. While it is true that the heaviest oil had the

highest viscosity, yet one of the lighter oils had a viscosity greater than two of the samples of the heaviest oil. The lowest viscosities observed, 106 and 226, were in a heavy oil, specific gravity 0.921. The lightest oil did not have the lowest viscosity.

There are heavy liquids that are not viscous and there are viscous liquids whose viscosity is exceeded by that of lighter liquids.

Viscosity arises from the mutual attraction of the molecules. An increase of temperature diminishes it and a decrease of temperature increases it. For the easily flowing oils we should employ the term *mobility*, as this is the opposite of *viscosity*. Just where the line of separation should be drawn, for fuel oils, I am not now prepared to say, but would venture to suggest that it be made a subject of investigation by the Institute's Committee on Petroleum and Gas. With such a wide range in fuel oils, 106 to 3,600, in what is termed "viscosity," it would appear that there is urgent need of a new nomenclature. There are five principal elements that enter into this matter:

1. Time.
2. Amount of oil.
3. Temperature.
4. Size and condition of orifice.
5. Head.

The term "viscosity" should relate to the *time* required for a given *amount* of oil at a given *temperature* to flow through a given *orifice* under a given *head*. This would clarify a much-troubled subject.

In *Technical Paper No. 3* of the U. S. Bureau of Mines, Washington, 1911, Irving C. Allen proposes the following specifications for fuel oil to be purchased by the government:

(1) In determining the award of a contract, consideration will be given to the quality of the fuel offered by the bidders, as well as the price, and should it appear to be to the best interest of the Government to award a contract at a higher price than that named in the lowest bid or bids received, the contract will be so awarded.

(2) Fuel oil should be either a natural homogeneous oil or a homogeneous residue from a natural oil; if the latter, all constituents having a low flash point should have been removed by distillation; it should not be composed of a light oil and a heavy residue mixed in such proportions as to give the density desired.

(3) It should not have been distilled at a temperature high enough to burn it, nor at a temperature so high that flecks of carbonaceous matter began to separate.

(4) It should not flash below 60° C. (140° F.) in a closed Abel-Pensky or Pensky-Martens tester.

(5) Its specific gravity should range from 0.85 to 0.96 at 15° C. (59° F.); the oil should be rejected if its specific gravity is above 0.97 at that temperature.

(6) It should be mobile, free from solid or semisolid bodies, and should flow readily, at ordinary atmospheric temperatures and under a head of 1 ft. of oil, through a 4-in. pipe 10 ft. in length.

(7) It should not congeal nor become too sluggish to flow at 0° C. (32° F.).

(8) It should have a calorific value of not less than 10,000 calories per gram (18,000

B.t.u. per pound); 10,250 calories to be the standard. A bonus is to be paid or a penalty deducted according to the method stated under section 21, as the fuel oil delivered is above or below this standard.

(9) It should be rejected if it contains more than 2 per cent. water.

(10) It should be rejected if it contains more than 1 per cent. sulphur.

(11) It should not contain more than a trace of sand, clay, or dirt.

The specifications for fuel oil for the U. S. Navy are as follows:

1. Fuel oil shall be a petroleum of best quality, free from grit, acid, fibrous and other foreign matter likely to clog or injure the burners or valves.

2. The unit of quantity to be a gallon of 231 cu. in. at a standard temperature of 60° F. For every variation of 10° F. from the standard, one-half of 1 per cent. shall be added to or deducted from the measured or gauged quantity for correction.

3. Sulphur in the oil must not exceed three-fourths of 1 per cent. by weight.

4. Free water and sediment in the oil shown by gasoline test must not exceed 1 per cent. by volume. The gasoline test shall be made as follows:

5. Samples taken at random should be thoroughly shaken and mixed; then 50 c.c. of sample placed in a 100-cm. graduated glass cylinder. An equal quantity of not less than 68° gasoline should be mixed with the sample in the graduated glass; the combined mixture of oil and gasoline should then be thoroughly shaken and mixed and allowed to stand not less than 6 hrs., when percentage of water and sediment will be taken by the inspector.

6. The oil shall not flash at a temperature less than 200° F., the test to be made by the Abel or Pensky-Martens closed cup method.

7. The oil shall have a gravity Baumé not greater than 30° at 60° F., determined from an average sample.

8. The oil shall flow freely and in a continuous stream through a $\frac{1}{2}$ -in. circular hole, under a 2-ft. head at a temperature of 40° F. Oil that fails to flow freely and in the required quantities to pump suction, pass through pumps, piping and burners at a temperature of 40° F. may be rejected.

9. The oil shall have a calorific value not under 144,000 B.t.u. per gallon, to be taken from an average sample or samples of the product as delivered, or made before delivery. In determining this value, the bomb calorimeter shall be used to determine the B.t.u. per pound from which the B.t.u. per gallon shall be calculated by using 8.331 lb. of distilled water per gallon at 60° F. and the specified gravity of the oil as determined by the Baumé test at same temperature.

Mr. Irving recommends a flash point not below 140° F., the Navy specifications a flash point not below 200° F. Mr. Irving recommends that the oil should be so mobile as to flow readily, at ordinary atmospheric temperatures and under a head of 1 ft. of oil, through a 4-in. pipe 10 ft. in length. The Navy calls for an oil that shall flow freely and in a continuous stream through a $\frac{1}{2}$ -in. circular hole, under a 2-ft. head, at a temperature of 40° F.

Mr. Irving recommends that the oil should not contain more than 2 per cent. of water or 1 per cent. of sulphur, or more than a trace of sand, clay, or dirt. The Navy allows 1 per cent., by volume, of free water and sediment and allows not over 0.75 per cent. of sulphur. It is probable that most of the heavy oils now sold in the Southwest would have no special difficulty in meeting any of these suggestions or specifications, with the exception of the mobile and viscosity tests. I have not had

time or opportunity for inquiring into the specifications for fuel oil as adopted by foreign governments, but this will be done as soon as possible.

Production

During the last six years, in which there has been so great a development of the fuel-oil industry, the total production of crude petroleum in the United States was 1,179,823,421 barrels of 42 gal. Of this amount, the Southwest produced as follows:

	Barrels
Louisiana:.....	40,673,880
Oklahoma.....	296,707,537
Texas.....	63,225,422
	400,606,839

Or about 34 per cent. of the total.

It is not known how much of the oil produced in these States was sold, for fuel purposes, as crude oil, or how much was first sent to the refineries and afterward sold. Such statistics as have been collected are incomplete and, therefore, misleading. Furthermore, not all of the residues from the stills goes into the fuel market, for considerable quantities are used for roofing, for road surfacing, for bitulithic purposes, etc.

Imports

The value of the imports for consumption of petroleum during the last five years is reported by the U. S. Bureau of Foreign and Domestic Commerce as follows:

1908.....	\$607,858
1909.....	197,023
1910.....	1,398,861
1911.....	2,410,884
1912.....	6,082,881
	\$10,697,307

The imports of Mexican crude and residuum have been as follows (*Fuel Oil Journal*, vol. v, No. 2, pp. 7, 8, Feb., 1914):

In 1912 the imports were 6,500,000 barrels, and in 1913, with the exception of December returns from Port Tampa and Jacksonville, Florida, were 16,360,779 barrels. By ports the figures are:

	Barrels	Value
Galveston.....	3,130,497	\$1,702,546
Sabine.....	2,490,613	1,328,529
New Orleans.....	2,318,297	1,385,898
New York.....	2,155,091	2,028,335
Port Arthur.....	1,501,524	856,736
Corpus Christi and Port Aransas.....	1,317,049	398,452
Tampa.....	972,550	605,195
Baltimore.....	914,469	557,628
Philadelphia.....	634,725	341,596
Jacksonville.....	295,570	173,720
Providence.....	114,000	55,378
Missing returns and miscellaneous.....	516,394	
	16,360,779	\$9,434,013

Exports—Territorial

The shipments of crude petroleum to Alaska, the Hawaiian Islands, and Porto Rico, during the years named, were as follows, in barrels of 42 gal.:

	Alaska	Hawaiian Islands	Porto Rico
1906.....	64,000		
1907.....	216,770	926,581	
1908.....	283,128	1,136,900	5,937
1909.....	334,283	1,034,797	1,211
1910.....	448,467	1,288,502	2,080
1911.....	431,959	1,114,084	12,299
1912.....	79,144	917,238	602
Total.....	1,857,751	6,418,102	22,128

Exports

The exports of crude oil during the last four fiscal years, ending June 30, were as follows, in barrels of 42 gal.:

	1909	1910	1911	1912
To Europe.....	1,619,037	542,233	797,102	1,226,228
To North America.....	1,989,612	2,755,732	2,949,956	3,183,372
To South America.....	242,448	722,706	661,764	543,794
To Japan.....	192,915			
Total.....	4,044,012	4,020,671	4,408,822	4,953,394

During these same years, the exports of residuum, in barrels of 42 gal., were as follows:

	1909	1910	1911	1912
Europe.....	92,070,389	112,792,362	102,430,883	111,321,764
North America.....	10,962,529	10,742,492	15,708,381	30,443,892
All others.....	155,115	520,924	5,258,924	26,573,822
Total.....	103,188,033	124,055,778	123,408,188	168,339,478

The exports of crude oil during these years remained fairly steady, but there was a considerable increase in the exportation of residuum, amounting to about 63 per cent. as between 1909 and 1912.

Prices

The prices of fuel oil in the Southwest as given by the *Fuel Oil Journal*, Houston, are as follows, at different points:

During the last three months of 1911, prices at Texas points varied from 70c. to 75c. a barrel, and at Louisiana points, from 63c. to 80c., the latter price maintaining for Mississippi river points.

During 1912, prices at Texas points were quoted at 70c. in January and 95c. to \$1 in December. At Louisiana points prices varied from 63c. in January to \$1 in December. In Oklahoma (Tulsa), they varied from 60c. in January to \$1.15 in December.

In 1913, Texas prices opened at \$1.10 to \$1.26, and advanced during the year to \$1.18 and \$1.30. At Louisiana points the prices varied from \$1 to \$1.25, and in Oklahoma (Tulsa) quotations in January were from \$1 to \$1.25, declining to 75c. to 85c. in December. During 1913, the prices of Mexican crude at Texas ports ranged from \$1.05 to \$1.20.

During the last two years there has been a stiffening of prices—viz., from 70c. to \$1.30 a barrel.

In the year 1900, just preceding the discovery of the Beaumont oil field, the average value of crude petroleum in Texas was \$1.04 a barrel. In 1901 it was 29c., and in 1902 it was 22c., the lowest average price ever reached in the State.

The change in the conditions surrounding the industry is well shown in the fact that the 28,136,189 barrels of crude oil produced in Texas in 1905 were worth \$7,552,262, or 27c. per barrel, while the 11,735,057 barrels produced in 1912 were worth \$8,852,713, or 75c. a barrel. Residuum oil is now selling 30 per cent. higher than crude oil in 1900, and about 170 per cent. higher than crude oil in 1902.

In 1907, the production of crude oil in Oklahoma was 43,524,128 barrels, valued at \$17,513,524, or 40c. a barrel. In 1912, the production was 51,427,071 barrels, valued at \$34,672,604, or 67c. a barrel.

In 1905, the production of crude oil in Louisiana was 8,910,416 barrels, valued at \$1,601,325, or 18c. a barrel. In 1912, the production was 9,263,439 barrels, valued at \$7,023,827, or 75c. a barrel.

The production of what is known as the Gulf oil field (coastal Louisiana and coastal Texas) reached its maximum in 1905, with 36,526,323 barrels. In 1912, the production was 8,545,018 barrels.

These figures would seem to indicate that these oils are now being distilled for the recovery of the lighter oils and the increase in the number of refineries substantiates this view.

It is not possible, at this time, to state just what proportion of fuel oil is crude oil, but it is certainly much less than it was a few years ago, and is likely to decrease in the future, insofar as concerns the Southwest.

With the exception of the crude oil imported from Mexico, a large proportion of the fuel oil used in the Southwest is residuum from the refineries. Just how much is used, we do not know, for the statistics to hand cover only the use by railroads, and even these are incomplete for the area involved.

Dr. David T. Day, in *Mineral Resources of the United States*, U. S. Geological Survey, 1912, gives a table to show the consumption of fuel oil by the railroads of the United States, from 1906 to 1912, inclusive. This table is as follows:

Consumption of Fuel Oil by the Railroads of the United States, 1906-1912.

Year	Length of lines operated with oil. Miles	Quantity of oil consumed. Barrels	Total mileage made by engines	Average no. of miles per barrel of oil
1906		15,577,677		
1907	13,573	18,849,803	74,079,726	3.93
1908	15,474	16,870,882	64,279,509	3.81
1909	17,676	19,905,335	72,918,118	3.66
1910	22,709	23,817,346	89,107,883	3.74
1911	30,039	29,748,845	109,680,976	3.69
1912	28,451	33,605,598	121,393,228	3.61
Total and average...	127,922	158,375,486	531,459,440	3.74

Dr. Day gives a list of 48 railroads (and systems) that used fuel oil in 1912, and of these 23 are in the Southwest.

According to the *Fuel Oil Journal* (vol. v, No. 2, pp. 66 to 68, Feb., 1914) the consumption of fuel oil by the railroads in Texas and Louisiana in 1913 was 11,148,254 barrels, as follows:

	Barrels
Southern Pacific.....	4,455,749
Gulf, Colorado & Santa Fé.....	2,342,703
Kansas City Southern.....	922,191
San Antonio & Aransas Pass.....	703,145
Frisco Lines.....	513,912
Trinity & Brazos Valley.....	409,459
Houston Belt & Terminal.....	104,240
All others.....	1,696,155
	11,148,254

In 1913, the Southern Pacific imported 1,784,143 barrels of Mexican crude for its own use as fuel.

Dr. Day states that the estimated consumption of fuel oil by railroads and for other industrial purposes, in California, in 1912, was 65,000,000 barrels. He also states that during the year 1912, the U. S. Navy used 21,000,000 gal. of fuel oil and that the consumption for 1913 will probably be 30,000,000 gal.

The President of the United States has set aside about 100 square miles of oil-producing land in the Elk Hills and Buena Vista fields, California, as a naval petroleum reserve. This reserve has an estimated capacity of 250,000,000 barrels.

For the U. S. Navy, tank storage is provided as follows:

	Barrels
Boston.....	36,000
Melville, R. I.....	36,000
Norfolk.....	36,000
Charleston.....	36,000
Key West.....	36,000
Guantanamo, Cuba.....	223,000
Pearl Harbor, H. I.....	200,000
	603,000

It is also stated that storage tanks are to be established at New York, San Francisco, San Diego, Puget Sound, Guam, and Cavite. At the Philadelphia Navy Yard there is maintained a well-equipped fuel-oil testing plant, where evaporative tests with various burners and boilers are conducted and instruction is given to the officers and men.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

A Proposed New Converter, and the Application of the Bessemerizing Process to the Smelting of Ores

BY HERBERT HAAS, SAN FRANCISCO, CAL.

(New York Meeting, February, 1914)

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I. INTRODUCTION

COPPER matte is now converted into blister copper at a cost of only \$5 per ton of copper, or, based on a 40 per cent. matte, \$2 per ton of matte, which is the record of at least one large smelting works; and it seems hardly likely that a further reduction, large enough to warrant a change in existing practice, could be made in bessemerizing costs. But the new type of converter here described and the new practice proposed are intended, not so much for the treatment of mattes, as for the extension of the process to the direct reduction in the copper converter of pyrite ores

with increased amounts of copper concentrates, without previous smelting of the pyrite ore in the blast furnace, or roasting of the concentrates in the MacDougall furnace and then smelting them in the reverberatory.

Let me explain at the beginning that the process here outlined will not displace existing methods of blast-furnace smelting, or roasting and reverberatory smelting. The term "pyrite ores" defines the character of the material that can be treated direct by a pneumatic process, namely, ores containing sufficient iron and sulphur to furnish the heat required to produce and sustain the smelting and bessemerizing temperature. And, the tonnage of concentrates that can be smelted direct in the converter depends likewise chiefly on their iron and sulphur contents; a decrease in these two constituents requires an increase of them in the matte of the charge. In other words, the matte should be, in such a case, low in copper and high in iron and sulphur, so that it may furnish the requisite heat to smelt the more siliceous copper concentrates, high in copper but low in iron. The copper concentrates will then take the place of the siliceous flux, for slagging the iron in the matte. In a number of smelters, where siliceous concentrates preponderate, the large converter can be used, at least partially, as a smelting furnace. This is already being done at a few plants, notably at the Great Falls smelter, in Montana, and that of the Cananea Consolidated Copper Co., in Sonora.

The chief items of cost in converter smelting would be the blast for bessemerizing, and the renewal of the magnesite lining. That the latter need not be prohibitive is evident from the fact that magnesite costs have been lowered already to 30c. per ton of copper, or only 12c. per ton of 40 per cent. matte converted, an amount of no great consequence. That magnesite lining consumption is likely to decrease with an increase in the size of the converter will be shown later.

If converter smelting is to find wide application in the treatment of copper ores of such character as those of Shasta county, Cal. (Mammoth Copper Co., Kennett; Balaklala Cons. Copper Co., Coram; Mountain Copper Co., Keswick), the Tennessee Copper Co. and the Ducktown Sulphur, Copper & Iron Co. in Tennessee, the Rio Tinto mines in Spain, etc., it must be cheaper than the present established practice of combined blast-furnace smelting and bessemerizing operations, or possess other compensating economic advantages.

As the speed with which such ores can be smelted and their impurities, iron and sulphur, eliminated depends entirely on the rate with which the air is pressed through the molten sulphides, I have concerned myself principally with a study of this most important factor of commercial success. To put through the converter the large tonnage of sulphide ores now smelted in the blast furnace, will require the supply to the converter of larger air volumes than are now customary. For the process would be the concentration, not of a matte, but of an ore, to blister copper;

and hence the air requirements per ton of copper would be greatly increased.

Technical efficiency requires that there should be no free oxygen in the waste gases after the air has passed through the molten sulphides. Such oxygen corresponds to a certain amount of excess of air, the compression of which has consumed power, while its presence lowers the temperature of the molten sulphides, since it has to be heated with them, and thus adds to the heat carried off in the waste gases. In the smelting and bessemerizing of ores at one operation in the converter such an excess of air is of course a much more serious matter than in bessemerizing mattes, since the loss per ton of copper produced would assume more serious proportions for the lower-grade copper ores than for the relatively high-grade copper mattes.

Finally, the air should be used at as low a pressure as possible, to lower the cost of compressing it.

The detailed discussion which follows is based on the above general premises.

II. DEVELOPMENT OF CONVERTER PRACTICE

Without reviewing the various attempts which preceded the successful solution of the problem by Pierre Manhès and Paul David in 1880, at

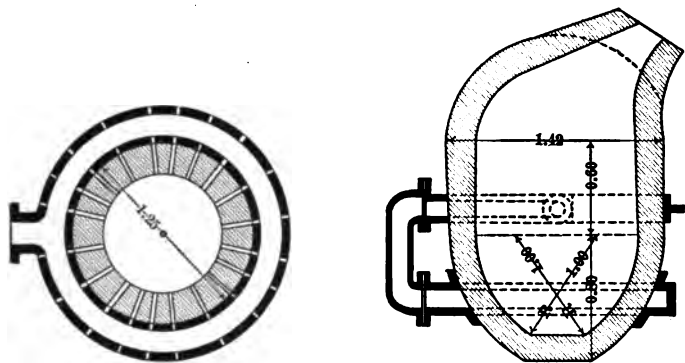


FIG. 1.—FIRST COMMERCIAL MANHÈS-DAVID COPPER CONVERTER.

the Védennes smelter in Vaucluse, France, it will suffice to describe the first converters installed at Eguilles in 1881, marking the introduction of Bessemer's process applied to the conversion of copper mattes on a commercial scale.¹

This converter is illustrated in Fig. 1, taken from Schnabel's *Metall-*

¹ For a more extensive article on this subject, see Development of Converter Practice, by H. Haas, *Mining and Scientific Press*, vol. cvii, No. 17, p. 653 (Oct. 25, 1913).

hüttenkunde, 2d ed., vol. i, p. 224. It had 20 tuyères, of 12 to 15 mm. diameter, 15 cm. above the bottom. The wind box surrounding the shell had small punch holes closed by wooden plugs, very much as is done to-day with the peepholes in the tuyères of silver-lead furnaces. A peculiar feature was the arrangement of the tuyères around the entire periphery of the shell, so that the blast could play on the matte from all sides.

"As a variety of mattes had to be treated (mattes high as well as low in iron and correspondingly low and high in copper), with the variety in custom ores handled and treated, it frequently happened when blowing low-grade matte that the enriched matte or white metal would be *below* the tuyères, so that the blowing was on slag, and the charge could not be blown to a finish, yielding copper. This difficulty was first overcome by removing the enriched matte or the white metal to another converter and finishing the blow in that manner. But it soon led to the adoption by Paul David, in 1885, of a cylindrical form of converter that could be turned around its central longi-

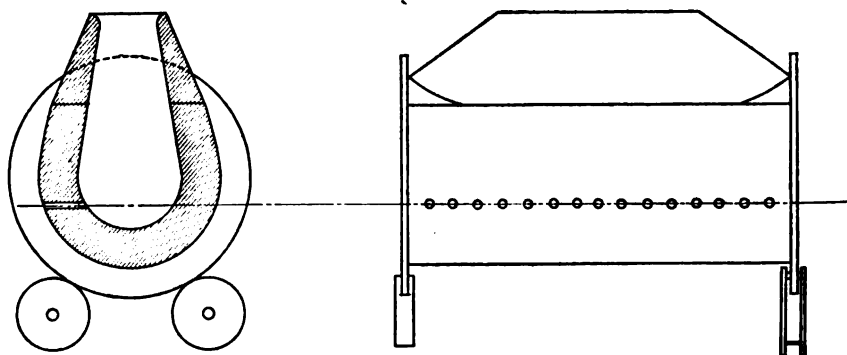


FIG. 2.—LEGHORN CONVERTER.

tudinal axis. With this construction the relative position of the tuyères could be changed (raised or lowered), so that a variety of mattes could be blown in the same converter and in one operation. It also reduced the blast pressure needed. It is this cylindrical form of converter that was introduced at Livorno, Italy (known as the Leghorn converter), and adopted later by the Copper Queen Con. Mining Co. for its Bisbee works."¹

On account of the reduced blast pressure required by the horizontal converter (Fig. 2), this form was generally adopted by most copper smelters, whereas the vertical shell continued in use, to my knowledge, only at the Parrot and Great Falls works in Montana, and the Aguas Calientes smelter in Mexico.

With the adoption of the magnesite-lined converter, the horizontal cylindrical shell has of late been superseded by the vertical shell. This change was made because the vertical cylinder supports the costly refractory lining better, and thus increases its life. In the horizontal type, one-

¹ *Loc.cit.*, p. 655.

half of the lining (neglecting the heads) is in the form of an arched roof, the other being an inverted-arch bottom, the two changing their relative position with the lowering or raising of the tuyères. This construction makes it very much more difficult, especially with the long Peirce-Smith cylinders, to absorb the internal strains set up by the unequal expansion and contraction of the lining, and prevent damage to the arched semi-cylindrical or semi-elliptical roof.

A feature of later converter construction that impressed me was the continued decrease of the matte surface simultaneously attacked by the air blast, with an increase in the size of the converter and the volume of the matte charge. The tuyères were also constantly increased in size, to deliver the larger quantity of air required for oxidizing the iron and sulphur with the necessary speed. This localization of the air to a relatively small portion of the converter is especially noticeable in the large 20-ft. Great Falls converters introduced in the Boston & Montana Reduction Department of the Anaconda Copper Mining Co., Great Falls, Mont. The large volume of air, about 20,000 cu. ft. per minute, is introduced along a strip of the periphery of the lining, about 20 ft. in length. The tuyère orifices are from 8 to 15 ft. from the opposite side of the converter. In the large Peirce-Smith converters, this distance is 10 ft. Here the localization of the oxidation is not quite so pronounced, as the tuyères are spaced over the entire length of the cylinder, from 20 to 33 ft. inside the lining, and a much smaller air volume is delivered. In this case an increase in the size of the converter was secured by increasing its length, which is as much as 37 ft. 2 in. for the converter of the Canadian Copper Co., the tuyères being distributed over a correspondingly longer space.

III. INCREASED AIR PRESSURE REQUIRED FOR VERTICAL SIDE-BLOWN CONVERTERS

I have already mentioned the increased blast pressure required in the vertical side-blown converter with a change in its position from the vertical. If it is tilted up, the tuyères farthest from the axis around which the converter is turned will blow on a shallower matte column than those nearer that axis; and, if turned down, the reverse will be the case. In a vertical cylinder turned around a diametral axis different points on its periphery, lying in a plane normal to the vertical axis of the cylinder, describe circles with radii of different lengths; whereas in the horizontal cylinder, points lying in a line parallel to the axis on which this cylinder revolves describe circles with radii of the same length.

The extreme height or depth to which diametrically opposed sides of a vertical cylindrical vessel become submerged or exposed by a contained liquid is shown in Fig. 3.

Let Fig. 3 represent a cylinder with diameter D and area $A =$

$\left(\frac{D}{2}\right)^2 \pi$; let V be its contained constant volume of liquid, and a and b respective minimum and maximum diametrically opposite heights to which the liquid fills the cylinder, in its different positions, a and b being equal when the cylinder is in the vertical position, and varying in length as the cylinder is tilted. In the figure, the surface of the liquid is represented by the line BC when the cylinder is vertical, by EF when it is tilted θ degrees. We have then

$$A \times \frac{(a+b)}{2} = V = \text{constant}; \quad \frac{a+b}{2} = \frac{V}{A};$$

$$b = a + c; \quad \frac{c}{D} = \tan \theta; \quad \text{and } c = D \tan \theta$$

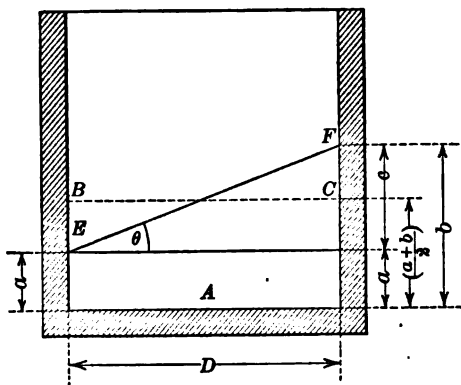


FIG. 3.—DEPTH TO WHICH OPPOSITE SIDES OF CYLINDRICAL VESSEL ARE SUBMERGED OR EXPOSED BY TILTING THE VESSEL.

Substituting, we have $b = a + D \tan \theta$ and

$$a + b = 2a + D \tan \theta;$$

$$a + \frac{D \tan \theta}{2} = \frac{V}{A}.$$

Therefore

$$a = \frac{V}{A} - \frac{D \tan \theta}{2} \text{ and}$$

$$b = a + D \tan \theta = \frac{2V}{A} - a$$

This formula holds good only so long as a or b is never < 0 or $> a + b$, since the bottom area A would then be uncovered by the liquid and other conditions would exist. The cylinder has here been considered in its simplest form, with a bottom normal to its axis. The inverse conical or arched bottom used in practice would introduce unnecessary complications into these calculations.

In a converter with diameter $D=16$ ft., and filled to a height of 3

ft. with liquid, if tilted in 5-degree steps, a and b have the following values, in round numbers:

Tilted Up	Value of a	Tilted Down	Value of b
5°	2.30 ft.	5°	3.70 ft.
10°	1.59 ft.	10°	4.41 ft.
15°	0.86 ft.	15°	5.14 ft.
20°	0.09 ft.	20°	5.91 ft.

At $\theta = 20^\circ 33' 30''$, $a = 0$ and $b = 6$ ft., so that if the converter is tilted further "down," the bottom A will begin to be exposed.

For a converter 8 ft. in diameter inside of lining (which corresponds closely to the 12 by 19-ft. shell used in practice) and filled with liquid to a height of 3 ft., A would not become exposed until $36^\circ 52'$ (nearly), when $a = 0$ and $b = 6$ ft.

This condition will bring about a difference in the static head (hereafter denoted H) of the liquid over the several tuyères. Considering the tuyères of one quadrant only, each of which has a companion tuyère in the other quadrant in the same relative (symmetrical) position, it will be seen that each tuyère orifice will move through a different arc when the converter is tilted, that of the tuyère orifice furthest away from the tilting axis being longest and that nearest the axis being the shortest.

The variation in the static head of the matte or "white metal" over the different tuyères during tilting is illustrated graphically in Fig. 4. The circle represents the perimeter at the inner side of the lining of a vertical converter cut by a plane normal to the vertical axis and through the centers of all tuyère orifices.

This plane, in vertical projection, is seen as a line TT' . 1, 2, 3, . . . 6 are tuyère orifices equidistant (approximately 15° each), which are projected on to TT' as (1), (2), (3), . . . (6). A is the axis around which the converter is tilted, in 5° steps, and to each radius (AC) the plane TT' (seen as a line in its vertical projection) is drawn normal. Lines marked -5° , -10° , . . . -20° show position of TT' when the converter is turned up, and 5° , 10° , . . . 20° show position of TT' when the converter is turned down.

With radii $A(1)$, $A(2)$, $A(3)$, . . . $A(6)$ arcs are drawn, which show the path each tuyère orifice travels when the converter is tilted around axis A . The intersections of these arcs with the lines $+5^\circ$ and -5° , $+10^\circ$ and -10° , . . . $+20^\circ$ and -20° show the position of each tuyère orifice for a given angle through which the converter is turned.

Lines drawn from these intersecting points normal to the plane TT' denote the static head H , assuming that TT' is the surface of the liquid, which would have as datum point the centers of the tuyère orifices when the converter is in the upright position. The height of any additional liquid above TT' would have to be added to the respective "heads" for each tuyère to get the total head for each tuyère below line TT' , and

the respective "heads" for each tuyère above the line TT' (-5° , -10° , etc.) would have to be deducted from the height of any additional liquid above TT' .

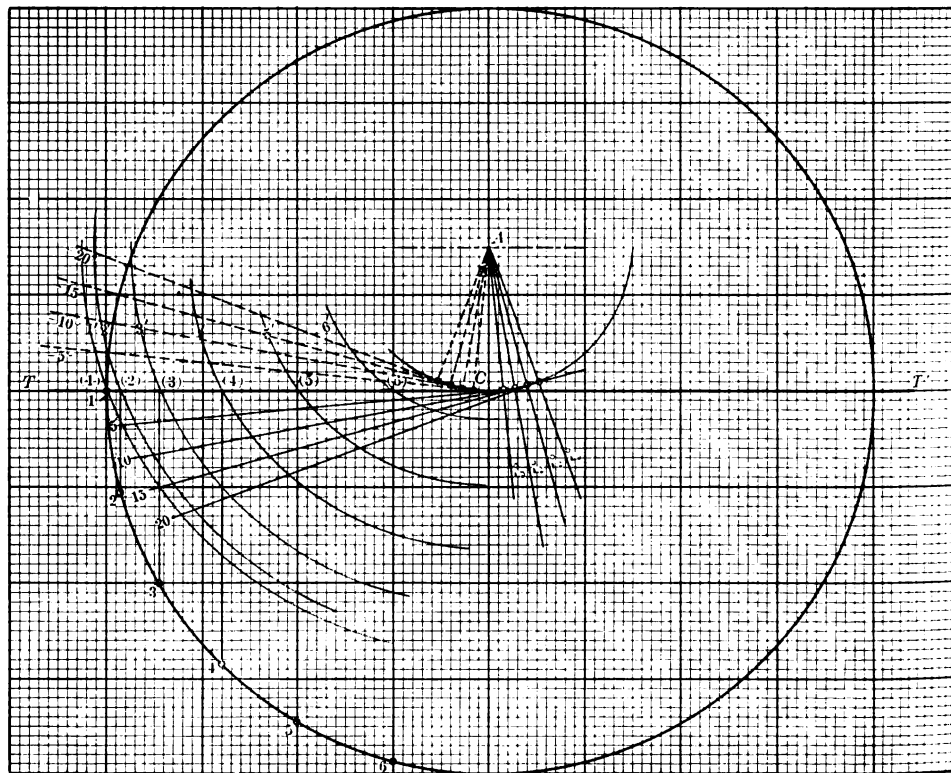


FIG. 4.

Table I shows the heads for the six equidistant tuyères of a converter of 16 ft. internal diameter, tilted at various angles, the axis of revolution, X , being 3 ft. above the plane in which the tuyère openings are situated.

TABLE I.—Heads in Feet for 16-ft. Converter, X 3 ft. Above Tuyères

	1	2	3	4	5	6
5°	0.750	0.700	0.650	0.500	0.350	0.200
10°	1.400	1.350	1.200	0.950	0.700	0.300
15°	2.000	1.925	1.700	1.400	0.950	0.425
20°	2.600	2.500	2.200	1.750	1.200	0.500
-5°	0.800	0.750	0.700	0.600	0.400	0.200
-10°	1.500	1.450	1.300	1.050	0.750	0.400

Table II gives the same data in millimeters. This table will also give the pressure of a column of water of the height stated in millimeters. Since a column of water at 4° C., 1 m. high and 1 m. (i.e., 10,000 sq. cm.) in cross-section weighs 1,000 kg., its pressure upon each square centimeter of base is

$$\frac{1,000 \text{ kg.}}{10,000} = 0.1 \text{ kg. or } \frac{1}{10} \text{ of an atmosphere.}$$

The pressure due to different "heads" of water expressed in millimeters in Table II can therefore be expressed in kilograms per square centimeter by dividing the amounts by 10,000; a water column 10,000 mm. or 10 m. high exerting pressure of 1 atmosphere. By multiplying the metric atmosphere by 14.2234, the pressure in pounds per square inch will be obtained.

TABLE II.—Heads in Millimeters for 16-ft. (4,877-mm.) Converter of Table I

	1	2	3	4	5	6
5°	228.6	213.4	198.0	152.4	106.7	61.0
10°	426.7	411.5	365.8	289.6	213.4	91.4
15°	609.6	586.7	518.2	426.7	289.6	129.5
20°	792.5	762.0	670.6	533.4	365.8	152.4
— 5°	243.8	228.6	213.4	182.9	121.9	61.0
—10°	457.2	442.0	396.2	302.0	228.6	121.9

TABLE III.—Static Head of Matte of sp. gr. 4.5, Expressed in Millimeters of Water. To convert into kilograms per square centimeter, divide by 10,000. To convert into pounds per square inch, multiply by 14.2234

Tuyères	5°	10°	15°	20°	—5°	—10°
1	1,029	1,920	2,743	3,566	1,095	2,057
2	960	1,852	2,640	3,429	1,029	1,989
3	891	1,646	2,332	3,018	960	1,783
4	686	1,303	1,920	2,400	823	1,359
5	480	960	1,303	1,646	549	1,029
6	274	411	583	686	274	549

TABLE IV.—Same as Table III, Except that Specific Gravity of "White Metal" is Taken at 5.5

Tuyères	5°	10°	15°	20°	—5°	—10°
1	1,257	2,347	3,353	4,359	1,341	2,515
2	1,174	2,263	3,227	4,191	1,257	2,431
3	1,089	2,012	2,850	3,688	1,174	2,179
4	838	1,593	2,347	2,934	1,006	1,661
5	587	1,174	1,593	2,012	670	1,257
6	335	503	712	838	335	670

In Tables III and IV the heads in millimeters of water are multiplied by 4.5 and 5.5 to secure the heads in millimeters of water for matte and "white metal" of that specific gravity.

The increase in static head over the different tuyères is quite considerable, even if the converter is tilted down but a moderate amount. We have the following differences in static pressure between tuyères 1 and 6:

Matte, sp. gr. 4.5

	Tuyère 1		Tuyère 6		Difference between 1 and 6	
	Kg. per Cm. ²	Lb. per Sq. In.	Kg. per Cm. ²	Lb. per Sq. In.	Kg. per Cm. ²	Lb. per Sq. In.
5°	0.1029	1.464	0.0274	0.390	0.0755	1.074
10°	0.1920	2.731	0.0411	0.585	0.1509	2.146

White Metal, sp. gr. 5.5

	Kg. per Cm. ²	Lb. per Sq. In.	Kg. per Cm. ²	Lb. per Sq. In.	Kg. per Cm. ²	Lb. per Sq. In.
5°	0.1257	1.788	0.0335	0.476	0.0922	1.312
10°	0.2347	3.338	0.0503	0.715	0.1844	2.623

The above static heads must be added to the heads due to the height of matte above the tuyère orifices, being due entirely to a tilting of the converter and the consequent change of position of tuyère orifices and their distance from the surface of the liquid. Thus with 1 ft. or (nearly) 305 mm. of matte above the tuyère orifices (when the converter is in a normal position), there would have to be added, expressed in millimeters of water, 1372.5 and 1677.5 mm. respectively to the above amounts, for matte and "white metal," or 0.13725 and 0.16775 kg. cm.²

When turning the converter "up," filling it first with matte or "white metal" 305 and 610 mm. above the tuyère orifices, we have the following pressures:

For Matte, sp. gr. 4.5

-5°, 1 ft. matte, tuyère 1	= 0.02745 at. or 0.390 lb. per sq. in.
Tuyère 6	= 0.1097 at. or 1.560 lb. per sq. in.
Difference between tuyères 1 and 6	= 0.08225 at. or 1.170 lb. per sq. in.
-5°, 2 ft. matte, tuyère 1	= 0.1646 at. or 2.341 lb. per sq. in.
Tuyère 6	= 0.2469 at. or 3.512 lb. per sq. in.
Difference between tuyères 1 and 6	= 0.0823 at. or 1.171 lb. per sq. in.
-10°, 2 ft. matte, tuyère 1	= 0.0686 at. or 0.976 lb. per sq. in.
Tuyère 6	= 0.2195 at. or 3.122 lb. per sq. in.
Difference	= 0.1509 at. or 2.146 lb. per sq. in.
-5°, 1 ft. "w. m." tuyère 1	= 0.0336 at. or 0.477 lb. per sq. in.
Tuyère 6	= 0.1441 at. or 2.050 lb. per sq. in.
Difference	= 0.1105 at. or 1.573 lb. per sq. in.
-10°, 2 ft. "w. m." tuyère 1	= 0.0838 at. or 1.192 lb. per sq. in.
Tuyère 6	= 0.2682 at. or 3.815 lb. per sq. in.
Difference	= 0.1844 at. or 2.623 lb. per sq. in.

When turning the converter "up," tuyère 1 would be exposed when the shell is tilted a little more than 5° , so that a matte bath 2 ft. above the tuyères is required to keep the highest tuyère (1) submerged.

It will be seen from the above that, in order not to have any great pressure differential between different tuyères, the converter must be either kept in a vertical position, or only slightly tilted up or down. With a fresh charge of matte, it is preferably tilted up; and after the charge height has been lowered by the removal of slag and impurities (sulphur), it is tilted down.

These figures clearly indicate how large an excess of air pressure is required in a side-blown vertical converter—an excess considerably greater than the actual pressure due to the difference in static head at the various tuyères, since the working pressure must be some multiple of the pressure due to the static head alone, because of the pressure losses due to friction and velocity. With the different resistances offered to the air at each tuyère orifice, the air flow will be concentrated through the tuyères of least resistance and the diminished air flow at reduced pressure through the lower and lowest tuyères will cause these to be filled with matte unless the blast pressure is considerably increased. This increased pressure will increase the air delivery through the tuyères with the least resistance only up to a certain point, the velocity increasing very much more rapidly than the delivery, which brings about a drop in pressure in these tuyères, an increased portion of the energy in the compressed air being represented by velocity and friction heads. This condition tends to adjust in a measure the air pressures at the different tuyère orifices to the static heads over them; but the power losses remain unaltered.

In blowing on a comparatively shallow charge, or when the converter has been turned down so as to blow on matte, it is not impossible that the tuyères nearest the rotating axis of the converter may be blowing on slag.

Notwithstanding these drawbacks, the advantages of the cylindrical shell, in properly securing the magnesite lining, and holding a large volume of matte in a tank-shaped vessel, have caused this type of converter to be preferred to the horizontal one.

IV. BOTTOM-BLOWN CONVERTER WITH ELEVATED TUYÈRE BOTTOM

The converter shown in Figs. 5, 6 and 7 was designed to overcome these disadvantages and at the same time retain the valuable features of the vertical shell. Fig. 5 is a vertical section through the center of the converter; Fig. 6, a horizontal section through the slag mouth; and Fig. 7, a rear view, showing the method of attaching the riding rings to the shell. The shell is made from vertical sheets in rolled sections, riveted together by butt straps. To grip as much as possible of the shell, with the heavy loads it has to hold, and to prevent unequal stresses and distortion, heavy

angle irons in the form of quadrants hug the shell over an arc of 90° , or one-fourth of the circumference. The trunnions are bolted to these angle-iron quadrants and receive the riding rings as a separate casting. To tilt the converter, one of the rings is provided with the customary gearing with suitable train of gears for reducing the speed of the electric motor drive (not shown).

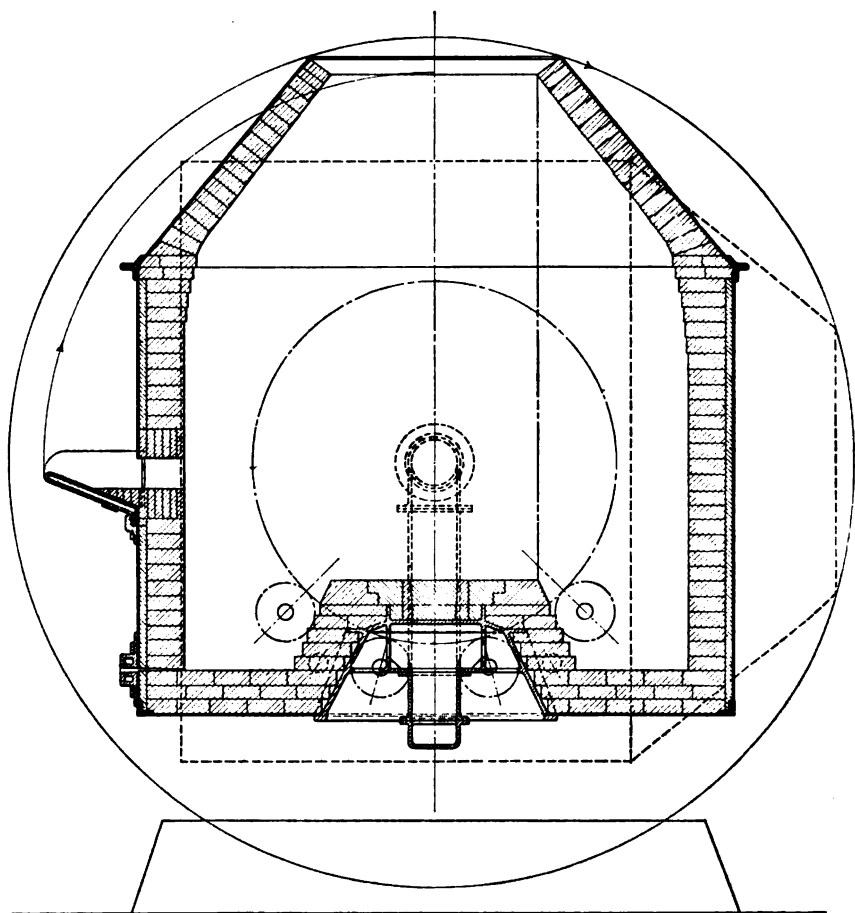


FIG. 5.—VERTICAL SECTION THROUGH HAAS BOTTOM-BLOWN CONVERTER WITH ELEVATED TUYÈRE BOTTOM.

The particular features of construction which distinguish my converter from those now in common use are: the elevated tuyère bottom restricted to a central portion of the converter bottom; a large slag spout for the continuous or intermittent removal of the slag; and a copper spout for the removal of "white metal" or blister copper. The converter can also be used with a flat bottom, in which the tuyères are restricted to the

central portion, dispensing with the elevated tuyère bottom. But the use of the latter has such obvious advantages (to be outlined more extensively below), that the flat-bottomed bottom-blown converter 20 ft. or more in diameter would hardly find application.

The air is delivered to a large wind box by a pipe connected with a stuffing box in the central part of one of the riding rings, connection with the air main being made with the usual stuffing-box glands. The shell shown is 20 ft. in diameter, 15 ft. high, with 7-ft. helmet. To reduce the bearing pressure and balance the body safely during tilting, with a change in its center of gravity, it is carried on four rollers for each riding ring.

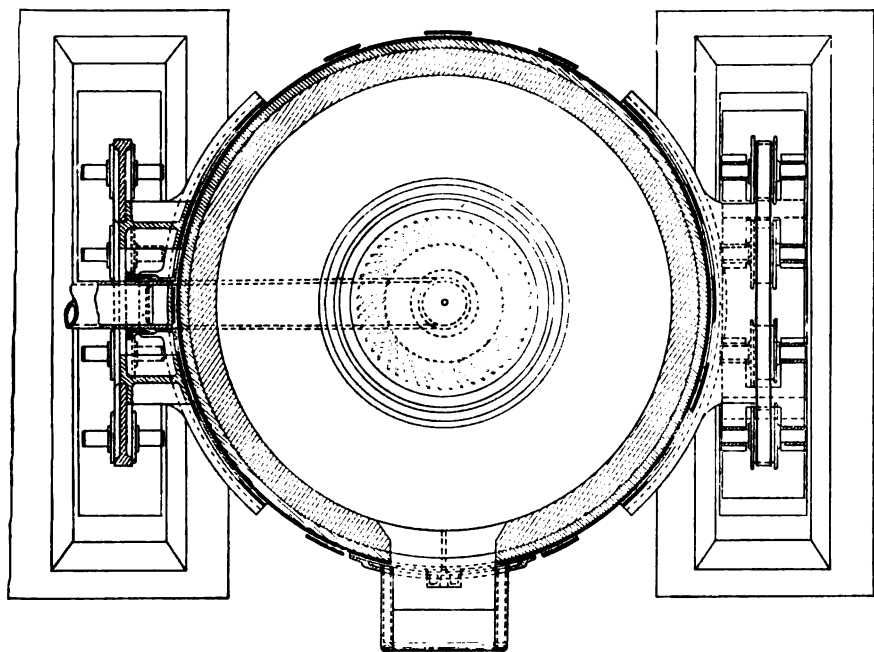
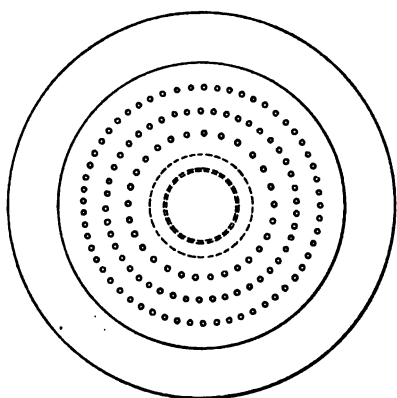


FIG. 6.—HORIZONTAL SECTION THROUGH SLAG MOUTH OF CONVERTER SHOWN IN FIG. 5.

The shell is lined with magnesite bricks, preferably 18 in. long by 6 in. thick, pressed in segments conforming to the inside diameter of the lined converter. The bricks are to be laid in a mortar made of finely ground magnesite and about 10 per cent. of anhydrous tar. A 4-in. space, left between the bricks and the shell, is filled with tightly rammed brasque. This back filling goes on while the brick lining is carried up, the ramming being uniform, so as to tighten the arch bricks equally.

The most radical departure from present construction is the proposed use of a large number of tuyères of small cross-sectional area, and confined to a central area in the converter bottom. As will be shown later, it is



TUYÈRE BLOCK, SHOWING ARRANGEMENT OF TUYÈRES.

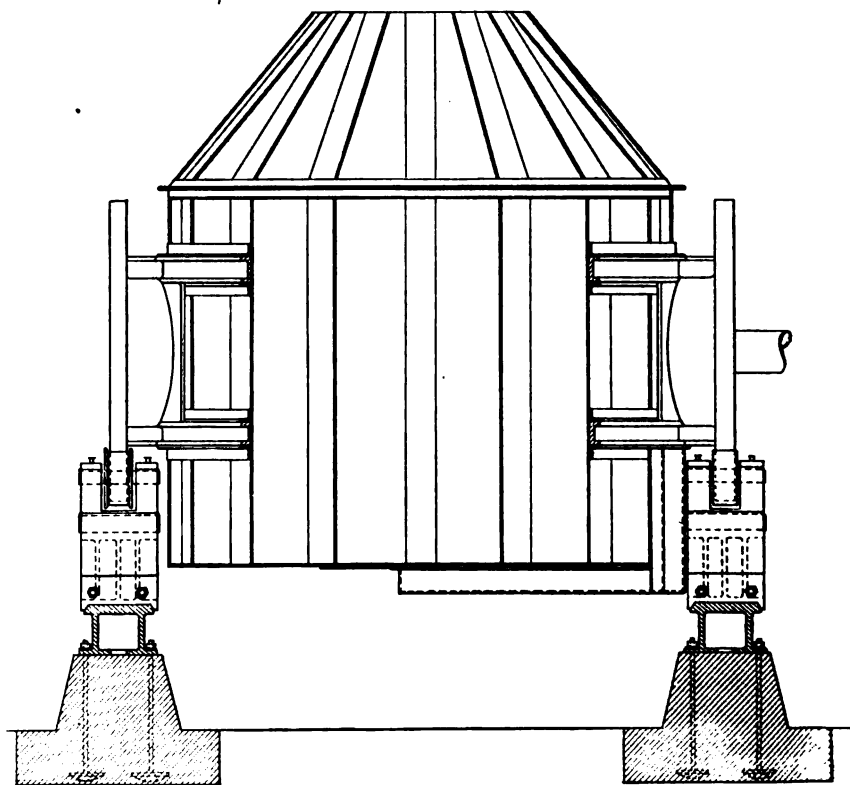


FIG. 7.—REAR VIEW OF CONVERTER OF FIGS. 5 AND 6, SHOWING ATTACHMENT OF RIDING RINGS.

entirely feasible to blow "white metal" to blister copper in a bottom-blown converter, provided the charge is large enough; hence it is not essential that this tuyère bottom should be raised above the surrounding non-perforated converter bottom; but the use of such a raised bottom offers some decided advantages. It is self-evident that this raised cone occupies only about one-fourth to one-third of the total interior converter area, so that it reduces for any height up to the top of this cone the total storage capacity of the converter vessel only a like amount, approximately. Above the cone or tuyère bottom, the entire interior area is available for storing or holding matte. In the annular sump, therefore, many tons of matte can be stored. When blowing, the converter is of course maintained in a vertical position, and the height of the matte column above the tuyère orifices can be kept shallow, requiring only a moderate air pressure. Successive charges of matte are added, and the slag is either continuously or intermittently discharged without interrupting the blowing. For intermittent slag discharge the converter has to be tilted down but slightly, to lower the slag discharge enough for skimming; and the difference in height of the molten material above the tuyères due to this tilting is imperceptible by reason of the smaller diameter of the tuyère bottom. When a sufficient amount of white metal has been accumulated, this is blown to blister copper, after the slag has been skimmed.

V. SMALL TUYÈRES FOR COPPER CONVERTERS

The subdivision of converter tuyères appears to be advantageous on chemical grounds, since increased contact surface between the molten sulphides and the oxygen in the blast will accelerate the burning process. When the air is concentrated in a relatively small number of large air currents, a portion of the air may be forced through the liquid metal before the oxygen could come in contact with the impurities to be burned off. It will take a longer time for a large air current to get diffused through the matte, than for a thin stream of air. The ideal way of introducing the air into the molten sulphides would be as a fine spray, distributed over a large surface, but this is obviously impractical for physical reasons. To prevent excessive power losses the tuyères must be large enough not to restrict unnecessarily the flow of the air. I believe, however, that the present practice of using 1½-in. to 2½-in. tuyères is going to extremes; and it appears that the limit of the air volume that can be forced into the present side-blown converters has already been reached in the Great Falls converter, since free oxygen is found in the converter gases. If the converter is to be applied successfully to the smelting of pyrite ores, 1,000 or even 1,500 kg. of air per minute should be supplied continuously to secure the necessary smelting speed.

Comparing the two largest side-blown converters, that of the Cana-

dian Copper Co.,³ which is a 37 ft. 2 in. by 10 ft. Peirce-Smith converter, with 44 tuyères, 1.25 in. in diameter, which are stated to be 7 in. apart, and that of the Great Falls reduction works, 20 ft. in diameter, with 62 tuyères, 1.75 in. in diameter, we have not only a relatively small number of air conduits of large cross-sectional area but these are all in close proximity to each other as well, so that they impinge only on a restricted matte surface. In my converter, in lieu of so few tuyères of large cross-section I would use from 250 to 400 (according to the supply of air) of small area. These would be distributed evenly over an area of from 30 to 50 sq. ft., so that the entering air would be divided over a correspondingly

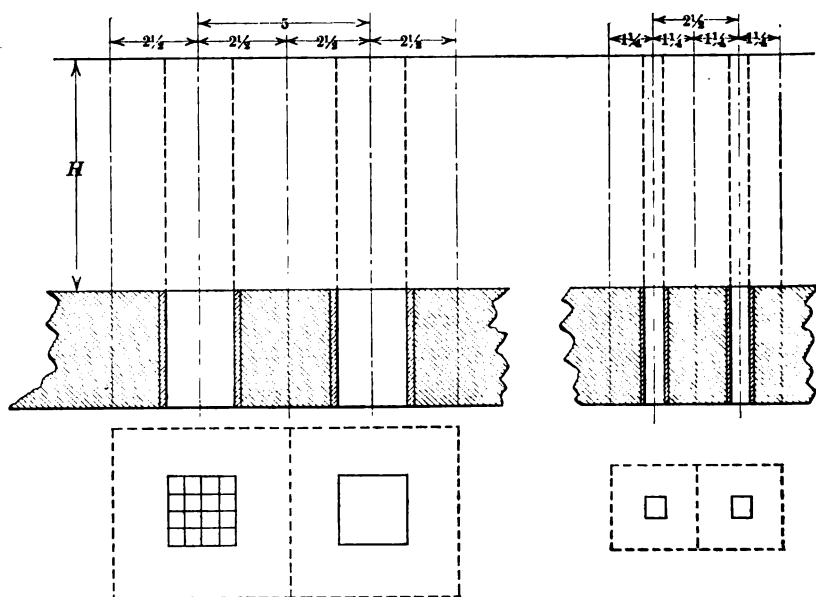


FIG. 8.—DIAGRAM SHOWING DIFFERENCE IN CONTACT SURFACE FURNISHED BY LARGE AND SMALL TUYÈRES.

large matte surface and would rise through a shallow matte column, bounded by a relatively quiescent "ring" of matte.

Fig. 8 illustrates in a general way the difference in contact surface offered by the air and matte with small and large tuyères. For clearness of illustration, square tuyères rather than circular ones have been chosen. In one case we have two 0.5 in. tuyères, surrounded by a square of matte 2.5 by 2.5 in. = 6.25 sq. in.; in the other case, two 2-in. tuyères, surrounded by a square of matte 5 by 5 in. = 25 sq. in. Assuming that 2.5

³ D. H. Browne: Recent Improvements and Additions to the Smelting Plant of the Canadian Copper Co. *Journal of the Canadian Mining Institute*, vol. xv (1912), through *Mining and Scientific Press*, vol. cvi, No. 1, p. 54 (Jan. 4, 1913).

kg. of air is delivered through the small tuyères and 40 kg. through the large ones, and that the matte column is 10 in. high, we would have the following conditions:

For 0.5-in.-tuyères.	For 2-in.-tuyères.
Tuyère area 0.25 sq. in.	4 sq. in.
Superimposed matte area 6.25 sq. in.	25 sq. in.
Ratio, $\frac{\text{matte area}}{\text{tuyère area}} = \frac{6.25}{0.25} = 25$	$\frac{25}{4} = 6.25$
Matte column = 6.25 by 10 = 62.5 cu. in.	25 by 10 = 250 cu. in.
Weight of matte column, at 0.08 kg. per cu. in. = 5 kg.	20 kg.
Ratio, $\frac{\text{matte volume}}{\text{tuyère area}} = \frac{62.5}{0.25} = 250$	$\frac{250}{4} = 62.50$
Ratio, $\frac{\text{weight of matte}}{\text{weight of air}} = \frac{5}{2.5} = 2$	$\frac{20}{40} = 0.5$

It will be seen that the contact surface of 16 0.5 in. square air currents projected into the fluid matte will be, for each inch length of such projection, 32 sq. in., whereas it is only 8 sq. in., or one-fourth as much, for the large tuyère. Likewise, the air is distributed and made to pass through a four-times greater weight of matte. This breaking up of the air into so large a number of streams of a moderate volume before it enters the molten sulphides, cannot be without marked effect on the velocity with which the oxygen can combine with the impurities, thus greatly accelerating the reactions. It will also prevent the throwing out of the converter charge, the excessive production of shot matte and shot copper, the fouling of the converter mouth, and the forming of accretions on it.

The prevention of noses and of that clogging of the tuyères which necessitates continued punching, and an increased blast pressure, is of even greater importance than an increased speed in the reactions. To heat 1 kg. of air to the temperature of the converter contents, $1,150^{\circ}\text{C.}$, requires in round numbers 300 cal. In the reaction $\text{FeS} + 3\text{O} = \text{FeO} + \text{SO}_2$, there are developed $-24,000 + 65,700 + 69,260$ (gas); total, 110,960 cal. or $\frac{110,960}{48} = 2,312$ cal. per 1 kg. of oxygen. To furnish this 1 kg. of oxygen, about 4.4 kg. of air have to be supplied, which, requiring 300 cal. per kilogram of air to be heated, would absorb about 1,320 cal.,⁴

⁴ To be exact, 0.67 kg. of oxygen of the 1 kg. in the 4.4 kg. of air enters into combination with sulphur, forming 1.34 kg. of SO_2 , the balance combining with the iron, forming protoxide of iron. We would therefore have, with a mean specific heat per kilogram of SO_2 of $0.125 + 0.0001 t$, or, at $1,150^{\circ}\text{C.}$, of 0.240 per kilogram, a heat content in 1 kg. of SO_2 at $1,150^{\circ}\text{C.}$ of 276 cal., and in 1.34 kg. SO_2 , therefore, 370 cal. The 3.4 kg. of nitrogen escaping unchanged as an inert gas would carry off, assuming a mean specific heat per kilogram of $0.2405 + 0.0000214 t$, or, at $1,150^{\circ}\text{C.}$, 0.26511 per

leaving not quite 1,000 cal. available for other heat requirements—in addition to the heat in the matte (about 250 cal. per kilogram of matte) and the heat of formation of the ferrous singulo-silicate.⁵

In the Thomas steel converter, the heat developed per kilogram of oxygen introduced is nearly double that developed per kilogram of oxygen in the copper converter, and after the air has been heated, there is also about 2,000 cal. available per kilogram of oxygen introduced. The high specific heats and high melting points of steel and of the extremely basic lime-silicate slag, require very much higher temperatures, and larger heat supplies, so that the net heat surplus is not so large as it at first appears. I have compared the two processes to see in what they differ, and why it should not be practicable to do away with “punching” the copper converter.

The $\frac{2,312}{4.4} = 525$ cal. developed per kilogram of air introduced into the copper converter, of which, as has been explained, 300 cal. are used to heat this air itself, leaving 225 cal. available for other requirements, can only be counted on *after* the 0.23 kg. oxygen in that kilogram of air has actually come in contact with the iron sulphide and oxidized the iron and sulphur. Reverting again to our comparison of the 0.5-in. tuyères with the 2-in. tuyères, we have in one case a matte prism 10 in. high, weighing 5 kg., in the other case a prism of the same height, weighing 20 kg., superimposed over each tuyère. Columns, say, only 1 in. high, would therefore weigh respectively 0.5 and 2 kg. With a heat content of 250 cal. per kilogram of matte, we would have heat contents respectively of 125 and 500 cal. in each 1-in. matte column. Introducing in one case 2.5 kg. of air per minute, we find that 300×2.5 , or 750 cal. of heat, have to be transmitted per minute, or 12.5 cal. per second, to heat this air. In the other case, 40 kg. of air will require 12,000 cal. per minute,

kilogram, $3.4 \times 305 = 1,037$ cal. We have therefore a total heat absorption by the waste gases of

$$\begin{array}{r} 370 \text{ cal. in SO}_2 \\ 1,037 \text{ cal. in N}^2 \\ \hline 1,407 \text{ cal.,} \end{array}$$

resulting from the introduction of 1 kg. of oxygen (4.4 kg. air, in round numbers). It will therefore be seen that 87 cal. more heat has to be supplied than has been credited above, or about 20 cal. per kilogram of air. Hence, 320 cal. per kilogram of air introduced is a more exact measure of the heat absorption. But for purposes of showing how much heat is needed *solely* for heating the air to 1,150° C., and the influence of a large concentrated air volume per tuyère on the formation of noses, the simpler method of calculation used above has been chosen.

⁵ In the preceding calculations and those to follow I have used the physical constants as given in Landolt-Börnstein's *Physikalisch-chemische Tabellen*, 4th ed. (Julius Springer, Berlin), as well as those given by J. W. Richards, in *Metallurgical Calculations*. The heat unit employed, as will be easily seen, is the French kilogram-calorie.

or 200 cal. per second. It will be seen that the matte column above the small tuyère will have to transmit but 10 per cent. of its heat per second, to the entering cold air, whereas that above the 2-in. tuyères would have to transmit 40 per cent. This example is not an exaggeration. In the large 20-ft. Great Falls converter with 2.25-in. tuyères, nearly 40 kg. of air per tuyère per minute is introduced. It is quite apparent that the matte cannot transmit a sufficient amount of heat fast enough to heat such a large volume of air *at its point of entry*, and that the matte becomes chilled, and noses are formed around the tuyère orifice.

During the second stage of the process, when blowing on "white metal," the conditions are indeed very much less favorable; since during this stage more heat is carried away by the waste gases than is generated by the burning of the sulphur to dioxide. This deficiency in heat must be made up from a surplus stored in the "white metal" during the first stage of the blow. This critical stage in the copper converter process will be more fully discussed later.

According to the reaction $\text{Cu}_2\text{S} + 2\text{O} = \text{Cu}_2 + \text{SO}_2$, there are developed — 20,300 + 69,260 (gas), or only 48,960 cal., the chief source of heat, since the iron has been eliminated during the previous stage.

Thus only $\frac{48,960}{32} = 1,530$ cal. per kilogram of oxygen,

or $\frac{48,960}{140} = 350$ cal. per kilogram of air, are developed.

From this it would appear that there was a surplus of 50 cal. available per kilogram of air supplied to the converter.

But the SO_2 formed has a higher specific heat than the air (O and N), so that more than 300 cal. per kilogram of air is carried off in the waste gases. We have at $1,150^\circ \text{C}$. a mean specific heat for SO_2 of $0.36 + (0.0003 \times 1,150) = 0.705$ per cubic meter, and a heat content of $0.705 \times 1,150 = 811$ cal., and for nitrogen a mean specific heat of $0.303 + (0.000027 \times 1,150) = 0.334$ and a heat content per cubic meter of $0.334 \times 1,150 = 384$ cal.

The amount of heat carried off in the waste gases per kilogram of oxygen introduced is therefore:

$$\text{In } \text{SO}_2 \quad \frac{2}{2.88} \times 811 = 563 \text{ cal.}$$

$$\text{In } \text{N} \quad \frac{3.4}{1.26} \times 384 = \frac{1,036}{1,599} \text{ cal. per kilogram of oxygen.}$$

We have seen that only 1,530 cal. per kilogram of oxygen is developed; so that 69 cal. will have to be supplied from the heat stored in the white metal. Expressed in calories per kilogram of air introduced, we have

$\frac{1,530}{4.4} = 348$ cal. per kilogram of air developed,⁶ and the waste gases carry off $\frac{1,599}{4.4} = 364$ cal. per kilogram of air introduced.

It is thus evident that a concentrated air stream not only decreases the speed of the chemical reactions, but also has a pronounced chilling effect. This chilling in itself will decrease the velocity of the chemical process. "The law that the velocity of chemical processes, *i.e.*, the ratio of the amount transformed to the time required, rapidly increases with rising temperature, is quite universal, and is valid for chemical processes of all kinds."⁷ The two conditions go hand in hand, as must be quite evident. With the smaller volume of air introduced, a relatively larger contact surface between air and sulphide, therefore increased mass action, is obtained, so that this air is not only more readily heated, but there is also secured at the very tuyère orifice a relatively larger proportion of the total heat from the oxidation of the sulphide. This is further augmented in my converter by the novel method of circulating the matte. With the tuyère bottom restricted to a central portion of the converter, the air-penetrated central matte column will be less dense than the matte surrounding it, which consequently will flow toward it. A portion of the gas-containing matte will ascend with the large volume of gas, overflowing at the surface and spreading toward the shell of the converter.⁸

There is therefore a constant and automatic migration of molten sulphide with its contained heat toward the tuyère bottom, when it comes in contact with the air. Thus the transmission of heat through the matte is aided by an actual flow of fresh matte. In the side-blown converter, where the air rises almost immediately after it leaves the tuyères, it is bounded on one side by the converter wall, and the fresh matte supply comes toward the tuyères from the opposite side only.

It should be plain that to prevent noses and the clogging of tuyères, by avoiding the chilling of the matte, in the manner here discussed, namely, by the use of a large number of small tuyères, it is not necessary to operate the converter any hotter than is now customary. The action would be quite automatic; because the air supply per tuyère has a definite relation to the amount of heat that can be transmitted by the matte immediately surrounding each focus of combustion.

⁶ This should be 350 cal. (see page 1089) but by neglecting more than one decimal figure and using rounded figures, slight inaccuracies result, but which do not affect the results appreciably, these errors not exceeding those introduced with other factors.

⁷ *The Principles of Inorganic Chemistry*, by Wilhelm Ostwald, transl. by Alex. Findlay, 2d ed., p. 66 (Macmillan and Co., 1904).

⁸ See my discussion of the paper of Messrs. Wheeler and Krejci, Great Falls Converter Practice, in *Bulletin* No. 83, Nov., 1913, p. 2648 *ff.*, for a more extensive explanation of the above phenomenon.

The longer life of a magnesite tuyère bottom in a copper converter than in a Thomas steel converter will permit the use of smooth tuyère pipes of either steel or copper, and materially reduce frictional resistance. These could be enlarged at the point of entry. As explained elsewhere⁹ the principal frictional losses are largely due to the partial closing and roughness of the tuyère orifices by adhering frozen matte and not so much to pipe-surface friction. With such wide distribution of the air, it is quite likely that the blowing can be done on shallow matte columns, and hence at low air pressures, without diminution of efficiency.

VI. EXAMPLE OF BLOWING A CHARGE IN A CONVERTER WITH AN ELEVATED TUYÈRE BOTTOM

To illustrate the working of the converter with the elevated tuyère bottom, and the effect of the latter on the static heads, I have prepared Table V, which is based on the following assumptions:

It is assumed that the matte has the composition, Cu, 30; Fe, 40; and S, 30 per cent., that is, it contains sufficient sulphur to form Cu_2S and FeS with all the copper and iron present; that at no time during the blowing of the matte to white metal is there a deficiency in sulphur, so that no metallic copper would be produced during the first stage of the process, and the air supply to the converter could be assumed as constant, conforming to the reaction, $\text{FeS} + 3\text{O} = \text{FeO} + \text{SO}_2$. For every pound of oxygen forced into the converter, there will be oxidized, according to this formula, 1.167 lb. of iron, and 0.67 lb. of sulphur, with the ratio of $\frac{1.167 \text{ Fe}}{0.67 \text{ S}} = 1.75$. During the second stage, when blowing the white metal to copper, 1 lb. of oxygen will oxidize 1 lb. of sulphur, according to the formula $\text{Cu}_2\text{S} + 2\text{O} = 2\text{Cu} + \text{SO}_2$.

The rate of conversion in Table V is based on an air supply of 200 kg. of oxygen, corresponding to some 870 kg. of air (about 680 cu. m., or 24,000 cu. ft.) per minute. This weight of oxygen would oxidize 234 kg. of iron to some 300 kg. of FeO per minute, forming about 425 kg. of slag of the composition $2\text{FeO} \cdot \text{SiO}_2$, and 133.4 kg. of sulphur to 266.8 kg. of SO_2 . In 30 min., this would correspond to an elimination of 7 (metric) tons of iron and 4 tons of sulphur. In a like period 13.2 tons of slag would be produced, which augmented by 0.3 ton of copper, going into the slag, would bring the total slag to 13.5 tons, containing nearly $2\frac{1}{2}$ per cent. Cu. The specific gravity of this slag has been assumed to be 3.5.

The blow is started with an initial charge of 60 (metric) tons of matte, of 4.2 sp. gr., 7.1 cu. m. (about 250 cu. ft.) of which would fill the annular sump up to the tuyère orifices, and 7.2 cu. m. (about 255 cu. ft.) would be above the tuyère block. It will be seen that instead of blowing against

⁹ *Bulletin* No. 83, Nov., 1913, p. 2656 ff.

Converter with an Elevated Tuyère Bottom

Calculated Matte Composition after each Blow			Calculated Specific Gravity of Molten Matte	Total Height of Matte and Slag, Mm.	Total Static "Head" of Matte and Slag, Kg. Cm. ²	Total Height of Matte and Slag, In.	Total Static "Head" of Matte and Slag, Lb. per Sq. In.	Tons Matte Remaining in Con.	Tons Slag Skm'd	Total Tons Slag Skimmed	Air Pressure 3 X Static "Head"	Air Pressure 2 X Static "Head"
Cu. %	Fe. %	S. %										
30.0	40.0	30.0	4.2	386	0.163 Matte	15.2 only	2.32 Matte	only			6.96	4.64
36.0	35.0	29.0	4.3	436	0.1711	17.2	2.44	48.7	13.5		7.32	4.88
				484	0.206	19.125	2.93	68.7			8.79	5.86
41.0	31.0	28.0	4.4	538	0.218	21.2	3.11	57.4	13.5	27.0	9.33	6.22
			4.2	586	0.253	23.125	3.60	77.4			10.80	7.20
44.0	29.0	27.0	4.5	617	0.257	24.2	3.66	66.10	13.5	40.5	10.98	7.32
52.5	22.0	25.5	4.65	461	0.191	18.2	2.71	54.80	13.5	54.0	8.13	5.42
65.5	11.5	23.0	4.9	304	0.1401	12.2	1.74	43.5	13.5	67.5	5.22	3.48
			4.2	352	0.155	14.125	2.23	63.5			6.69	4.46
65.5	11.5	23.0	4.9	399	0.167	15.7	2.36	52.2	13.5	81.0	7.08	4.72
			4.2	447	0.202	17.625	2.85	72.2			8.55	5.70
65.5	11.5	23.0	4.9	495	0.214	19.5	3.05	60.9	13.5	94.5	9.15	6.10
			4.2	543	0.248	21.43	3.54	80.9			10.62	7.08
65.5	11.5	23.0	4.9	589	0.260	23.2	3.70	69.6	13.5	108.0	11.10	7.40
78.0	2.0	20.0	5.3	418	0.184	16.5	2.62	58.3	13.5	121.5	7.86	5.24
			4.2	466	0.219	18.43	3.11	78.3			9.33	6.22
76.0	3.0	21.0	5.2	518	0.234	20.5	3.34	67.0	13.5	135.0	10.02	6.68
			4.2	566	0.269	22.43	3.83	87.0			11.49	7.66
75.0	4.0	21.0	5.2	615	0.285	24.20	4.05	75.7	13.5	148.5	12.15	8.10
			4.2	663	0.320	26.13	4.54	95.7			13.62	9.08
74.0	5.0	21.0	5.2	645	0.300	25.45	4.27	84.4	13.5	162.0	12.81	8.54
			4.2	693	0.335	27.38	4.76	104.4			14.28	9.52
73.0	5.4	21.6	5.15	794	0.375	31.20	5.59	93.1	13.5	175.5	16.77	11.18
81.0		19.0	5.5	432	0.238	17.00	3.40	83.2			10.20	*6.80
84.0		16.0	6.0	342	0.205	13.50	2.92	80.8			8.76	5.84
86.5		13.5	6.4	275	0.176	11.00	2.51	78.3			7.53	5.02
89.0		11.0	6.8	217	0.150	8.50	2.13	75.9			6.39	4.26
92.0		8.0	7.3	161	0.118	6.4	1.68	73.4			5.04	3.36
95.0		5.0	7.8	114	0.089	4.4	1.75	71.0			3.75	2.50
100.0			8.5	54	0.046	2.2	0.66	67.5			1.98	1.32

* Blowing on "white metal" with 120 kg. O per min., about 15,500 cu. ft. air per min.

a height of $\frac{14.3 \text{ cu. m.}}{18.6 \text{ sq. m.}} = 0.770 \text{ m. (18.33 in.)}$, and having to counter-balance a static head of $\frac{0.770 \times 4.2}{10} = 0.323 \text{ kg. per square centimeter}$ (4.6 lb. per square inch), we only have to blow against 0.386 m. (15.2 in.) of matte corresponding to a static head of 0.163 kg. per square centimeter (2.32 lb. per square inch). After 30 min. of blowing, some 13.5 tons of slag (sp. gr. 3.5) will have been formed, occupying some 3.86 cu. m.; and, since 18.6 sq. m. (200 sq. ft.) is the area of the converter inside of the lining (diameter 16 ft., or 4.877 m.), this slag would have a height of $\frac{3.86}{18.6} = 0.208 \text{ m.}$ The pressure due to this static head is $\frac{0.208 \times 3.5}{10} = 0.073 \text{ kg. per square centimeter (1.04 lb. per square inch)}$.

During the same 30 min., the elimination of 4 metric tons of sulphur and 7 tons of iron (which have gone into the slag) has caused the matte to shrink in volume, and increase slightly in specific gravity, so that its depth above the tuyère bottom is now only some 0.228 m. (9 in.), exerting a static pressure of 0.098 kg. per square centimeter (1.40 lb. per square inch). But there is also above it a column of slag, 0.208 m. (8.2 in.) high, which exerts a pressure of 0.073 kg. per square centimeter (1.04 lb. per square inch); so that, before the slag is skimmed, we have the combined head due to matte and slag of 0.171 kg. per square centimeter (2.44 lb. per square inch). After skimming the slag, 20 (metric) tons of fresh matte are added, which increases the matte column from 0.228 m. to 0.484 m. (19 in.), and the pressure from 0.098 kg. to 0.2056 kg. per square centimeter.

During the blow on white metal, the oxygen supply is lowered to 120 kg. per minute, eliminating only 2.4 tons of sulphur in 20 min. With the rapid reduction of the copper during this stage, the specific weight of the metal increases quickly, bringing about a rapid shrinking of the bath, so that the static pressure drops rapidly, and is only 0.66 lb. per square inch at the finish of the blow.

When skimming the slag, the converter is tilted but slightly, the slag (about 140 cu. ft. after 30 min.) being discharged through the slag opening on the side. Blowing is not stopped while the slag is being discharged. To do this quickly, the slag opening is large (4 ft. wide and about 18 in. high), so that, with the slag occupying a space of 4 ft. by 0.25 ft. = 1 sq. ft. in the slag opening, it would have to be discharged at the rate of about 1.2 ft. per second to complete skimming in 2 min. In the last two columns, the pressure due to static head has been multiplied by 3 and 2, the mean of which would about correspond to the actual air pressure (gauge pressure) required in operating the charges. As will be discussed later, for the higher columns of liquid the ratio between working air pressure and static head will be more nearly 2, and for the low columns

more nearly 3. This is due to the fact that the various frictional resistances offered by the matte to the air by the tuyère pipe surface, roughness of the orifice, etc., are about the same, regardless of the height of the matte column, so that with an increased height of liquid the pressure must only be increased about an amount corresponding to the increase in static head. Frictional losses, as such, will be lessened with an increase in air pressure.

VII. THE INFLUENCE OF METALLIC COPPER IN MATTES ON THE RATE OF ELIMINATION OF SULPHUR IN CONVERTING

To satisfy myself that the elimination of the sulphur and the iron in the hypothetical Table V conforms to actual converter practice, I compared the iron and sulphur ratios (copper = 100) calculated from this table with those given in "Table III, Recalculation of Mathewson's Data, given in his Table III," by D. H. Browne, *Trans.* xli, p. 312 (1911), which are based on samples taken while blowing a charge of matte, and are therefore not hypothetical. Both are plotted together in Fig. 9, the curves based on the Mathewson-Browne data being marked *M*.

It will be seen that, notwithstanding my assumption of an ideal matte, the iron elimination in this hypothetical blow corresponds very closely to that in actual converter practice, whereas a striking difference appears in the sulphur curve. This difference is due to sulphur deficiency in Mathewson's matte, showing that metallic copper was present in it. A striking characteristic of the *M* sulphur curve is its flattening as the "white metal" stage is reached, showing that there is a decrease in the rate of elimination of the sulphur, before blowing on "white metal" begins, and that it approaches the ideal matte $x\text{Cu}_2\text{S}$, $y\text{FeS}$ at that stage of the blow. This led me to seek further for a satisfactory explanation. The hypothetical curves illustrate the conversion process according to the formula $x\text{Cu}_2\text{S}$, $y\text{FeS} + 3\text{O} = x\text{Cu}_2\text{S} + y\text{FeO} + \text{SO}_2$, in which ideal matte the cupric sulphide is not acted upon until all iron and the sulphur combined with it have been eliminated, when the dissociation of the cuprous sulphide and the oxidation of its sulphur begin.¹⁰

As is well known, mattes do not conform to this ideal composition, but carry varying amounts of metallic copper in solution or mechanical suspension. The amount of metallic copper a matte can thus hold is influenced by the amount of ferrous sulphide in the matte—the metallic

¹⁰ In Table V, on which the curves in Fig. 9 are based, I did not attempt to avoid slight inaccuracies, due to assuming round numbers for the atomic weights (Cu, 64; S, 32; O, 16; Fe, 56); to a percental composition of 30 Cu, 40 Fe, 30 S, which does not quite, though nearly enough (see Table VII, columns 1, 5 and 7, for 30 per cent. copper matte), correspond to the form $x\text{Cu}_2\text{S}$, $y\text{FeS}$; and to the deduction of a portion of the copper (0.3 ton) which goes into the slag. To do so would have made the calculations unnecessarily involved and the presentation of the subject less clear.

copper decreasing with a decrease in ferrous sulphide and an increase in cuprous sulphide.

Under certain conditions metallic iron decomposes cuprous sulphide according to the reaction $\text{Cu}_2\text{S} + \text{Fe} = \text{FeS} + 2\text{Cu}$.¹¹

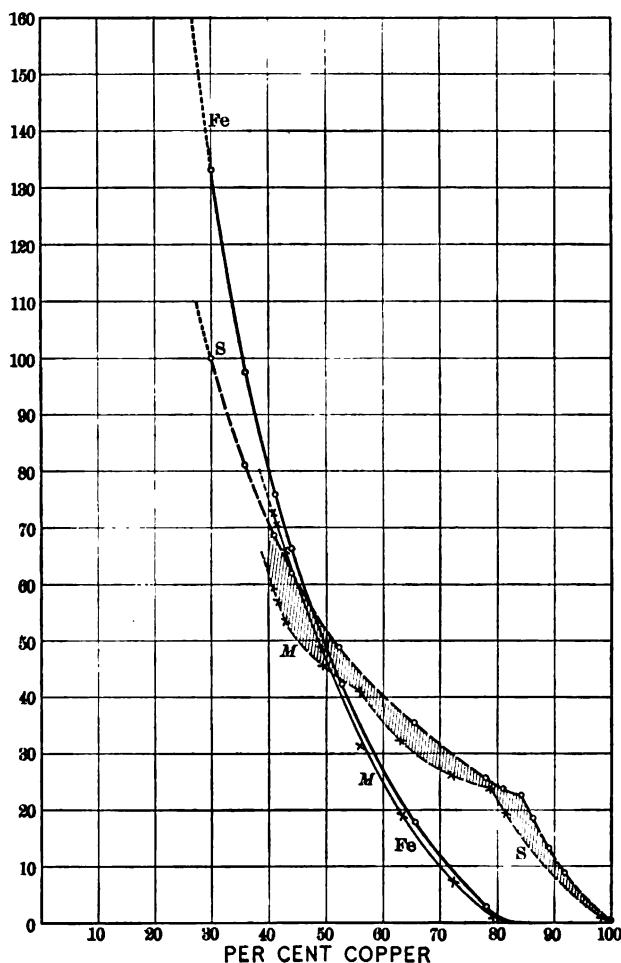


FIG. 9.—SULPHUR AND IRON RATIOS. COPPER = 100.

By a study of the researches of Gibb and Philp and Paul Roentgen,¹² as well as of the chapter on mattes by Peters,¹³ and the papers of Van

¹¹ Constitution of Mattes Produced in Copper Smelting, by Gibb and Philp. *Trans.*, xxxvi, p. 669 (1905).

¹² Zur Kenntnis der Natur des Kupfersteines (Paul Roentgen). *Metallurgie*, vol. iii, pp. 479 to 487 (1906).

¹³ *Principles of Copper Smelting*.

Liew,¹⁴ Mathewson,¹⁵ and Browne,¹⁶ I arrived at the conclusion that this reaction can be read in either direction—i.e., is a reversible reaction; the metallic copper, under certain conditions, being also capable of decomposing the ferrous sulphide, and combining with the sulphur.¹⁷

Gibb and Philp, and Roentgen, found that with an increasing proportion of FeS the separation of metallic copper grew larger, whereas mattes with less than 10 per cent. FeS may or may not contain metallic copper in mechanical suspension.

"Nor will Cu_2S dissolve an excess of either of its constituents Cu or S, when matte contains less than 10 per cent. FeS."¹⁸

This is confirmed by Mathewson's data. Calculating the amount of FeS and Cu_2S that could be formed with the sulphur present, I ascertained the amount of metallic copper in that matte, from which the metallic-copper ratios, with $\text{Cu}_2\text{S} = 100$ as a basis, were figured. Taking as a basis his Table III,¹⁹ I obtained the results shown in my Table VI.

In this table, the gradual reduction of metallic copper with a decrease of FeS is quite noticeable, until practically all FeS has been eliminated.

Browne says:

"A curious point of resemblance is the little jog or reversal in the sulphur-curve, which occurs when the iron is reduced to 5 per cent. All the curves of individual blows show this. The only explanation I can offer is that at this point a certain amount of metallic copper or copper-nickel separates out as metal, and, falling to the bottom of the converter, leaves the sulphur with which it was united to be absorbed by the over-

¹⁴ Trans., xxxiv, 418 (1903).

¹⁵ Trans., xxxviii, 154 (1907).

¹⁶ Trans., xli, 296 (1910).

¹⁷ The work of Bornemann and Schreyer, *Das System Cu_2S -FeS* (*Metallurgie*, vol. vi, p. 619, 1909) agrees in most of its results with that of Roentgen, and in the main with that of Hofman, Cayless and Harrington, *The Constitution of Ferro-Cuprous Sulphides*, Trans., xxxviii, 142, 1907). But it should be borne in mind that these investigators usually start with a variety of mixtures having different proportions of Cu_2S and FeS, and speculate on the changes taking place when such mixtures are melted and cooled, and on the nature of the compounds formed, to account for the separation of metallic copper from such mixtures, or solutions. In practice, the mattes converted usually lack in sulphur, so that either the copper or the iron cannot be present as Cu_2S and FeS. Nothing points to the presence of metallic iron in the commercial grade of matte. The fact that all of these investigators agree on the separation or the presence of metallic copper, which increases with increasing content of FeS to a certain point, and then decreases, justifies the assumption that the matte sent to the converters holds metallic copper either in solution or suspension. Bornemann and Schreyer found that already with 8 molecular per cent. of FeS, metallic copper began to separate out. With increasing content of FeS the metallic copper separated out increased and was at its maximum between 33 and 35.7 molecular per cent. of FeS, then the metallic copper decreased gradually with a further increase in FeS, was very small with 66 molecular per cent., and with 79 molecular per cent. FeS was hardly discernible.

¹⁸ Peters, *op. cit.*, p. 402.

¹⁹ Trans., xli, 312 (1910).

lying matte, which makes the matte at this point higher in sulphur than it was before the metal separated out."²⁰

TABLE VI.—*Recalculation of the Data in Mathewson's Table III*

Relative Elimination of Iron, Sulphur, and Arsenic in Bessemerizing Copper-Mattes, *Trans.*, xxxviii, 154 to 161 (1908). Taken from David H. Browne's Paper: The Behavior of Copper-Matte and Copper-Nickel Matte in the Bessemer Converter, *Trans.*, xli, 296 to 317 (1910) at top of p. 312.

Composition			Calculated Composition			Ratio of Met. Cu with Cu ₂ S = 100
Cu Per Cent.	Fe Per Cent.	S Per Cent.	Cu ₂ S Per Cent.	FeS Per Cent.	Met. Cu. Per Cent.	Per Cent.
40.68	29.60	24.30	37.00'	46.50	11.08	30.0
42.50	27.80	22.70	34.00	43.70	15.30	45.0
49.16	23.70	22.60	45.50	37.20	12.76	28.0
55.86	17.50	22.90	59.50	27.50	8.26	16.7
62.97	12.10	20.40	67.50	19.00	8.97	13.3
72.06	5.30	19.10	79.00	8.60	8.86	11.2
78.96	0.90	18.70	90.95	1.41	6.20	8.3
81.32	0.40	15.90	78.35	0.63	18.64	23.2

I believe that, in the converter, a portion of the sulphur that is combined with the iron, instead of being oxidized to SO₂, combines with the metallic copper to form Cu₂S, increasing the amount of the latter and decreasing that of SO₂ (which is the measure of sulphur elimination) until nearly all the iron has been eliminated.

At the high temperatures used in metallurgy, a reversal of reactions frequently occurs. Decomposition of SiO₂ and re-absorption of the silicon by the steel takes place in the Thomas converter, and this is true, to an even greater extent, of manganese oxide (MnO), which is re-absorbed as Mn. A well-known example is the oxidation of heated iron to an oxide by water vapor, with the formation of hydrogen. Hydrogen may likewise reduce the iron from its oxide, metallic iron and water being formed. These phenomena are due to mass action.

"Not only the nature, and say, the temperature, are the determining factors for the occurrence of a chemical process, but also the ratio of the substances present to the given volume or the concentration, as well."²¹

This would also explain why in practice the ratio of elimination of $\frac{\text{Fe}}{\text{S}}$ is more nearly 2, instead of 1.75 (see p. 1091). A portion of the FeS sulphur enters into combination with the metallic copper, to be eliminated

²⁰ *Loc. cit.*, p. 304.

²¹ Wilhelm Ostwald, *op. cit.*, p. 100.

during the second stage, in the blow on "white metal." There is, at times, also a certain amount of iron already oxidized introduced with the siliceous flux, which increases the iron elimination.

The close agreement between the hypothetical curves and the *M* curves based on actual practice, demonstrates also the correctness of the reaction $\text{FeS} + 3\text{O} = \text{FeO} + \text{SO}_2$, with the exception above noted as caused by a sulphur deficiency, and the consequent presence of metallic copper in the matte.

VIII. AIR REQUIREMENTS FOR CONVERTING AND POWER NEEDED FOR COMPRESSING AIR

Table VII shows the theoretical and "commercial" or actual air requirements for converting. A slightly greater theoretical air weight is here indicated, since it is assumed that no metallic copper exists in the matte; *i.e.*, there is no deficiency in sulphur. The air weights given are also applicable to sulphide ores, since these have usually other constituents besides copper, iron, and sulphur, which would diminish the weight of air required per ton of ore. The "commercial" air consumption and horse-power-hours used in compressing that weight of air, as stated in columns 13 and 14, should be secured at all well-managed works, in converting the usual mattes. If they are not obtained, either there is something radically wrong with the blowing engines, or air is needlessly allowed to escape, or an excessive air pressure is used.

It should be noted that no allowance has been made for copper losses. It is assumed, for instance, that 100 tons of 1 per cent. "matte" produce 1 ton of copper.

Carr B. Neel²² says that the Peirce-Smith converters at the Garfield plant of the American Smelters' Securities Co. used from 4.86 to 4.94 tons of air, and from 107.5 to 109.8 h.p.-hr., per ton of copper. Since the mattes converted at that plant probably vary between 40 and 45 per cent. copper, it will be seen in columns 13 and 14 of Table VII that this represents good practice. It should not be forgotten that these converters received only from 6,000 to 9,000 cu. ft. of air per minute at that time, so that the chances for air losses were greater than if one converter received 30,000 cu. ft. of air per minute.

Figures which I have based on average yearly operations at one of the largest copper smelters, with the old-style acid-lined shells (96 by 126 in.), and a yearly production of 60,000 tons of copper, show an air consumption of 6.76 (short) tons and 155 h.p.-hr. per ton of copper. This small difference of only, say, 45 h.p.-hr. per ton of copper, amounts to 2.7 million h.p.-hr. per year, or 312 h.p.-years. At this particular plant,

²² *Engineering and Mining Journal*, vol. xci, No. 19, p. 964 *et seq.* (May 13, 1911).

the h.p.-year cost was, at that time, \$70, exclusive of interest, depreciation and amortization on the power equipment. The increased power cost amounted therefore to \$21,840 per annum.

TABLE VII.—*Theoretical and Commercial Air-Requirements for the Conversion of Matte of Various Compositions*

Composition of Matte							Oxygen and Air Required per Metric Ton of "Matte"		Air Required per Ton of Copper, Metric Ton	H.P.-hours, Adiabatic Compression	H.P.-hours, 80 Per Cent. Dynamic Efficiency	Commercial Air Requirements per Ton of Copper, Metric Ton	Commercial H.P.-hours Required per Ton of Copper
Cu, Per Cent.	S, Per Cent.	Cu ₂ S, Per Cent.	FeS, Per Cent.	Fe, Per Cent.	S, Per Cent.	Total S, Per Cent.	O, Kg.	Air, Kg.					
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)	(13)	(14)
1	0.25	1.25	98.75	62.85	35.90	36.15	541.5	2,360	236.000	3776.00	4720.00	295.000	5900.00
2	0.50	2.50	97.50	62.10	35.40	35.90	536.5	2,340	117.000	1872.00	2340.00	146.250	2925.00
3	0.75	3.75	96.25	61.25	35.00	35.75	532.5	2,320	77.333	1237.60	1547.00	96.667	1934.00
4	1.00	5.00	95.00	60.40	34.60	35.60	528.5	2,300	57.500	920.00	1150.00	71.875	1437.50
5	1.25	6.25	93.75	59.65	34.10	35.35	523.5	2,280	45.600	729.60	912.00	57.000	1140.00
10	2.50	12.50	87.50	55.67	31.81	34.33	502.3	2,180	21.800	348.80	436.00	27.250	545.00
15	3.75	18.75	81.25	51.69	29.56	33.31	481.1	2,100	14.000	224.00	280.00	17.500	350.00
20	5.00	25.00	75.00	47.71	27.29	32.29	458.9	2,000	10.000	160.00	200.00	12.500	250.00
25	6.25	31.25	68.75	43.73	25.02	31.27	437.7	1,900	7.600	121.60	152.00	9.500	190.00
30	7.50	37.50	62.50	39.75	22.75	30.25	416.5	1,810	6.033	96.53	120.66	7.540	150.80
35	8.75	43.75	56.25	35.78	20.47	29.22	394.2	1,740	4.977	79.63	99.54	6.221	123.90
40	10.00	50.00	50.00	31.81	18.19	28.19	372.9	1,620	4.050	64.80	81.00	5.062	101.30
45	11.25	56.25	43.75	27.84	15.91	27.16	351.1	1,530	3.400	54.40	68.00	4.250	85.00
50	12.50	62.50	37.50	23.86	13.64	26.14	329.4	1,430	2.860	45.76	57.20	3.575	71.50
55	13.75	68.75	31.25	19.88	11.37	25.12	308.2	1,340	2.440	39.04	48.80	3.050	61.00
60	15.00	75.00	25.00	15.91	9.09	24.09	286.4	1,250	2.080	33.28	41.60	2.600	52.00
65	16.25	81.25	18.75	11.93	6.82	23.07	264.7	1,150	1.770	28.32	35.40	2.213	44.30
70	17.50	87.50	12.50	7.95	4.55	22.05	243.5	1,060	1.520	24.32	30.40	1.900	38.00
75	18.75	93.75	6.25	3.97	2.28	21.03	221.7	960	1.280	20.48	25.60	1.600	32.00
80	20.00	100.00	0.00	0.00	0.00	20.00	200.0	870	1.090	17.44	21.80	1.363	27.30

Column 1 of Table VII gives the copper content of an ideal matte, i.e., one containing sufficient sulphur to permit all the copper to be present as Cu₂S, and all the iron as FeS. This composition, and the air requirements for it, are applicable to pyrite, FeS₂, also, since the second atom of sulphur is usually distilled off without being oxidized, the FeS₂ splitting up into FeS and S. Column 2 gives the sulphur that is in combination with the amount of copper stated in column 1; column 3, the amount of Cu₂S; column 4, the balance of FeS, to make up 100 per cent.; column 5, the iron; column 6, the sulphur combined with the iron, as stated in column 5; and column 7, the total sulphur combined with the copper and iron; column 8, the weight of oxygen, and column 9 that of air required per ton of "matte" of the percental composition stated in the previous columns.

In column 10 are given the metric tons of air required per metric ton of copper for mattes with varying copper content, corresponding to the

compositions stated in the first 7 columns. These are *theoretical* air requirements.

On the assumption of 1 atm. as initial or atmospheric pressure, and 1.65 atm. (9.25 lb. per square inch gauge pressure) as the working pressure required, the adiabatic compression of 1 cu. m. of air per hour will require theoretically, according to the well-known formula, about 0.02 h.p. On this basis, the column in Table VII, showing the theoretical horse-power-hours required per ton of copper, and the following column, allowing for the loss of power in the machinery (80 per cent. efficiency), have been calculated for each matte of the composition given. But since there are other losses, leaks, etc., a still larger volume of air must be compressed. Assuming that 80 per cent. of the amount indicated by engine displacement is actually delivered, we obtain the figures of air requirement and horse-power-hours given in the last two columns of the table. This represents the "commercial efficiency" attainable with efficient blowing equipment and careful operation. The punching of tuyères being entirely eliminated, and the leakage of air in every possible way prevented, this efficiency in the delivery of the air should be as high as 90 per cent., assuming complete utilization of the oxygen in the converter itself.

The air volume has been figured on the basis of 1.25 kg. as the weight of 1 cu. m. of air, so that 0.8 cu. m. = 1 kg. of air.

IX. COST OF POWER TO FURNISH AIR REQUIRED IN CONVERTING

The power for blowing being the chief item of expense, let us look into the cost of producing it, with efficient prime movers. With good steam turbines, in the sizes (upward of 1,000 h.p.) that would here come into question, and directly connected, as they would be in this case, to turbo blowers, we can count on a steam consumption of 12 lb. per effective horse-power-hour, and with a good boiler 1 lb. of coal will evaporate 8 lb. of water; so that with 1.5 lb. of coal an effective horse-power-hour is produced. With oil fuel, an evaporation of 14 lb. of water from 1 lb. of oil can be obtained in good average practice, so that not quite 0.9 lb. of oil would furnish one effective horse-power-hour. Where fuel is expensive, Diesel engines for operating the blowing engines may be advantageously used, since, with the favorable load-factor of a continuously operated blowing plant, they would generate, a horse-power-hour for less than 0.5c., with fuel oil costing \$1.50 per barrel (of 330 lb.), and including 12.5 per cent. interest and amortization on the capital invested.²³

At most large smelting works, power should be generated at a cost not

²³ See my article, Principles of the Diesel Oil Engine, *Engineering and Mining Journal*, vol. xcv, No. 17, p. 843 (Apr. 26, 1913).

exceeding 1c. per horse-power-hour, which we may use as a basis for calculating the cost of furnishing the air for converting. The horse-power-hours in column 14, Table VII, would then give us the cost in cents of furnishing the air per metric ton of copper.

Other power requirements, for tilting the converter, etc., would be very small items per ton of copper. The cost of charging the converter, with proper mechanical arrangements and equipment, should not exceed that of charging blast furnaces, nor should attendance for the converter be greater than for the blast furnace.

Moreover, we must credit the total operation of bessemerizing and smelting in the converter with the saving of the cost of the previous blast-furnace smelting which it renders unnecessary. In pyrite smelting about 2,000 cu. m. or 2.5 tons of air are used in the blast furnace (much less efficiently than in the converter) per ton of charge smelted. The power used to compress this very much larger amount of air, even at its lower pressure, is about one-half that used in converting an equal quantity of copper. Then there is the cost of coke, of which, in the average practice of pyrite smelting, scarcely less than 5 per cent. of the weight of the charge is used. At \$10 per ton, which is about the price ruling at many Western works, this item alone would be 50c. per ton of charge smelted.

The most serious grounds on which the practice of converter smelting may be condemned, are the metal losses. Very likely, it might be necessary to use a reverberatory furnace or reverberatory forehearth, to which all of the discarded converter slag would be run, and from which the entrained copper would be separated by the occasional addition of "green" ore. The low-grade matte thus made would be added to the converter charges. The reverberatory-furnace gases would be passed under waste-heat boilers, as is the customary practice now.

It should not be overlooked that the converter slag has a heat content of about 400 cal. per kilogram, or 400,000 cal. per metric ton. As the efficiency of the reverberatory furnace (not taking into consideration the recovery under boilers of the heat of waste gases) is about 25 per cent., this heat in the slag represents about 1,600,000 cal. of fuel heat expended in the reverberatory, or slightly more than 1 barrel of fuel oil, which contains 1,500,000 cal. (nearly 6,000,000 B.t.u.). Of course, the saving in fuel that would result by having the slag already molten would only be a portion of the above amount.

The investment required for a converter smelting plant would be considerably less than for a plant with MacDougall roasters, reverberatory furnaces and converter department, or blast furnaces and converter department.

Where the gases can be utilized commercially for the manufacture of sulphuric acid, they have the advantage of great concentration of SO_2 , without the admixture of carbon dioxide and monoxide.

X. THE FEEDING OF SULPHIDES THROUGH CONVERTER TUYÈRES

The radical differences between the converter-smelting and blast-furnace smelting processes indicate a method of charging sulphide ores into the converter unlike that which is practiced at blast furnaces. In the blast furnace, the shaft is filled with layers of ore, flux, and fuel, which gradually descend to the smelting zone, at a short distance above the tuyères. The melted portions of the charge trickle through portions not yet melted, and drop below the tuyère zone into the crucible, in which, or in a forehearth outside of the furnace, the slag and matte are separated. The charge above the tuyères acts as a screen, preventing excessive throwing out of fine material, and absorbs also a portion of the sensible heat of the gases.

In the converter we have a layer of slag on top of a layer of sulphide. If the sulphide ore is charged through the converter mouth, it must be melted and pass through the slag into the molten sulphide, before it can be acted upon by the ascending air currents. The charging of pulverulent ores and concentrates through the converter mouth must be condemned also on account of the unavoidable mechanical losses and large quantities of flue dust that would result. It is evident that the large volume of gases passing through the mouth at a velocity of more than 2,000 ft. per minute would carry small particles with it.

Since it is desired, not merely to *melt* the sulphides, but also to decompose them, eliminate iron and sulphur, and retain copper, their introduction in a finely pulverized condition at the very place where the chemical process takes place, namely, in the tuyère zone of the converter, appears to offer many advantages. The process would not differ in principle from that of using pulverized coal, blown into a furnace for firing purposes, or into a slagging producer for generating gas, except that the material would be blown into a molten medium and become absorbed by it.

The small particles of sulphide ore would be melted very much more readily and rapidly, and being in intimate contact with air (oxygen), their iron and sulphur would at once begin to burn and thus furnish at least a portion of the heat absorbed in their fusion. The great velocity of this chemical process would warrant the expense of finely grinding coarse ore, so that the fines could be blown into the converter. A much greater mass of oxygen can take part in the burning process if the particles are in a state of fine subdivision; and, by reason of the greatly increased mass of the reacting elements, the speed of the reactions would be greatly increased. Not only the increased mass action, but the fact that it would be conducted at a relatively high temperature, would augment the velocity of the chemical process. It would be a simultaneous roasting, smelting, and converting, all of which steps are now carried on

separately, the first in MacDougall furnaces, the second in reverberatory furnaces, and the last in the converter on a concentrated ferrous-cuprous-sulphide.

Temperature influences the rate of oxidation tremendously. Substances combine with oxygen at all temperatures, but with *very different velocities*.

"The higher the temperature, the more rapid is the combination; on the other hand, as the temperature falls; the process becomes slower and slower, and soon diminishes so as to be inappreciable."²⁴

"By time measurements of the progress of many chemical reactions, the approximate rule has been obtained, that the velocity of chemical reactions, on an average, is doubled by a rise of 10° C. in the temperature. That is to say, if a reaction at a given temperature requires, say, a quarter of an hour to reach a certain point, at a temperature 10° higher it would require only $7\frac{1}{2}$ minutes, and at one 10° lower, 30 minutes. If the temperature is lowered 100° , a $2^{10} = 1,024$ times longer period is necessary, or in our example, about 11 days. On descending further 50° , or, on the whole, only the moderate amount of 150° , it would be a year before the reaction had proceeded so far as it had done in a quarter of an hour at the higher temperature."²⁵

Conversely, assuming that the chemical reaction in roasting sulphides, for instance, requires 900 sec. to reach a certain point at, say, 500° C., according to the above rule, the same reaction, conducted at $1,150^{\circ}$ C., would have proceeded as far in 900×0.5^{64} part of a second of time. It will be seen that the expression 0.5^{64} conveys such an infinitesimally small period of time that the mind can hardly grasp it. This expression solved gives a fraction with 16 zeros to the right of the decimal point preceding the number 4,878 . . . Even the slight increase of 150° C. in temperature would reduce the time of 900 sec. to 0.054931 . . . sec.

These examples are merely cited to show the powerful influence of temperature on the speed of reactions. It appears desirable, in order to accelerate the process of oxidation, to introduce the sulphides in a finely divided form at the focus of the chemical reactions, rather than melt the coarse lump ore slowly and pass it through the slag into the molten sulphides below.

Similar considerations, and others also, would favor the introduction of the siliceous fluxing ores in a finely powdered condition through the tuyères. They would then come immediately into intimate contact with the FeO, as the iron is oxidized. The tendency to form Fe_3O_4 , which appears to be particularly pronounced at the low temperatures at which the basic-lined copper converter is operated, would then probably be lessened, as the silica would be introduced at the very point where the iron is oxidized. A possible explanation for the formation of Fe_3O_4 may

²⁴ Wilhelm Ostwald, *op. cit.*, p. 66.

²⁵ Wilhelm Ostwald, *op. cit.*, p. 107.

be the existence, at times, at the tuyères, of a temperature lower than that of the formation of the ferrous silicate.²⁶

Such practice would not present unsurmountable mechanical difficulties. Assuming the air to weigh 1.25 kg., and the FeS 4.8 tons per cubic meter, and 11.5 tons of air to be required for the oxidation of 1 cu. m. FeS, the iron to FeO and the S to SO₂, we find the air volume to be more than 9,000 times greater than the FeS volume which the air is to oxidize. The silica required to flux this amount of iron, to form 2FeO, SiO₂, would be about 1.6 tons, occupying, at 2.3 sp. gr., 0.7 cu.m., which, if added to the 4.8 tons iron sulphide, would make the combined volume less than 1-5400th of that of the air required to oxidize the FeS. In other words, each cubic meter of air would have to carry with it into the converter 0.7 kg. of ore mixture, consisting of iron sulphide and siliceous flux. This large volume of air would be an efficient carrier of the finely ground ore.

Such practice would also offer an opportunity of controlling the process, especially the SiO₂ content in the slag, the basicity or acidity of which affects the amount of corrosion of the magnesite lining. The silica content could be controlled by decreasing or increasing the rate of feed. Under this practice, the slag would rise up through the molten bath with the rising gas stream: and, at an adequate converter speed, sufficient slag for a continuous discharge would be produced.

We have seen (footnote 4, p. 1087) that a heatsurplus of 905 cal. is available per kilogram of oxygen, after deducting the heat (1,407 cal.) carried off by the waste gases from the oxidation of the FeS to FeO and SO₂, and that 1 kg. of oxygen (in addition to the sulphur oxidized) oxidizes 1.17 kg. Fe to 1.5 kg. FeO, forming about 2.1 kg. of slag. This slag has a heat content of about 400 cal. per kilogram,²⁷ of which about 25 per cent. is furnished by the heat of formation of the ferrous silicate,²⁸ so that about 300 cal. per kilogram of slag would have to be

²⁶ If magnetite is formed in the copper converter, of which there seems to be little doubt, it would be interesting to know in what form it exists in the slag or the converter contents. Is it held in suspension, as unfused particles or crystals, or is its melting point lowered by being mixed with other minerals either chemically or mechanically? Fe₃O₄ has a high melting point. Kohlmeyer (*Metallurgie*, vol. vi, p. 323, 1909) determined it at 1,527° C., that of Fe₂O₃ at 1,565° and showed that, for these oxides generally, the melting point rises with an increase in oxygen and is between 1,500° and 1,600° C. The heat content is also very high. According to Richards's determinations, it is $0.1447 t + 0.000188 t^2$. Even at 1,150° C. this would amount to 415 cal. per kilogram of Fe₃O₄. If it does not enter into chemical combination with silica, as is likely the case, more than 100 cal. from the heat of formation of the ferrous silicate is lost (per kilogram of ferrous silicate).

²⁷ Fayalite, Fe₂SiO₄, Q, cold (0°-1,040° C.), 310 cal. latent heat of fusion, 85 cal. (Vogt, through Richards, *loc. cit.*).

²⁸ H. O. Hofman, *Trans.*, xli, 1910, p. 508, found an average value of 109 cal. per kilogram of ferrous singulo-silicate (2FeO, SiO₂).

furnished; and as 2.1 kg. of slag are produced from the oxidation of 1.17 kg. Fe, 630 cal. have to be supplied from the surplus of 905 cal. per kilogram of oxygen, leaving some 275 cal. still available. A *résumé* of these calculations, in tabulated form, here follows:

Heat Balance of Ferrous Sulphide, Converter-Smelting Process

$$\begin{array}{rcl} \text{FeS} & + 3\text{O} & = \text{FeO} + \text{SO}_2 \\ - 24,000 & 65,700 + 69,260 \text{ (gas)} & \\ & + \frac{110,960}{48} \text{ cal.} & = 2,312 \text{ cal. developed per kilogram of oxygen.} \end{array}$$

Process conducted at 1,150° C.

<i>Heat Absorbed</i>		<i>Heat Developed</i>	
1.34 kg. SO ₂ @ 276 cal.	= 370 cal.	Heat of formation of ferrous singulo-silicate, @ 100 cal. per kg.	
3.40 kg. N ² @ 305 cal.	= 1,037 cal.	2.1 kg. = 210 cal.	= 9 per cent.,
Total in waste gases	= 1,407 cal. = 56 per cent.	Net heat of oxidation (after deducting heat of dec. of FeS)	
2.1 kg. of ferrous singulo-silicate slag @ 400 cal.	= 840 cal. = 33 per cent.	2,312 cal.	= 91 per cent.
Surplus, by difference, for radiation and other losses	= 275 cal. = 11 per cent.		
	2,522 cal. = 100 per cent.	2,522 cal.	= 100 per cent.

It is not safe to allow less than 100 cal. per kilogram of FeS for radiation losses, and as $\frac{88 \text{ FeS}}{48 \text{ O}} = 1.833$ kg. FeS take part in the above process, 184 of the 275 cal. are lost in that way, leaving a final surplus of 91 cal. per kilogram of oxygen introduced, or, in round numbers, 50 cal. per kilogram of FeS.

On the basis of 400 cal. heat content in 1 kg. of slag and 250 cal. in the cuprous sulphide, to melt every unit (in this case figured as being $\frac{1}{10}$ part of 1 kg. FeS) of Cu₂S added to 1 kg. FeS would require 2.5 cal. and to melt every unit of silicate 4.0 cal., which indicates plainly how moderate must be the amount of diluents that are non-productive of heat. Any iron that is already oxidized, and silica that is no longer free, but combined with ferrous oxide, alumina, or lime, act as such diluents. The cuprous sulphide, likewise, is non-productive of heat until the second stage, that of blowing "white metal" to blister copper, is reached. While the process is initially started with molten matte, having a heat content of 250 cal. per kilogram., this cannot be counted on continuously, if the converter-smelting process is to be self-supporting from the heat-producing elements in the ore alone.

The above data convey the importance for the success of the process of very large converters, with large storage capacity for molten matte, and

rapid speed in converting; i.e., large air supply. If there is, for instance, a *net* surplus of only 10 cal. per kilogram of oxygen, with 1,100 kg. air supply, per minute, or 250 kg. of oxygen per minute, there will be available a net surplus of 2,500 cal. per minute, whereas with an air supply one-half or one-fourth as large, only 1,250 or 625 cal. would be available. There is, therefore, great danger that the rate of surplus heat supply will lag behind the rate of heat radiation, and check the process. Of such heat surplus only about one-third to one-half goes to increase the bath temperature, the remainder escaping with the waste gases, the temperature of which it increases.

The above calculations point to the profitable use of air enriched with oxygen, or the moderate preheating of the air which could be done with the converter waste gases, so that ores with a smaller ferrous sulphide and higher silicate content could be handled.

The process is nevertheless under more perfect control of the operator than any pyrite blast-furnace smelting process can be; since, by reducing the amount of ore fed or stopping its supply entirely for short periods of time, he can draw a heat supply from the molten bath. This bath, absorbing any excess heat produced as the blow progresses, will also give off a portion of its heat, during such critical stages in the process as may occur for a variety of reasons. It has been shown (p. 1091) that about 30 metric tons of matte can be stored below the elevated tuyère bottom. This large weight of matte, in addition to at least a like amount stored above the tuyère bottom, acts as an efficient heat accumulator containing fully 15,000,000 cal., and capable of absorbing 200,000 cal. more, a large portion of which can be given off during critical stages in the process. This is undoubtedly one of the most important features that determine the success or failure of a converter smelting process. As already explained, this is *not* to be replenished by fresh matte, after the applications are once started, as that 250 cal. are lost for every kilogram of molten matte that is to be superseded by *raw unmelting* sulphide ore.

The velocity of converting smelting, as well as the weight of the charge in the converter, must be so great that radiation losses are reduced to a minimum, and too rapid cooling of the bath is prevented. Likewise, excess heat developed must be absorbed, so that it can be given off if an occasional change in the ore supply to one having a decreased heat content (less Fe and S) should require it. Such a large molten bath acts very much like a flywheel, absorbing and giving off energy, which in this case is in form of heat.

The forming of slag near the tuyère orifices by this method of blowing through the tuyères finely ground sulphide ores, with the requisite amount of siliceous ore flux, or concentrates (which always contain a certain amount of silica), makes it doubtful if punching can be entirely avoided. To do the work of punching mechanically and automatically,

I have designed a mechanical tuyère-punching apparatus, already described elsewhere.²⁹

The foregoing remarks regarding the thermo-chemical characteristics of the converter smelting process, demonstrate that a siliceous flux should have as little iron in combination with it as possible. Such iron would furnish no heat, since it cannot be further oxidized; and the heat of formation between uncombined SiO_2 and FeO is likewise lost if a silicate of iron, instead of free silica, is used. It also reduces the amount of silica available for fluxing purposes. In this respect conditions in the converter smelting process are the same as in the ordinary pyrite-smelting blast-furnace process, and demand a quartzose ore, high in *free* silica.

XI. CONSTRUCTION OF TUYÈRE BOTTOM

The tuyère bottom, of which a long life is demanded, is naturally one of the vital parts of a bottom-blown converter.

In devising such a bottom, I was greatly aided by a study of Thomas steel practice and a comparative study of iron metallurgy. Such comparisons are always of value; and copper metallurgists can undoubtedly learn much from the iron and steel men. The ingenious designs of the American engineer Alexander L. Holley, by which quick renewals of Bessemer converter bottoms and shells were made possible, and helped early to make the Bessemer process a commercial success, and which were applied with profit to the Thomas converter, should be familiar to all copper metallurgists. Of late years copper-smelting practice has approached in scale of operations that of iron and steel works, and has adopted some of the apparatus commonly used in steel works.

In the Thomas converter practice particularly, there is much of value to the copper metallurgist, because it uses basic-lined converters, and has been developed to a high state of perfection during the last 34 years, since its start on a practical scale in 1879 at the works of Bolckow, Vaughan & Co., Middlesbrough, England.³⁰

In the preparation of the basic linings made of dolomite, and, occasionally of magnesite, and that of the tuyère and needle bottoms, a great deal of work has been done, since the success of this process depended principally on the life and cost of the linings. A bit of interesting history is the use of a solution of sodium silicate as a binder of the lime lining by Sidney Gilchrist Thomas, when he prepared his first experimental converter that could hold a 6-lb. charge.³¹ The use of sodium silicate as a

²⁹ *Bulletin* No. 83, Nov., 1913, p. 2664 ff.

³⁰ *Dreissig Jahre Thomasverfahren in Deutschland*, *Stahl und Eisen*, vol. xxix, No. 38 (1909).

³¹ *Stahl und Eisen*, *loc. cit.*, and F. W. Harbord, *The Metallurgy of Steel*, 2d ed., p. 56.

binder for basic linings in copper converters is said to have been discovered accidentally at the Anaconda Copper Mining Co.'s works.

Another interesting historical fact is the presentation by Thomas, in a paper delivered May 8, 1879, before the Iron and Steel Institute, of his conclusion that the elimination of the phosphorus in the steel converter depended mainly on the preparation of a highly basic slag with silica below 20 and $\text{CaO}(\text{MgO})$ above 30 per cent. Such a basic slag required, of course, also a basic lining. He also recognized at that early period the great potential value of the Thomas slag as a phosphate fertilizer.³²

What made the use of the basic lining in the Manhès-David converter a success, was, aside from the adoption of large converters, the realization that the basic lining, to avoid excessive corrosion, demanded a very basic slag. Thomas, from the very beginning of his operations, started to protect the basic converter lining by the addition of basic fluxing materials, since it is very much cheaper to eliminate impurities by adding lime to the converter charge than to have the lining corroded in furnishing lime for that purpose. A moderate amount of lime might be a desirable addition to the copper converter, in making the slag more fluid and lighter, and permitting wider variations in temperatures, *i. e.*, higher temperatures in the converter, without increasing the magnesite corrosion.

In view of the great amount of information collected and experience had in the use of basic linings abroad, the employment of expensive linseed oil as a magnesite binder in the Peirce-Smith converters seems somewhat surprising. In Europe anhydrous tar is used almost exclusively as a binder of dolomite or magnesite mortar in joining bricks, or tamping into place the ground dolomite or magnesite. This tar cakes when the molten metal comes in contact with it, and makes a very desirable and tight joint. This practice can no doubt be used to advantage also by the copper smelter. I have heard of its adoption by copper-smelting works in Siberia. The increase in life of lining has been due chiefly to the high pressures used in making the dolomite bricks, 500 atm. being no longer uncommon.

The method of constructing needle bottoms may be the same as that used in the Thomas steel industry. The finely ground magnesite or dolomite is mixed with from 5 to 10 per cent. of anhydrous tar, and tamped in a form having the shape and diameter of the desired bottom. For this work a Versen machine is usually employed. The form or mold is a slightly tapered cast-iron ring with a cast-steel bottom plate, which has as many holes as there are tuyère openings in the bottom. The forming ring and its perforated bottom plate rest on a supporting table, which is lowered when the bottom has been tamped. There is also a plate, provided with as many steel needles as there are needle tuyère holes in

³² *Stahl und Eisen, loc. cit.*

the bottom. These needle bottoms are made fairly thick, 900 mm. (about 3 ft.) being a standard size. The needles are made still longer to clear the bottom. Below the cast-steel bottom plate forming the bottom of the form is a perforated diaphragm, which acts as a guide plate to the many needles, keeping them vertical and parallel. The plate to which the needles are secured is also guided by heavy steel rods, and can be forced up or down by a feed screw. The rammer proper is carried by a movable lever arm, and is mechanically operated, traveling from the center of the form to its periphery and back again, describing concentric arcs on the dolomite or magnesite mass while the form is gradually being filled. While this ramming or tamping is going on, the table on which the form rests revolves continuously. To secure a uniformly tamped bottom, the number of strokes of the rammer is proportioned to the diameter through which the rammer travels at any time.

The tuyère needles are forced into the rammed mass by the screw feed, but are always kept several centimeters below the top of the mass, so that the hardened steel needle points are not damaged by the rammer. The finished bottoms resting on the steel bottom plate are then run on trucks to kilns, holding as many as six bottoms each, and are burned. The bottoms, after burning, are pressed into place in the converter by a hydraulic jack, and when in place, the bottom plate forms the top portion of the wind box.

The size of the bottoms varies, of course, with the size of the converters. Those of the Burbacher Hütte,³³ for 24-ton converters, are 1.9 m. (75 in.) in diameter, 0.9 m. high, and have 253 air holes or tuyères 15 mm. (0.59 in.) in diameter. Those of the 21-ton converters at the Thomas steel works of the Aachener Hütten Aktienverein in Rothe Erde³⁴ are 1.9 m. in diameter at the bottom, and 1.825 m. at the top, are 0.9 m. high, and have 200 holes, 18 mm. (nearly 0.72 in.) in diameter. This practice could be modified, somewhat, to adapt it to the different conditions of copper-converter practice. The tuyère holes would be made slightly larger, and conically enlarged at the bottom, tapered for one-third, and straight for about two-thirds, of the total thickness of the bottom. Smooth steel or copper pipes would be inserted into these holes, to avoid excessive friction.

Another method of making these bottoms would be by means of hydraulic presses, forcing together layers of the ground magnesite mixed with 10 per cent. of tar, and then burning the entire bottom. The holes would be drilled subsequently in the hard mass.

It would not be necessary to have this apparatus and the kilns at

³³ Das neue Thomasstahlwerk der Burbacher Hütte, *Stahl und Eisen*, vol. xxviii, No. 46, p. 1641 (Nov. 11, 1908).

³⁴ Das neue Thomasstahlwerk des Aachener Hütten Aktienvereins in Rothe Erde, *Stahl und Eisen*, vol. xxvii, No. 43, p. 1525 (Oct. 23, 1907).

every works. Since two sizes of converters, of, say, about 12 ft. and 20 ft. diameter of shell, would fill the needs of most works, these could be made standard, and each could have a standard-size bottom. A centrally located establishment could prepare and ship to the different works the bricks, bottoms, or ground material. Deposits of excellent dolomite and magnesite are available in California; and there appears to be no good reason why the imported article from Greece should be preferred, when it can be secured nearer home, particularly by Western smelting works. In such an enterprise the different works could undoubtedly co-operate profitably.

The European, especially the German, steel works³⁵ used to burn their own dolomite, necessitating kilns and brick-pressing departments, etc.; but of late years one company performing these operations for most of the steel plants provides them with excellent material, and is also enabled thereby to save valuable by-products, which can only be done commercially in a large centrally located plant.

XII. THERMO-CHEMISTRY OF THE MANHÈS-DAVID AND THOMAS CONVERTER PROCESSES.

The fact that, in iron converting, the tuyères are not punched while blowing, as they are in copper converting, and are only occasionally inspected, led me to investigate and compare the thermo-chemical and physical conditions of the two processes. I endeavored also to ascertain the reasons for the early failures in attempting to blow white metal to blister copper in bottom-blown Bessemer converters.

As a basis of thermo-chemical comparison, I made use of the investigations of the Thomas process by F. Wüst and L. Laval,³⁶ and of a hypothetical heat balance of the Manhès-David converter by Prof. J. W. Richards.³⁷

The work of Wüst and Laval is particularly valuable, as the first determination of the composition of the contents of the Thomas converter

³⁵ By reason of its large deposits of iron ore high in phosphorus, Germany profited more than any other country by Sidney G. Thomas's invention, so that the Thomas process has found its widest application in Germany, where the total production of steel by the basic process in 1913 probably exceeded 17,000,000 tons, of which about 10,000,000 tons were made in the Thomas converters, and the remainder in basic Siemens-Martin furnaces. Only about 600,000 tons were made by the acid open-hearth, and 200,000 in the acid converter.

³⁶ Experimentelle Untersuchung des Thomasprozesses, *Metallurgie*, vol. v, Nos. 15 and 16, pp. 431-462, and 471-489 (1908.)

Experimentelle Untersuchung des Thomasprozesses (abstracted from *Metallurgie*), *Stahl und Eisen*, vol. xxix, No. 4, p. 121 (Jan. 27, 1909).

³⁷ In chapter xiv, Application of Thermo-chemistry, p. 551 ff. in *Peters's Principles of Copper Smelting*.

during the different stages of the process; the weights of the impurities and iron burned off during these stages, and the analysis of gases under actual large-scale operating conditions. This is a difficult piece of work, since the duration of a blow is only about 15 min., so that samples have to be taken at very short intervals of time, if the nature of the reactions and the rate of the elimination of impurities are to be determined with any degree of accuracy. Silicon, for instance, is removed to the extent of 90 per cent. in less than 2 min.

From 9 to 10 samples of the converter contents were taken (iron and slag separately), at intervals of 0.5 to 2.25 min. Gas samples were taken for corresponding periods of blows, from a number of converters, and the mean of the gas analyses from five different charges was taken as representing the composition of the gas.

From the data in the papers named, I have prepared Table VIII, reducing all figures to a basis of calories of heat developed and expended per ton of material charged. For the Manhès-David process, the two stages of blowing the matte to white metal, and the white metal to blister copper, have been considered separately, basing the heat balance on calories per metric ton of matte, and per metric ton of white metal charged. The figures for the Thomas converter are those for 10-ton charges of pig iron, reduced to a basis of 1 metric ton.

To secure a better criterion of the heat balance of the processes, the heat of oxidation, *i.e.*, heat developed, and the heat carried off in the gases, including heat of decomposition of water in the Thomas converter, and the very appreciable heat of decomposition of the sulphides in the copper converter, are stated by themselves, without taking into consideration the heat content of the molten metals. Then the complete heat balance follows, including that item.

Comparing the two processes, we see that there is not a very great difference in the amount of heat by oxidation of the constituents eliminated, per ton, from pig iron and from matte. With mattes, the total heat developed during the blow to white metal will be greater, the greater the iron and sulphur in the matte, that is, the lower its grade in copper. The pig iron brings with it 40 per cent. of the total heat, whereas the matte carries only 32 per cent. But 68 per cent. of the total heat is developed in the copper converter, and only 60 per cent. in the steel converter, from the oxidation of impurities. Neglecting the heat contained in the molten metals, there is an excess of 240,000 cal. over heat expended in the steel converter per ton of pig iron charged, and only 193,000 cal. per ton of matte in the copper converter. Numerically, this may look unfavorable for the copper-matte charge, but when the greater specific heat of the steel and the very much higher temperatures demanded by the Thomas process (1,600° C. toward the finish of the blow) are borne in mind, the excess of 47,000 cal. per ton of pig iron

TABLE VIII.—*Heat Balances of Thomas (Steel) and Copper Converters Compared*

Thomas Converter			Manhès-David Converter		
Calories per Ton of Material Charged	Pig Iron, 1,000 kg.		Matte, 1,000 kg.		"White Metal" 1,000 kg.
Heat developed.....	410,000		533,000		400,000
Heat expended.....	170,000		340,000		462,000
Heat excess.....	240,000		193,000	Def.	-62,000
Heat in pig iron, matte, and "white metal".....	276,000	40.0%	250,000	32.0%	290,000 42.0%
Heat of oxidation.....	410,000	60.0%	533,000	68.0%	400,000 58.0%
Total heat.....	686,000	100.0%	783,000	100.0%	690,000 100.0%
Heat in gases inc. decomp. of H ₂ O.....	170,000	24.8% (abt. 20,000 cal.)	240,000	30.8%	320,000 46.3%
Decomposition of sulphides.....			100,000	12.7%	142,000 20.5%
Heating of lime, and carried away by slag.....	140,000	20.4%	133,000	17.7%	17,000 2.4%
Heat in steel, "white metal" and blister copper.....	306,000	44.6%	176,000	22.5%	138,000 20.0%
Radiation and other losses..	70,000	10.2%	134,000	17.0%	73,000 10.8%
Total heat.....	686,000	100.0%	783,000	100.0%	690,000 100.0%

charged is seen to be essential to the process. This accounts for the large difference in the heat contents of the steel and the white metal—306,000 cal. and 176,000 cal. respectively. As the impurities in the pig iron are burned out, the melting point of the refined metal as it gradually approaches the composition of steel is continuously raised, and additional heat is required to keep the bath in a molten condition. The pig iron has a heat content of only about 250 to 270 cal., and a comparatively low melting point (about 1,200° C.), whereas the steel requires a heat content of 335 to 350 cal. (sp. heat while liquid, 0.22; latent heat of fusion about 35 cal.; melting point between 1,560° and 1,600° C.,³³ depending on composition.) In Table VIII the heat content is expressed per ton of material charged. As only about 0.9 ton of steel is made from 1 ton of pig iron charged, 306,000 cal. per ton of pig iron charged are equal to

³³ Melting-point determinations of pure iron vary. Richards (*Metallurgical Calculations*, Part I, p. 67) gives it at 1,600° C., according to Roberts-Austen. Carpenter (*Metallurgie*, vol. v, p. 679, 1908) determined it to be 1,510°. (See Ernest J. Kohlmeier, *Metallurgie*, vol. vi, p. 333, 1909.) In Landolt-Börnstein's *Physikalisch-chemische Tabellen*, melting-point determinations made by a number of different authorities range from 1,502° to 1,532° C.

340,000 cal. per ton of steel made, or 340 cal. per kilogram of steel. Likewise, since it takes from 1.5 to 1.7 tons of matte to make 1 ton of white metal, assuming the matte composition customary at copper smelters to-day, the 176,000 cal. per ton of matte charged are equal to from 264,000 to 300,000 cal. per metric ton, or 264 to 300 cal. per kilogram, of white metal made.

An interesting condition is disclosed in the second period of the Manhès-David process, when blowing the white metal to blister copper. Deducting the heat expended in heating the waste gases, and the heat absorbed in decomposing the sulphide, Cu_2S , into 2Cu and S , from the heat of oxidation of the sulphur to sulphurous acid anhydride, SO_2 , there is a heat deficiency of 62,000 cal. The heat balance also shows that 66.8 per cent. of the *total* heat is used for heating the gas and decomposing the sulphide, whereas only 58 per cent. is developed. This deficiency of 62,000 cal., besides 90,000 cal. lost by radiation and the production of a slight amount of slag, has to be supplied by the heat accumulated in the molten white metal during the preceding operation of blowing the matte to white metal. The heat balance discloses the fact that these 152,000 cal. are furnished by the white metal, leaving, in the blister copper made, 138,000 cal. per ton of white metal charged. Since about 0.8 ton of blister copper is made from 1 ton of white metal charged, the 138,000 cal. represent 173,000 cal. per ton of blister copper.

The feasibility of the second stage of the Manhès-David process is due to a combination of physical properties of the materials handled. The specific heat of the white metal, Cu_2S , is considerably higher than that of copper. At $1,200^\circ \text{C}$. the mean specific heat of Cu_2S is 0.22 (Bellati and Lussana),³⁹ so that its heat content at that temperature, adding 25 cal. for the latent heat of fusion, is some 290 cal. per kilogram. That of the matte is probably not far from that of the white metal. The specific heat of copper at its melting point is very materially less (0.133, Richards)⁴⁰ and its heat content at the melting point, $1,085^\circ \text{C}$., is 162 cal. (Richards) per kilogram. From 170 to 175 cal. per kilogram of blister copper will keep this in a molten condition at a safe temperature (about 60° to 100°C .) above its melting point. Then we have in the copper converter the very reverse of the conditions in the steel converter, where an increase of heat must be provided for the steel to be made from the pig iron, while in the copper converter, we gain heat from the change of the Cu_2S into copper, as follows: 1.25 kg. of Cu_2S contains $290 \times 1.25 = 362.5$ cal. is converted to 1 kg. of blister copper, requiring only 175 cal. to keep it fluid—a gain of 187.5 cal. per kilogram of copper.

³⁹ Through Richards, *loc. cit.*

⁴⁰ Landolt, *op. cit.*, gives the specific heat of liquid copper at $1,084^\circ \text{C}$. as 0.15563 (Glaser), and of solid copper. at from 26° to 148°C ., as 0.11064.

Since this amount is furnished by 1.25 kg. of Cu_2S , it represents 140 cal. per kilogram of Cu_2S . If the original heat was 270 instead of 290 cal., the net gain becomes 130 cal. (There is, of course, an additional source of heat in the oxidation of the S to SO_2 .)

This liberation of 130 or 140 cal. by each kilogram of Cu_2S in its transformation into copper, is available for other heat requirements than that of keeping the copper liquid. If there are no compensating heat losses (radiation, heat in waste gases, slag) and absorptions, the copper may become greatly overheated. In converting with great speed this can easily happen, through the rapid accumulation of the heat increment.

We have previously seen (p.1089) that, when only the theoretical amount of oxygen is used and no free oxygen escapes with the gases, the quantity of heat carried off by the waste gases for every kilogram of air introduced is about 16 cal. in excess of the net heat of oxidation developed; and, since 870 kg. air are required *theoretically* (see Table VII, column 9) to convert 1 metric ton of Cu_2S into 0.8 ton of copper, this deficiency is only 13,000 to 14,000 cal., but an increase of only 130 kg. of air would bring it up to 62,000 cal., the amount given in the heat balance, Table VIII, which was deduced from an entirely different source of information. There is, moreover, 90,000 cal. to be provided for radiation losses, and any heat carried away in whatever slag may be skimmed off.

Supposing no excess of air above the theoretical oxygen requirements to be introduced, and deducting the 16 cal. more of heat carried off by the waste gases than is developed by the oxidation of the sulphur, there are liberated about 100 to 125 cal. per kilogram of white metal in its conversion to blister copper, making available for other uses from 100,000 to 125,000 cal. per ton of white metal charged. The white metal, therefore, does not have to be very much overheated during the first stage of the process to carry it through the more critical second stage.

While the bath temperature will increase, if no excess of air is passed through it, its overheating is checked, to a certain extent, by the increase in the specific heat of the converter gases and the metal, so that an increased amount of heat is carried off by the gases. It may be stated, from approximate estimates, that in lieu of increasing 1° , the bath temperature increases only one-third that amount, the greater heat carried off by the gases reducing by two-thirds the increment in temperature.

This critical stage in the Manhès-David process explains the early failures of blowing white metal to blister copper in a bottom-blown (Bessemer) converter. The foregoing thermo-chemical data demonstrate also that white metal can be brought to blister copper in a bottom-blown converter, provided the charge of white metal is sufficiently large, and the converting is done rapidly enough to generate and store sufficient heat to supply the different heat losses occurring during that stage.

To keep the tuyères open, the necessity of subdividing the air currents and reducing the weight of air per minute per tuyère, exists to a greater extent during this stage than in blowing on matte, since the air must be heated principally by direct contact with the copper, when the latter has been brought down into the tuyère region, and its heat-producing impurities (in the main, sulphur) have been burned out.

It does not make much difference whether this copper is oxidized to CuO or Cu_2O , the molecular heats of formation of which are 37,700 and 43,800 cal. respectively, or 590 and 342 cal. per kilogram of copper, depending on whether CuO or Cu_2O is formed. Expressed per kilogram of oxygen, this is 2,356 and 2,738 cal. respectively, or 540 and 625 cal. per kilogram of air introduced, which is 240 and 325 cal. (see p. 1088 *ff.*) more than is needed to heat that amount of air from 0° to $1,150^\circ \text{C}$. A like amount of heat will, of course, be absorbed, when the oxides react with the sulphides, so that this heat of oxidation cannot be reckoned as being available.

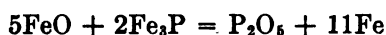
The clogging of the tuyères in the very small copper converter during this second stage was, perhaps, not due so much to the "chilling of the copper" (usually assigned as the reason for the first failures in applying Bessemer's bottom-blown converter to the conversion of white metal to blister copper) as to the fact that the copper oxides formed required very much more heat content than the copper, to keep them molten. The specific heats of these oxides, even at very low temperatures, point to that. I was not able to find a determination of the specific heats of these oxides at $1,100^\circ$ or $1,200^\circ \text{C}$., except a determination by Regnault (through Richards) of 0.142 for temperatures between 12° and 98° for CuO , which points to a high specific heat at higher temperatures. It is therefore quite likely that a portion or all of the surplus heat of oxidation (after the air had been heated) would be absorbed by the oxides formed, and yet be insufficient to keep these molten, and stop their reactions with the cuprous sulphide.

In a large converter, especially of the type I propose, the air weights entering through the tuyères are so moderate, and are distributed over so large a matte surface, that with the shallow bath and the nearly equal specific gravities of the cuprous oxide and sulphide, every opportunity is given for a commingling of the two and bottom blowing appears entirely feasible, even if the elevated tuyère bottom were discarded for a flat bottom.

According to Wüst and Laval there is also a critical stage in the Thomas process, which occurs from 4 to 5 min. after starting the blow, and during which the available excess heat developed is so small that it renders the success of the process questionable, if the metal bath had not been superheated previously (by the burning of the silicon and a portion of the carbon) so as to act as an accumulator of heat, which is given off during this

critical period. This condition resembles very closely that of blowing matte to white metal, and storing the heat excess developed during this stage in the white metal, which then liberates it and helps the process over the critical stage of the following period.

After the silicon has been burned off, the iron begins to be oxidized to FeO in increasing amount. The phosphorus in the pig iron is contained as a phosphide of iron, Fe_3P , held in solution in the iron. The FeO formed reacts with this phosphide, according to the formula:



The oxidation of the phosphorus furnishes the necessary heat⁴¹ to create the high temperatures needed to eliminate the phosphorus. The lime-silicate slag, containing about 60 per cent. CaO, is exceedingly viscous and has a high melting point and heat content, between 530 and 550 cal. In copper converting, the slag contains about 400 cal. only.

XIII. RELATIVE RATE OF ELIMINATION OF IMPURITIES IN THE MANHÈS-DAVID AND THOMAS PROCESSES

Table IX shows the rate of elimination of impurities from a 10-ton charge of pig iron in a Thomas converter, as calculated from the data given in the paper by Wüst and Laval.⁴² It will be seen that the average rate of elimination of impurities is 97.5 kg. per minute, for charge I, and 131.0 kg. per minute for charge II, or a mean of 114 kg. per minute. In addition to the 1,462 to 1,860 kg. of impurities eliminated, there is added to the charge lime amounting to from 14 to 17 per cent. of the weight of the charge of pig iron, so that in about 15 min. from 2.8 to 3.56 tons of material are eliminated. In a 25-ton converter the weight of the impurities eliminated is about 2.5 times greater, or 250 to 325 kg. per minute, or, with the lime added, from 5 to 8 tons in 15 min.

In the copper converters at the Garfield smelter the weight of the iron and sulphur eliminated per minute is about 67 and 34 kg. respectively, and in the Great Falls converter about 180 kg. iron and 90 kg. sulphur per minute, from which it will be seen that we are only approaching now in copper converting a speed which has not been out of the ordinary in steel converters for a number of years past.

In view of the variations in the iron and sulphur content of mattes

⁴¹ The high temperature toward the last stage of the Thomas process increases the loss of iron, which goes into the slag as an oxide (FeO). This loss is considerable, amounting on the average to about 3 per cent. of the pig-iron charge, so that the basic open-hearth process is being adopted more extensively, which eliminates the impurities equally well, and in which the oxygen is partly furnished from the iron ores charged with the pig iron. Moreover, Thomas steel usually has a higher phosphorus content than basic open-hearth steel, about 0.06 as compared with 0.03 per cent. (*Stahl und Eisen*, vol. xxix, No. 4, Jan. 27, 1909, p. 133.)

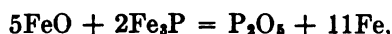
⁴² *Op. cit.*

treated, and the different combining weight of oxygen with different elements, I propose that the capacity of a converter be stated by naming the amount of oxygen or air in kilograms or pounds that is pumped into the converter. This can be the only correct measure of converter capacity, permitting comparisons between different plants.

TABLE IX.—*Elimination of Impurities in Iron in the Thomas Steel Converter*

Pig Iron	C	Si	P	Mn	S	Fe	Minutes	Total Imp. Eliminated in Kg.	Total Imp. Eliminated in Kg. per Min.
	351.50	50.40	210.54	89.08	18.50	9,760	Bath 10,480 Kg.		
							Min. Sec.		
1	31.60	46.67	3.09	6.02	8.40		1-30	95.78	64.00
2	50.10	3.12	11.07	39.88	-2.20		1-45	101.97	58.30
3	72.50	-0.10	6.00	1.35	-0.75		2-00	79.00	39.50
4	64.74	-0.19	11.18	-3.33	0.21	95	1-30	167.61	112.00
5	60.24	-0.09	8.00	-9.11	0.21	108	1-30	167.25	112.00
6	63.20	0.50	31.95	3.84	1.77	67	1-45	168.26	96.00
7	5.88	-0.37	89.01	-2.11	4.46	70	1-45	166.87	95.00
8	1.85	0.02	39.36	20.93	-1.41	174	1-30	234.75	157.00
9	-0.05	0.33	4.93	10.87	0.91	264	1-45	280.99	160.00
	350.06	49.89	204.59	68.34	11.60	778	15-00	1,462.48	97.50
Pig Iron	376.88	91.47	197.15	116.81	10.14	10,245	Bath 11,040 Kg.		
							Min. Sec.		
1	26.17	89.30	3.33	58.18	2.21		2-00	179.19	89.60
2	43.45	0.88	17.76	16.59	-1.34		2-00	77.34	38.70
3	110.26	0.44	7.17	14.35	0.85		2-15	133.07	59.00
4	102.04	-0.08	11.38	2.92	0.73	201	2-00	317.99	159.00
5	92.46	0.23	18.98	6.78	-1.30	184	1-45	301.15	172.00
6	1.15	-0.07	125.58	-5.21	4.70	236	2-15	362.15	161.00
7	0.04	0.12	1.12	6.39	1.97	310	1-30	319.64	213.00
8	0.03	0.01	1.17	2.14	0.90	186	0-30	170.25	340.50
	375.60	90.83	186.49	102.14	8.72	1,097	14-15	1,860.78	131.00
								1,462.48	97.50
							Average	3,323.26	114.00

In the reaction



stated above, 5O (weighing 80) oxidizes only 2P (weighing 62), but 5Fe (weighing 280); *i.e.*, to oxidize 1 kg. of P requires $\frac{80}{62} = 1.29$ kg. O, and 1 kg. O oxidizes only $\frac{62}{80} = 0.775$ kg. P, whereas $\frac{280}{80} = 3.5$ kg. Fe combine with 1 kg. O, and $\frac{80}{280} = 0.286$ kg. O is required to burn 1 kg. Fe. In other words, 1 kg. oxygen will oxidize 4.52 times as much iron as phosphorus. To emphasize the importance of the effect of the different oxygen requirements of the elements to be eliminated upon the rate of elimination or speed of converting, the following figures are subjoined.

Steel Converter

C to CO	1 kg. C	requires	1.33 kg. O
	1 kg. O	burns	0.75 kg. C to CO
C to CO ₂	1 kg. C	requires	2.67 kg. O
	1 kg. O	burns	0.375 kg. C to CO ₂
Si to SiO ₂	1 kg. Si	requires	1.125 kg. O
	1 kg. O	burns	0.89 kg. Si to SiO ₂
P to P ₂ O ₅	1 kg. P	requires	1.29 kg. O
	1 kg. O	burns	0.775 kg. P to P ₂ O ₅
Mn to MnO	1 kg. Mn	requires	0.29 kg. O
	1 kg. O	burns	3.44 kg. Mn to MnO

Copper Converter

Fe to FeO	1 kg. Fe	requires	0.286 kg. O
	1 kg. O	burns	3.5 kg. Fe to FeO
S to SO ₂	1 kg. S	requires	1 kg. O
	1 kg. O	burns	1 kg. S to SO ₂
Cu to CuO	1 kg. Cu	requires	0.25 kg. O
	1 kg. O	burns	4.00 kg. Cu to CuO
Cu to Cu ₂ O	1 kg. Cu	requires	0.125 kg. O
	1 kg. O	burns	8.00 kg. Cu to Cu ₂ O

It will be seen that the oxygen requirements of the elements to be burned off are considerably greater in the Thomas than in the copper converter, so that a relatively much greater weight of oxygen (air) has to be supplied to the former, to burn off an equal weight of impurities. This accounts for the large air weight used in modern steel converter plants—namely, more than 1,000 kg. air per minute for 25-ton converters.

By a proper distribution of the air, I look for the use of greatly increased air weights in copper converters.

It is evident that by stating the air weight used in a converter, its capacity is also fixed, since the air or oxygen is used according to chemical laws.

(The following atomic weights were used in figuring the weight of oxygen required for different elements: C, 12; Si, 28.4; P, 31; Mn, 55; Fe, 56; S, 32; Cu, 64; O, 16.)

XIV. ADVANTAGES OF LARGE CONVERTERS

The advantages of large converters with a high rate of speed in converting may be summed up as follows:

1. Smaller capital investment for plant. It is apparent that a 20-ft. converter to which air is supplied at the rate of 1 ton or more per minute, so that it will do the work of three to four converters with proportionally smaller air supply, will cost materially less for the installation of apparatus and buildings.

2. Decreased operating cost.

3. Decreased power cost, since blowing can be done on shallow charges with lower air pressures.

4. Reduction of air losses by using only one or two very large converters instead of a number of smaller ones, and by eliminating the punching of tuyères and tuyère valves.

5. Further saving in installation cost of converter plant and in operating cost by substituting electrically operated matte-ladle cars for the traveling cranes now commonly used.

The use of ladle cars is undoubtedly the coming practice for charging converters. The siliceous ores can be charged from chutes placed above the converter mouths, on the side of the converter building aisle, and supplied by larries or conveyors. As the basic-lined converters are stationary furnaces, and are lined in place, the large powerful traveling cranes are no longer needed, since the shells are no longer lifted from the stands and carried to the relining department.

I have before me designs of a ladle car with ladle capable of holding 25 tons of matte. The entire car weighs 50 tons when loaded. The ladle is tilted electrically by a direct-current 10-h.p. motor; and the car is propelled by a 25-h.p. direct-current motor, from a cab in front of the car. Its speed is 300 ft. per minute, gauge standard (4 ft. 8½ in.). The maximum wheel pressure per wheel, with two 4-wheel trucks, is only 6.25 tons. Such a car can be safely supported by a comparatively light steel trestle bridge, with the bents close together, so as to reduce the beam spans and the size and weight of the beams.

A 50-ton 3- or 4-motor crane (with main hoist of 50 tons and auxiliary of 15 tons), which is a relatively small-sized crane for modern converter plants, with a span of only 26 ft., weighs 30 tons (without its load), and with an 80-ft. span fully 55 tons (without its load). The 26-ft. span is entirely too short for a modern large-sized converter plant. The larger crane requires 30 h.p. for propelling the crane, at about half the speed of

the ladle car; 12 h.p. for propelling the trolley; a 35-h.p. motor for the main hoist, and a 5-h.p. motor for the auxiliary hoist. The maximum wheel pressure per wheel of this crane is 41 tons, nearly seven times as great per wheel as that of the ladle car. This great wheel pressure demands heavy columns and large girders for the crane-ways. This heavy steel construction can be avoided by the use of a ladle car. For elevating the car from one level to another, electric or hydraulic elevators would be used.

By dispensing with traveling cranes, the operatives of the converter plant will become more efficient, being less demoralized by the cranes passing constantly and rapidly over them, with loads of molten metal, which at times slops out of the ladles. The ladle-car operators would not be so much or so continually exposed to the sulphur dioxide fumes, or be so greatly disturbed by matte explosions, as are crane operators.

As most converter plants were originally designed for the acid-lined converter, which required powerful cranes for economical operations and the moving about of the shells, as well as of the ladles, such cranes are still made use of for the basic-lined converters. But in new plants the advantages of ladle cars should be carefully considered.

6. Reduced magnesite consumption. The size of the converter shell may influence the magnesite consumption, since, with an increase in the size of the converter, the ratio between lining surface in contact with the molten material and the volume of that material becomes smaller. The following is a comparison of a 12-ft. with a 20-ft. converter (having diameters of 8 and 16 ft. inside of lining respectively):

12-ft. Converter		20-ft. Converter
Bottom area.....	50 sq. ft.	200 sq. ft.
Circumferential area, for each foot in height.....	25 sq. ft.	50 sq. ft.
Cubical contents:		
1 ft. height.....	50 cu. ft.	200 cu. ft.
2 ft. height.....	100 cu. ft.	400 cu. ft.
3 ft. height.....	150 cu. ft.	600 cu. ft.

Ratio of Total Surface in Contact with Matte (bottom + circumferential area) to Volume of Matte:

Height		
1 ft.	$\frac{75 \text{ sq. ft.}}{50 \text{ cu. ft.}} = 1.5$	$\frac{250 \text{ sq. ft.}}{200 \text{ cu. ft.}} = 1.25$
2 ft.	$\frac{100 \text{ sq. ft.}}{100 \text{ cu. ft.}} = 1.0$	$\frac{300 \text{ sq. ft.}}{400 \text{ cu. ft.}} = 0.75$
3 ft.	$\frac{125 \text{ sq. ft.}}{150 \text{ cu. ft.}} = 0.835$	$\frac{350 \text{ sq. ft.}}{600 \text{ cu. ft.}} = 0.583$

These ratios are decidedly unfavorable to the smaller converter, in which the surface coming in contact with the molten metal is 25, 33.3, and 43 per cent. greater per cubic foot of contents for 1 ft., 2 ft., and 3 ft. height of bath, respectively, than in the large converter.

7. Application of the large converter to the smelting of pyrite ores and concentrates, and the simultaneous converting of the molten sulphides to white metal or blister copper. Everything points to the successful application of the large basic-lined vertical converter to the smelting of pyrite ores and concentrates direct. The extent to which this practice can be carried out and the amount of the tonnage that can be smelted and converted without having recourse to a fresh supply of molten matte, depends principally on the iron content of the raw ore or concentrates charged.

XV. EXPERIMENTAL WORK

For study of the phenomena attending the blowing of air through a liquid, I constructed the glass model of a converter shown in Fig. 10.

I used a 13.5-in. diameter arc-lamp clear glass globe, with both ends open, having had the two rims ground parallel to each other. The wind box was made of lignum vitæ, the glass globe rim setting into a groove or recess in the wooden wind box, with a soft rubber gasket between wood and glass. A wooden cover, also of lignum vitæ, grooved and provided with a gasket in like manner as the wind box, fitted over the top part. This cover had a number of holes through which the air could escape, when liquids were poured into the vessel. The globe was clamped between the wind box and top by $\frac{3}{8}$ -in. rods in the manner shown. The tuyère plate into which the tuyères were set was screwed to the wind box, the joint being made water and air tight with a soft rubber gasket. A half-inch nipple, *N*, was screwed into the body of the wind box. A half-inch rubber hose connected the wind box with the half-inch air-supply pipe, the air supply being controlled by a needle valve. The air pipe was connected with a manometer consisting of a glass tube, a 1-m. scale divided into millimeters, and a bottle containing acidulated water, colored with methyl orange for better observation.

The tuyère plates were made of $\frac{1}{8}$ -in. steel, nickel plated to prevent rusting. To each plate were fitted 16 tuyères, $\frac{3}{32}$ in. internal diameter, each 1.5 in. long. The plates were drilled, the tuyère pipes set into the drill holes, and the joints soldered. One plate was fitted with the vortex tuyères, the other with straight vertical tuyères. The vortex tuyères, shown in Fig. 11, have three distinct angles with reference to the plate, giving them a forward, outward, and upward direction. The height of the liquid above the tuyère orifices was ascertained by filling the converter with 1 liter at a time, with the tuyères plugged, and cali-

brating the globe, the central rod, and taking actual height measurements of the liquid above the plates, so that by adding the liquid in amounts of 1 liter at a time, while an experiment was in progress, the

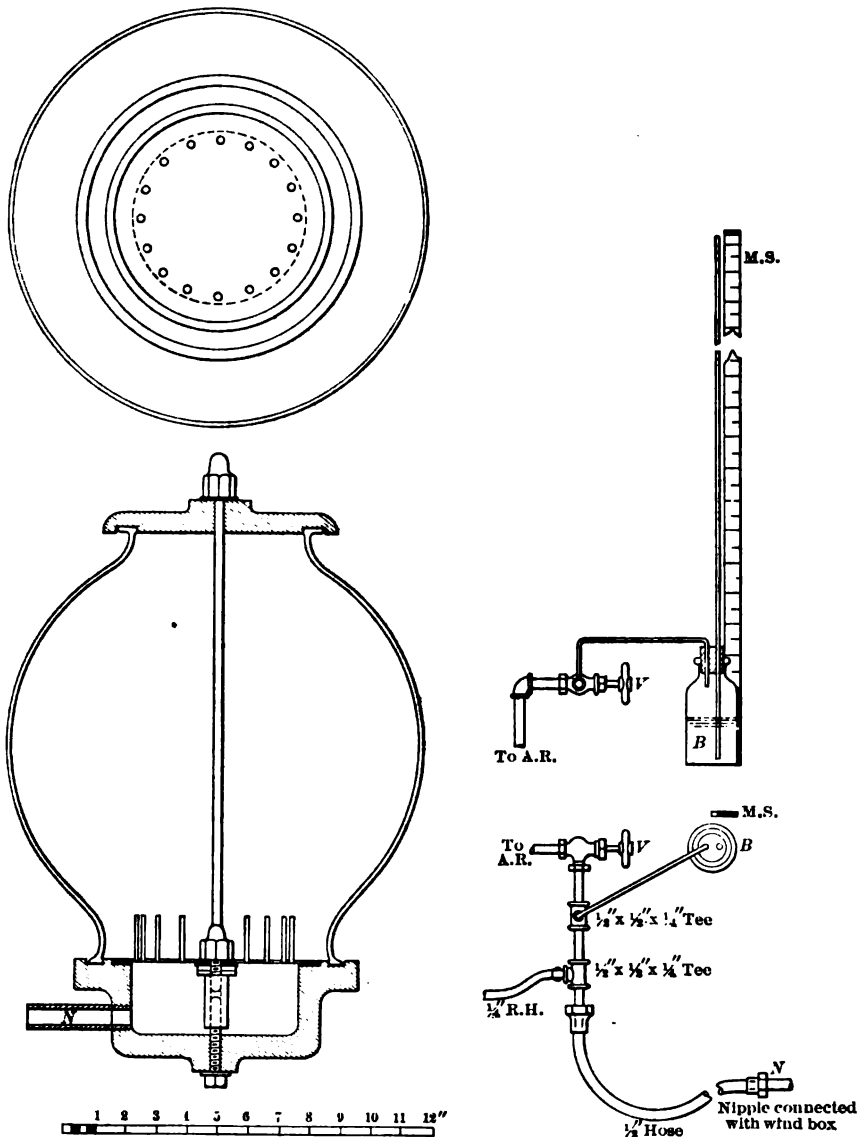


FIG. 10.—EXPERIMENTAL (GLASS) CONVERTER.

height of the liquid would be known. Necessary corrections were made in the difference of elevation of the vortex and vertical tuyères, to obtain the net static head.

The glass globe was held by a ring supported with trunnions in a wooden stand. When the vortex tuyères were used, positive rotation was imparted to the liquid, the speed of which decreased when the head was more than 50 mm.; and with a further increase in the head, the water was made so turbulent by the action of the air forced through it that the rotation became irregular. With water the speed of rotation varied from 5 to 18 ft. per minute. I could not find that a fixed relation existed between speed, air pressure used, and head. An interesting condition disclosed was that the speed of rotation in the central part of the converter was practically the same as at the outside. The arrangement of the tuyères in a ring produced a crest, also ring-shaped, where the largest portion of the air escaped, causing violent ebullition at that



FIG. 11.—VORTEX TUYÈRES.

place. Outside and inside of this geyser-like ring, the liquid was comparatively calm. Cork ball floats, slightly weighted with lead and covered with paraffine, were introduced, and their rotations timed with a stop watch. The outside floats rolled along the glass shell, the different perimeters of which at different heights of liquid could be quite accurately determined. This could not be so easily done with the float inside of the "geyser." But whereas the outside float traveled very slowly, the inside one traveled very much faster, and the diameter of the ring through which it traveled could be determined pretty closely by many observations. Then the distance traveled was ascertained.

The vortex tuyères⁴³ were originally devised for use in acid-lined con-

⁴³ H. Haas, U. S. Patent 832665, Oct. 9, 1906.

verters with the idea of setting the matte in rotation and thus securing uniform corrosion of the lining. It is also thought that the matte, sweeping over the tuyère orifices, would keep them free of noses. For basic-lined converters, the straight vertical tuyères can be formed in the tuyère bottom much more easily, and from thermo-chemical considerations it appears that tuyères of small area, with moderate air supply, will keep themselves open.

Earlier experiments made with the glass model indicated lower air pressures, for the same heads of liquid, with the vortex than with the straight tuyères; but this result was so contrary to hydrostatic laws that I was not satisfied with it; and I made another series of experiments later, to check, under more favorable conditions, the results first obtained. These experiments showed the pressures to be practically identical, under the same heads, for the vortex and the vertical tuyères.

I also used brine solutions, of specific gravity 1.2, which had no appreciable effect on the speed of rotation. Mercury, corresponding to a height of 680 mm. of water column, was also set in rotation, brass balls being used as floats. With mercury, the average speed of rotation was 5.8 ft. per minute for the outer circle. In the inner circle it was from 5.2 to 6.1 ft.

The tuyères were all of the same material, and had the same internal diameter, the same length, and the same size of orifice, so that the frictional resistances in the air pipe, tuyère pipes, etc., could not vary very materially for like air pressures and volumes. The manometer was set to a fixed air pressure, with the air discharging into the atmosphere. Then measured quantities of water were added, and after each addition of water the manometer reading was taken. The pressure did not increase in proportion to the head of water added, the ratios $\frac{\text{Head}}{\text{Pressure}}$ indicating the known fact that with an increase in head a relatively lower excess pressure is needed to balance the static column than with a low head. This is due to the fact that the work lost through frictional resistances is practically the same, whether a low or a high static column of liquid is opposed to the air pressure, and that with an increase in pressure frictional losses become less. These curves do not indicate that the amounts of pressure opposed to the heads as stated are necessary to avoid liquid flowing into the wind box. $\frac{H}{P} = 0.80$ was still sufficient to keep the tuyères open, but the rate of flow of air was naturally greatly lessened.

The tuyères, being confined to a central portion of the vessel, concentrated the air to that portion, where it rose, carrying with it a portion of the liquid, heavily charged with air. The action produced was that of centrally rising currents of air and liquid, rising beyond the surface and

overflowing toward the glass shell, outward. "Geyser" describes the action perhaps better than any other word I know.

An increase in the height of the liquid intensified the ebullition, which at times gave to the liquid a swashing motion.

An increase in the air flow, which in effect is the same as a restricted number of tuyères with a large air supply concentrated at a limited area, had the effect of throwing a larger amount of liquid up at the surface of it. With a further increase in the air supply, this would be violent enough to throw the liquids out of the globe.

Tilting the converter had invariably the effect of concentrating the air flow through those tuyères the orifices of which were nearest the surface of the liquid, *i.e.*, had the lowest static head opposed to them.

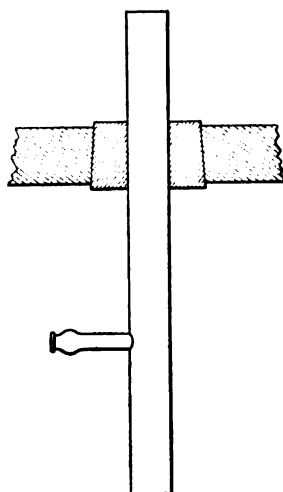


FIG. 12.—TUBE FOR STUDY OF EFFECT OF ENLARGING TUYÈRES.

To study the effect of an enlargement of the tuyères on the air pressure required to keep the liquid from running into the tuyère pipes, the following experiments were made: Five glass tubes were prepared, each provided with a small glass-tube connection on the side, which was connected with the manometer by a rubber hose and closed by a pinch cock. One end of the large tube was connected with the air-supply pipe, the other end was passed through a rubber stopper, fitted into a hole in the bottom of a glass cylinder or tall vessel. A sketch of the tube is shown in Fig. 12. The tubes had the following inside diameters: No. 1—2.8; No. 2—5.2; No. 3,—6.6; No. 4,—10.2; No. 5,—16 mm.

A measured amount of water was used, and the top of the rim of each glass tube was inserted the same distance into the glass cylinder, and the same head of liquid was opposed to each tube. The air pressure was then

gradually decreased, and the instant water began to flow into the tuyère, the manometer reading was observed. It was found that water ran into tube No. 2 at a 3 per cent. higher air pressure than into the smallest tube No. 1, and into No. 3 at a 7 per cent. higher air pressure than into No. 1. It was not feasible to continue the experiments further with the facilities at my disposal, since with the larger tubes, the concentration of the air volume was so great that the bulk of the water was thrown out of the cylinder, even with very moderate air pressures, notwithstanding the fact that the height of the cylinder was about 18 in.

The water invariably broke into the tuyère pipe near the rim of the pipe wall and adhered partly to the pipe wall.

If we consider the air column in a tuyère as a piston, moving forward and pressing upon another "air piston," we have a number of such air pistons, each pressing the superimposed one forward into the matte. Being composed of a gas, this "piston" is very elastic, and if we were to take actual pressure measurements, we would very likely find that the pressure is not uniform through the entire cross-section of the pipe. The gas molecules composing these "pistons" are in constant motion, and by enlarging the diameter of the gas column portions of these "pistons" may disintegrate and the liquid may break through them. To prevent this possibility a safe margin of excess pressure must be carried. This excess pressure demanded for safety may easily exceed the saving in power due to a lessening of the frictional resistances from the use of a smaller tuyère-pipe surface with tuyères of cross-section.

Large-scale experiments using water and different-sized air pipes, duplicating converter conditions, that can be closely observed, and from which accurate deductions can be drawn, would undoubtedly throw much light on these very complex phenomena, about which comparatively little is known.

Many of the observations made with the experimental glass model of a converter have already been discussed in a previous communication⁴⁴ in views regarding the action of air currents on matte in the converter. By actually observing in miniature the physical occurrences taking place in a model of a converter, one receives a much better understanding and a graphical picture of the action of the air in the large converter.

In conclusion I wish to thank Dr. Frederick G. Cottrell, Dr. L. H. Duschak, D. R. Kellogg of the U. S. Bureau of Mines, and Leroy E. Caverly for much valuable assistance and many suggestions given me during my experiments.

⁴⁴ *Bulletin* No. 83, Nov., 1913.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

Recrystallization of Limestone at Igneous Contacts

BY C. K. LEITH, MADISON, WIS.

(New York Meeting, February, 1914)

At the outset I would like to make it clear that I do not enter this discussion in a controversial spirit, but in an attempt to contribute something helpful to an understanding of a difficult problem. I have too high regard for the sterling quality of the work of the men who have studied this subject in detail to offer anything in the way of essential contradiction to their statements of fact. Difference of opinion arises from differing valuation of the possible alternative hypothesis which these facts suggest.

Some degree of recrystallization in limestone contact zones has been recognized by many investigators. Earlier investigators, for the most part, assigned an important, if not the most important, rôle to recrystallization in development of these zones. With the growing recognition of introduction of ores and gangue materials into the contact zone from igneous rocks through the medium of primary magmatic solutions, there has been a tendency to ascribe to this process most, if not all, of the chemical and mineralogical characteristics of the contact zone. This has involved a correspondingly diminished emphasis on recrystallization of substances already there as a factor in the process, and in some cases even the complete elimination of this hypothesis. From detailed study of a few contacts, casual observation of others, and a general familiarity with the literature, some of us have been led in recent years to raise the question whether the pendulum has not swung too far away from recrystallization toward direct introduction from igneous sources, and to argue for more recognition of the part played by recrystallization.¹ The inevitable sequence has been that those of us who have taken this view have been charged, at least by inference, with emphasizing recrystallization to the total exclusion of the alternative process. Scientists, like other men, like to classify and pigeon-hole views under simple and definite designations, leaving out qualifications which would tend to make the classification more difficult to state. This has made the problem seem more definite and simple, but has tended to obscure the fact that the disagreement is primarily not one relating to essential facts, but one of

¹ Barrell, Joseph: *Geology of the Marysville Mining District, Montana; a Study of Igneous Intrusion and Contact Metamorphism, Professional Paper No. 57, U. S. Geological Survey* (1907); Leith, C. K., and Harder, E. C.: *The Iron Ores of the Iron Springs District, Southern Utah, Bulletin No. 338, U. S. Geological Survey* (1908).

emphasis. The relative importance of processes seems to vary greatly in different districts. Until many more contacts have been carefully studied, agreement as to the relative importance of processes in general is perhaps not to be expected.

Evidences for recrystallization, briefly summarized and without qualification, are as follows:

1. So far as there is recrystallization it relates mainly to part of the silicate minerals and the residual carbonates of the contact zone. By no stretch can it explain the metallic minerals. The development of silicates from the lime or magnesia carbonates involves the elimination of all the carbon dioxide and some of the lime and magnesia, with recrystallization into silicates of part of the lime and magnesia together with other impurities which may be present, such as silica, iron, kaolin, and other substances. In certain districts the composition of part of the silicate zone (usually the outer part) corresponds approximately to the composition of the original carbonate rock, minus carbon dioxide and a part of the lime and magnesia. No analyzed samples have shown exact correspondence. It would be difficult to find exact correspondence because of later replacements, because of original variation of beds, and because of difficulty of confining sampling only to the recrystallized zone; but in some cases there is a remarkable tendency toward the constancy of silica-alumina ratios in comparison of original limestones and the supposedly recrystallized phases. The ratio is not absolutely maintained, but the variation in the silica-alumina ratio is slight as compared to the variations which are found in the parts of the contact zone in which materials have been clearly introduced. It would be remarkable if substances brought in at random from magmatic sources should approximate so closely the composition of residual impurities of limestone. A most striking case of this, which has been worked out quantitatively on a large scale, is the contact metamorphism of cherty iron carbonates by great masses of gabbro in the Lake Superior country. Here the iron-silica ratio of the altered phase corresponds almost exactly with that of the original carbonate rock, the change being merely an elimination of carbon dioxide. Analyses have been taken from many thousands of samples brought up in drill cores and in continuous sections across the formation.²

2. Secondary silicates of contact zones have often been found to be localized along cherty beds or around fragments of chert in the carbonate. Again the Lake Superior region furnishes an illustration in that cores brought from a depth of many hundred feet, where there has been no chance of surface alteration, and at some distance from the intrusive, show the development of secondary iron silicates, principally grünerite, along

² Van Hise, C. R., and Leith, C. K.: *The Geology of the Lake Superior Region, Monograph LII, U. S. Geological Survey*, p. 546 (1911).

contact of carbonate and siliceous layers, in a rock which is so dense there is little or no possibility for the introduction of these substances from without. The ratio of silica to iron has been almost exactly maintained. And yet these are clearly developed under influence of intrusives.

3. The similarity of secondary silicates in limestones and marbles far removed from igneous contacts to some of those developed at contacts is also suggestive evidence of recrystallization along contacts.

4. Elimination of carbon dioxide and lime is postulated under either hypothesis; "replacement" or "recrystallization." The natural consequence of elimination is recrystallization of the residual materials, whether or not these are supplemented by accessions from magmatic sources. There is no good *a priori* reason why accessions should always exactly balance elimination, especially when the physical conditions of intrusion are considered—and there is no satisfactory quantitative proof that they have. Under physical conditions which have been supposed to attend the earlier stages of intrusion of a magma it is easy to conceive of pore spaces caused by elimination to be closed as fast as formed, thereby reducing volume, and, in fact, it is usual to conceive of the pressure actually being a factor in the elimination. Under the replacement hypothesis we find it necessary to assume that whatever the pressure conditions were, whether those tending to close up openings or not, the materials taken out and those introduced were delicately balanced in volume; that just enough is introduced in any one place to take the place of that which is left.

5. The reduction in volume required by the recrystallization hypothesis cannot in most cases be disproved. So far as original textures are retained, as they are in some districts, then it is possible to infer, rightly I think, that the volume has not been considerably reduced, and, therefore, that elimination has not taken place except by equivalent introduction of new materials. But the supposedly recrystallized substances are usually in a structurally amorphous zone which may well be the residual of an original mass many times greater. Opponents of the recrystallization hypothesis have argued that the necessary elimination of substances, and consequent reduction of volume, is too large to be reasonable. The reasonableness or unreasonableness of this is a difficult point to argue. It is largely a matter of personal opinion. To me it does not seem inherently improbable. Elimination is equally necessary to the alternative hypothesis of introduction of the materials from magmatic sources. Without elimination it is necessary to assume an enormous increase in volume to take care of enough new material to give an average composition of the contact phase.

6. Discrimination of two phases of contact metamorphism is essential to an interpretation of conditions of formation of contact zones.

Students of contact metamorphism may to much advantage study

the mathematical theory of heat conduction as applied to an igneous contact. We are indebted to Professors Ingersoll and Zobel³ for an illuminating discussion of the principles of heat flow from an igneous rock of given dimensions into surrounding limestone. Their conclusions, which seem to be well based on general physical principles, are especially interesting in showing the remarkably slow progress of a heat wave into the limestone. Quoting from Ingersoll's and Zobel's discussion of a hypothetical case:⁴

"The conclusions to be drawn from the curves are: first, that the cooling is a very slow process, occupying tens of thousands of years; second, that the boundary-surface temperature quickly falls to half⁵ the initial value and then cools only slowly, and also that for a hundred or more years there is a large temperature gradient over only a few meters and a very slow progress of the heat wave; third, the maximum temperature in the limestone, or the crest (so to speak) of the heat wave, travels outward only a few centimeters a year. The mass behind it will then suffer a contraction as soon as it begins to cool, and the cracking and introduction of mineral-bearing material is doubtless a consequence of this."

Especially significant is the inference from the curves of heat flow that in advance of the heat wave the rock is tending to expand, therefore, to be compressed, whereas, following it during a long period of time there is contraction and the development of cracks. These conditions seem to favor two principal phases of contact metamorphism.

As the igneous mass advances into limestone it presumably is exerting mechanical pressure, judging by deformation at some contacts, and at the same time sending out heat into the surrounding rock, which, itself, increases the pressure. It is difficult to avoid the conclusion that for a time at least the adjacent rocks are under considerable pressure and that this pressure would favor elimination. It does not seem at all necessary or probable that under pressure this elimination should be immediately followed by introduction of other substances from the magma, or, putting it in another way, that substances from the magma should always so closely follow elimination as to replace molecule by molecule the original materials and thereby prevent any reduction of volume. As the crest of the heat wave advances into the surrounding limestone, lower temperatures follow, with the result that there is contraction and the development of openings. This contraction may effect not only the lime-

³ Ingersoll, L. R., and Zobel, O. J.: *An Introduction to Mathematical Theory of Heat Conduction with Engineering and Geological Applications* (Ginn & Co., 1913).

⁴ *Loc. cit.*, pp. 128 to 129.

⁵ Strictly speaking, the initial temperature of the boundary surface would be somewhat higher than this; for the conductivity of hot igneous rock is considerably greater than that of the cold limestone although, in order to be able to handle the problem, we have had to consider their thermal constants as the same. The temperature of the boundary surface for the first hundred years or so could best be estimated from equation (49) of Art. 80. The error introduced by assuming the diffusivities to be the same becomes less and less as the cooling proceeds.

stone but the intrusive itself. Into such openings the magmatic solutions may freely enter, and there are deposited the ores and some of their gangue materials. At the same time these solutions may replace the materials of the surrounding rock to a greater or less extent.

That contact metamorphism of limestone has been accomplished in two successive phases has been pretty well proved at certain contacts. It seems probable that when attention is directed specifically to this feature it may be found at others. The first phase seems to be characterized by the production of an amorphous, homogeneous, silicate mass, not definitely associated with fissures. In some cases this is discriminated sharply from, in other cases it merges gradually into, a phase characterized by sulphides and other ore-bearing minerals with their gangue materials, which occur much more largely in fissures. These fissures may often be seen to traverse the silicate zone of the first phase. The minerals of the later phase, both because of their composition and because of evidence of their transportation, cannot be regarded as recrystallizations of materials in place. They afford evidences of introduction from magmatic sources.

The two phases of alteration may merge one into the other both in time and place. The later phase may be expected to obliterate to some extent the earlier phase. Ordinarily the later minerals differ from the earlier ones, but certain silicates, quartz, and other minerals, may be common to both.

I do not attempt to cite evidences in detail from specific localities. My purpose is rather to outline the case for recrystallization. W. L. Uglow,⁶ in a recent paper, has cited evidences and references in some detail and in a forthcoming paper will cite more. I do not contend that all contacts will be proved to show important recrystallization or even that all of the illustrations cited in Mr. Uglow's paper are valid ones. I hold only that recrystallization has been proved in enough places and to sufficient degree to warrant its citation as a usual accompaniment of the process of introduction of magmatic materials. In some cases it seems to be important. In others evidence of it is slight or absent, though in these cases it may be masked by the introduction of materials in the second phase of contact metamorphism. Its complete absence in the nature of the case is difficult to prove quantitatively. Advocates of the recrystallization hypothesis have not, so far as I know, held that it was sufficient to explain all contact phenomena. They have offered it only as an explanation of one phase of contact alteration. Failure to consider this hypothesis involves failure to consider the possibilities of a two-phase alteration which seems to me to be the probable key to much contact metamorphism. With the majority of economic geolo-

⁶ Uglow, W. L.: A Review of the Existing Hypothesis on the Origin of the Secondary Silicate Zones at the Contacts of Intrusives with Limestones, *Economic Geology*, vol. viii, No. 1, pp. 19 to 50; No. 3, pp. 215 to 234 (Jan., Apr., 1913).

gists, I recognize the conspicuous evidence of the introduction of magmatic materials. My plea is that this hypothesis be not magnified to the exclusion of the recrystallization hypothesis. Quantitative studies of contacts based on adequate sampling have unfortunately been rare. Without them, conclusions can be only qualitative and not exclusive.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

Uniform Mining Legislation in All the States Based on Federal Act

BY C. L. COLBURN, DENVER, COLO.

(New York Meeting, February, 1914)

THE statutory requirements of the States and the district rules are, broadly speaking, similar. There is just enough difference to make it tantalizing.

DIFFERENCES IN STATE LEGISLATION GOVERNING THE LOCATION OF MINING CLAIMS

Discovery

Discovery is the first step in the location of the claim. The provision of the Revised Statutes (2320) is, "No location of a mining claim shall be made until the discovery of the vein or lode within the limits of the claim located." Strictly speaking, discovery may be defined as the finding of ore or metalliferous rock in place in a well-defined vein. This definition was too strict, so discovery extended later to cover such indications of the presence of ore, within rock in place, as would justify an experienced miner in spending his time and money with a reasonable expectation of finding ore in paying quantities. According to Colorado statutes a discovery must show a well-defined crevice; in Arizona, a lode, deposit, or mineral in place; in Montana, a well-defined crevice or valuable deposit; in Nevada, a lode or deposit of mineral in place.

It is wrong to require a discovery. An honest prospector, wishing to obey the law, finds himself compelled, in most cases, to evade this point. In Clear Creek district, Colorado, a quartz seam will constitute a discovery; in Butte, a red capping; in Leadville or Ely, it is any place where a man can find a piece of vacant ground big enough upon which to sink a 10-ft. hole. If the vein or deposit outcrops a man can show a discovery, but if it does not outcrop it is foolish to compel him to show one; and this point has been gently side-stepped. Any hole 10 ft. deep now constitutes a discovery. It is no use having laws that the people cannot consistently obey. No sane man will dig unless he expects to find mineral, and he should be protected while digging. He should be required to show that he has mineral ground only when he comes to patent.

Time to Sink Discovery Shaft and Complete Location

When a discovery has been made, the discoverer is entitled to retain undisputed possession of his claim for a reasonable time in order to complete the acts of location required by law. The length of time for this purpose is not fixed by the law of the United States, but is left to the law of the State or the regulations of the district. In Colorado, Idaho, Montana, North Dakota, South Dakota, Oregon, and Wyoming, he has 60 days to sink a 10-ft. discovery shaft and to complete his location. In Arizona, Nevada, New Mexico, and Washington, the time is 90 days. In Alaska, a discovery shaft is not required; in California and Utah, the matter is left to district rule. Most of the States allow the lode to be exposed in an open cut, cross-cut, or tunnel instead of the 10-ft. hole. Some States specify the amount of excavation work necessary to show the discovery.

Marking the Location of the Ground

The provision of the Revised Statutes (2324) is as follows: "The location must be distinctly marked on the ground, so that its boundaries can be readily traced." The law of the United States does not define what kind of marks shall be made, or where they shall be placed. In Colorado, the surface boundaries must be marked by six posts, one at each corner and one at the center of each side line, hewed or marked on the side toward the claim. In New Mexico, four substantial posts or monuments, set one at each corner of the claim, are all that is necessary. North Dakota requires two posts, one at the center of each end line, in addition to the six posts required in Colorado. Arizona requires six posts set 4 ft. above ground or stone monuments 3 ft. high. Montana requires posts 4 in. square by 4 ft. 6 in. long, 1 ft. in the ground, and a mound of earth or stone, 4 ft. in diameter and 2 ft. high, placed around each post. If a stone, not a rock in place, is used, it must be 6 in. square by 18 in. long, set two-thirds its length in the ground. In Oregon the posts need only stand 3 ft. above ground or there may be mounds of stone 2 ft. high. In Utah the details of marking are left to district rules.

Posting of Notice of Location

The rules and regulations of the several mining districts or States generally require the locator to post upon his claim a notice of his location. What this notice must contain is governed by the particular statute or regulation applicable in the district where it is made. Generally it contains the date, extent of claim, name of the claim, and name of the locator.

Record of Location Certificate

The final step in the location of a claim is the filing of the location certificate. This is not required by the U. S. law, and therefore is left to the States and districts. In Arizona, Idaho, Utah, Oregon, and New Mexico the statutes require the making of location certificates in duplicate, one to be placed on the claim and one filed for record. In the other States this is not required. In Montana and Idaho the location certificate must be sworn to before it will be admitted to record.

In the majority of cases the location certificate must be filed with the County Clerk and Recorder of the county in which the claim is located. In North and South Dakota the certificate is filed with the Register of Deeds; in Washington with the County Auditor. In some States the certificate may be filed with either the County Clerk and Recorder or with the Mining District Recorder; in other States the claimant must file with both the County and the District Recorder. Alaska is divided into three Recording Divisions which are subdivided into Recording Districts. Location certificates are filed with the Recorders of these districts.

Size of Claim

The maximum length of a lode claim is fixed by Congress at 1,500 ft. The width may vary from 50 to 600 ft., according to State or district regulation. North and South Dakota fix the width of a claim at 300 ft., but allow the counties to increase or decrease it within the Congressional limit. The width of claims in Colorado was formerly 300 ft. and in some counties 150 ft. Recently the legislature amended the laws, increasing the width to 600 and 300 ft., respectively, thus doubling the size in all sections of the State. In all the other States the width is 600 ft., so that a claim equals a little more than 20 acres. The tendency seems to be in favor of the larger claims.

SUGGESTIONS LEADING TO UNIFORMITY

Congress can make uniform laws concerning the appropriation of mining claims. This can be done by leaving nothing whatever for the States to do. A man wishes to take up a homestead. He does so under the Federal law and Land Office regulations. The States have nothing to do with the matter until the man "proves up" and receives patent to his land. It would be a bad mix-up if the States could regulate the size and shape of these homesteads, the amount of soil that the homesteader should till each year, and countless minor matters. The miner should be put in the same position as the homesteader. He should make his application directly to the government and should be relieved of State interference.

The Federal law of 1872 and the State statutes were all based on the theory that valuable minerals were found either in steeply dipping veins or in placers. The prospector in those days was looking for just such deposits. To-day mining conditions have changed. The old prospector has gone. Modern metallurgical practice has made it possible to treat low-grade ore at a profit. Mines are now sought and found not only in steeply inclined veins, but also in large irregular masses and in flat-lying deposits. This, then, is surely an opportune time for Congress to revise our mining laws.

The following suggestions will show how the mining laws can not only be made uniform, but simplified:

1. Mining claims should be taken up according to legal subdivision. Forty acres should constitute a claim.

2. Upon proper application the claim should be withdrawn from the public domain and held for the use of the claimant; just as farming land is withdrawn and held for the homesteader. No conflicting claim of any kind should be granted as long as the claimant obeys the law.

3. The claimant should be required to perform 1,000 cu. ft. of development work each year. Proof of this work must be filed before Feb. 1, for the previous year.

4. The claimant should be allowed to apply for patent after excavating 5,000 cu. ft. of material. He should at that time show that he has mineral ground.

5. The claimant should be compelled to apply for patent within five years.

6. After a patent is issued the claimant should be entitled to the surface and all mineral lying under it, and between planes passing through the center of the earth and the boundaries of the surface.

7. Failure to comply with any of the above requirements should constitute a forfeiture of the claim, and the land should be thrown open for other claimants.

8. One claimant should not have more than eight claims under application at any one time.

Laws based on the above suggestions would serve equally well for all kinds of mineral deposits and for all kinds of minerals.

In the case of oil and natural gas a claimant should be required to sink 400 ft. of well each year, and he should also be debarred from sinking a well within a fixed minimum distance from his boundaries.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.

The Classification of Public Lands

BY GEORGE OTIS SMITH, WASHINGTON, D.C.

(New York Meeting, February, 1914)

THE Secretary of the Interior in his recent report to the President has defined the new public-land policy, which is in fact "but a new application of an old policy." His words may be more acceptable on this subject of land classification than the repetition of phrases which have served some of us for several years. Secretary Lane says of the past:

"Congress has always been most generous as to the disposition of the national lands. . . . There was land for all, and it was the government's glad function to distribute it and let those profit who could. . . . And this generous donor was not so petty as to discriminate between kinds of lands, the uses to which they could be put, or the purposes which those might have who got them. Land is land, save when it contains minerals; this was roughly the broad principle adopted. To classify was a task too difficult or not worth while. The lands would classify themselves when they arrived in individual ownership. And so the door was opened for monopoly and for fraud."

Of the reaction and the evolution of a new policy, the Secretary continues:

"A reaction was inevitable. If lands were to be withdrawn from public service, why might not the government do the withdrawing itself? The old philosophy that 'land is land' was evidently unfitted to a country where land is sometimes timber and sometimes coal; indeed, where land may mean water—water for tens of thousands of needy neighboring acres. For the lands of the West differ as men do, in character and condition and degree of usefulness. We had not recognized this fact when we said 'land is land.' Lands fitted for dry farming and lands that must forever lie unused without irrigation; lands that are worthless save for their timber; lands that are rich in grasses and lands that are poor in grasses; lands underlain with the non-precious minerals essential to industry or agriculture; lands that are invaluable for reservoir or dam sites—these varieties may be multiplied, and each new variety emphasizes the fact that each kind of land has its own future and affords its own opportunity for contributing to the nation's wealth."

"So there has slowly evolved in the public mind the conception of a new policy—that land should be used for that purpose to which it is best fitted, and it should be disposed of by the government with respect to that use. To this policy I believe the West is now reconciled. The West no longer urges a return to the hazards of the 'land to land' policy. But it does ask action. . . . It asks that the machinery be promptly established in the law by which the lands may be used."

Following Secretary Lane's thought, the public lands need to be used, but it is equally important that they be used right. The day of accidental or incidental classification is past. Either disposition of these lands with their varied values should not precede classification or the terms of that disposition must be such as to neither overlook nor disregard the possible results of future classification.

As a matter of fact, two methods of wise disposition of public lands are possible; the one would require the determination of the land's highest use as the antecedent and prerequisite of any transfer of title from the government; the other method would limit whatever title is granted so as to give recognition to known or anticipated land characters. Either system would in theory recognize the principle of land classification and express its purpose of ultimate full utilization, but in practice the one system necessitates official classification before use, while the other permits immediate development without endangering full utilization. The one method means delay, the other involves limited titles.

Current public-land legislation, whether we review recent enactments or read proposed bills, is seen to express this idea of full and highest use of the land and abandons the old policy of "land is land." The various separation Acts of the past few years permit the agricultural use demanded by present-day needs, without sacrificing the future utilization of the mineral estate. The procedure seems both safe and sane for the government to issue the limited surface title both where classification has shown the existence of two or more estates in the land and where future investigation not now practicable may be expected to develop mineral possibilities.

Classification of land is simply the determination of its natural values as fixed by the use or uses to which the land can be put. The practicability of land classification is contingent upon the quantity of facts available to the classifier. The elimination of coal land from non-coal land, or of irrigable from non-irrigable land, or of mineral land from non-mineral land, may possess all degrees of certainty and accuracy. So it follows that where an official classification is possible because of a fortunately adequate knowledge of its character, the highest utilization both present and future can be at once insured by the separation of the different values and by their appropriate disposition. If, on the other hand, knowledge of the land's resources is at best only imperfect or can be expected only in the course of future settlement and development, then the disposition of

known values for present use should not await formal classification, but future classification should be provided for by the issue of limited patents.

Classification of public lands and separation of estates wherever lands have a double value is simply the common-sense method of safeguarding public interest against fraud in the case of known or determinable resources and against speculation in unearned increments in the case of resources at present unknown or only suspected.

Land classification is a business policy that we should adopt as individuals, were we landed proprietors; as citizens having special technical acquaintance with the public lands and their resources can we advise any course of action less favorable to the interest of our fellow citizens?

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Application of Electricity to Mines and Mills of Witherbee, Sherman & Co., Inc., Mineville, N. Y.

BY S. LE FEVRE, MINEVILLE, N. Y.

(New York Meeting, February, 1914)

THE application of electricity to the mining and beneficiation of the magnetic iron ores of the Mineville, N. Y., district, on Lake Champlain, has resulted in economies and enhanced quality of product, without which only a comparatively small quantity of the present output could be marketed. The output of the Witherbee, Sherman & Co. mines is about 700,000 tons per annum, and of this about four-fifths is treated by magnetic separation to increase the iron content and reduce the phosphorus.

The use of electricity for the magnetic separation of the ore from the gangue is therefore of primary importance, and makes possible the application of electric power for mining and preparing the ore for concentration. The fact which speaks loudest in favor of the application of electricity is this: It is possible for about 25c. per ton of shipping ore to furnish all the power required to put the ore on cars. This includes mining a hard ore underground, from a depth of 900 ft., hoisting to surface, crushing to $\frac{1}{4}$ or $\frac{1}{8}$ in. in size, and separating from the gangue. This also includes lights about the works, and all miscellaneous power, such as about the shops.

The first plant for the generation of electric power for use at Mineville was completed in 1903, and was one of the first for the application of electricity to mining iron ore. This was a Nordberg compound condensing steam engine, direct connected to a 750-kw. alternating-current generator, 3,300 volts, 3 phase, 25 cycles. All the plants added later use the same current but with higher voltage. The map, Fig. 1, shows the power plants and the transmission system; Figs. 2 and 3 are views of the shaft houses and the mills.

The power from the first plant was distributed to Mills 1 and 2, one 2,500 cu. ft. compressor, and to the Harmony hoist, which was driven by a 300-h.p. motor operating four 10-ft. drums. Our experience with the hoist covers largely the development of electric hoisting from 1903 to date, and in some cases improvements were worked out to satisfy our needs. The motor is still operating as installed, but the control apparatus has been radically changed several times. When installed, the iron re-

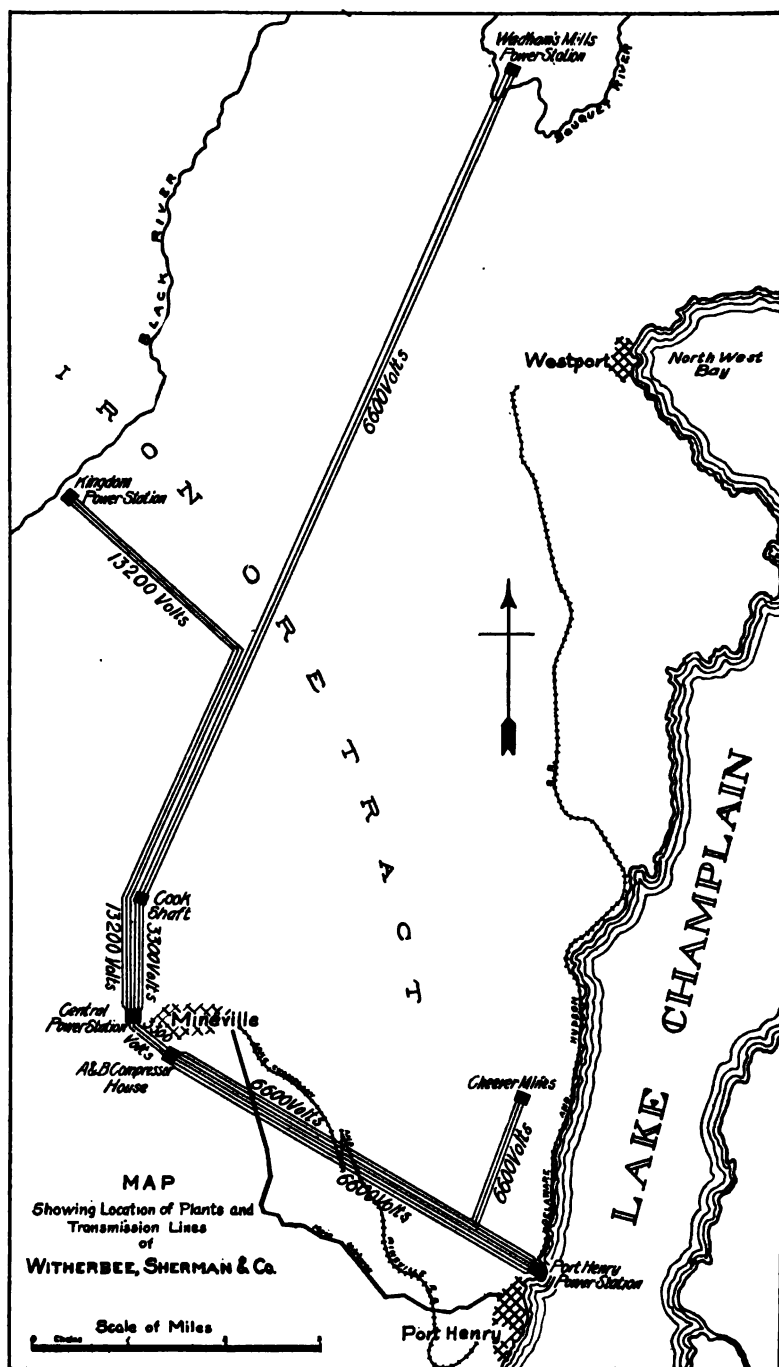


FIG. 1.—MAP SHOWING LOCATION OF PLANTS AND TRANSMISSION LINES OF WITHERBEE, SHERMAN & Co.

sistance grids were light, and they would vibrate when the rush of current entered, sufficiently to touch and short circuit, and burn out; it was a



FIG. 2.—JOKER, BONANZA AND CLONAN SHAFT HOUSES, MILLS NOS. 1 AND 2, AND NO. 1 POWER STATION.

regular thing to replace those burnt out, between shifts. We then increased the weight about four times and got very good results. Still later we installed a water rheostat, which has been in operation about two



FIG. 3.—VIEW OF NO. 4 MILL.

years and has cost practically nothing in repairs, besides being very much smoother in operation and acceleration.

The heavy duty on the control of this hoist is due to the four drums attached to one motor. When one car is hoisted slowly for men, and the other cars have ore in them, the motor gets full load at slow speed and a very heavy current must be carried by the resistance in the rotor circuit.

As the depth and output increased the 300-h.p. motor was overloaded, and the delays in hoisting due to handling men were considerable. On that account a new double-drum hoist is now being installed, and two drums of the old hoist will be discarded.

The hoist serving the Joker shaft (Fig. 4) has double drums, 10 ft. in diameter, 6-ft. face, geared to a 500-h.p. motor, and hoists 3 tons



FIG. 4.—DOUBLE-DRUM HOIST FOR JOKER SHAFT IN NO. 1 POWER STATION.

of ore in each skip at 900 ft. per minute, the two skips being in balance. It takes 1 kw. of power to hoist 1 ton of ore 375 ft. This hoist will deliver forty 3-ton skips per hour from a depth of 900 ft. This allows 3 min. for a round trip of each car.

The ore includes lumps up to 2 or 3 ft. in diameter mixed with fines. This is drawn out of a pocket, through a door 4 ft. square, opened and closed by a 12-in. diameter air lift, into a measuring chute 12 ft. long on an angle of 45°, at the bottom of which is a lip chute raised to hold back the skip load. As soon as the car lands, the lip chute is dropped, the car loaded, and the signal given to hoist in from 15 to 30 sec. The

measuring chute is then filled while the skip goes up and back. A little skill in following the descending car with the loading lip chute often makes the signal to hoist reach the engine room at the same instant with the roar made by the dumping of the ore from the other car in the shaft house. Heavy cast-iron grids are used for resistance on this hoist and have given no trouble. Double-pole contactors are used, which are operated by a master controller on the operating platform (Fig. 5). The hoisting costs about 2c. a ton for power. With the steam hoist the cost was 7c. per ton, a saving of 5c. per ton in favor of electric hoisting.

We are now installing four new hoists underground. The depth of the workings and the shape of the ore bodies make it advisable to

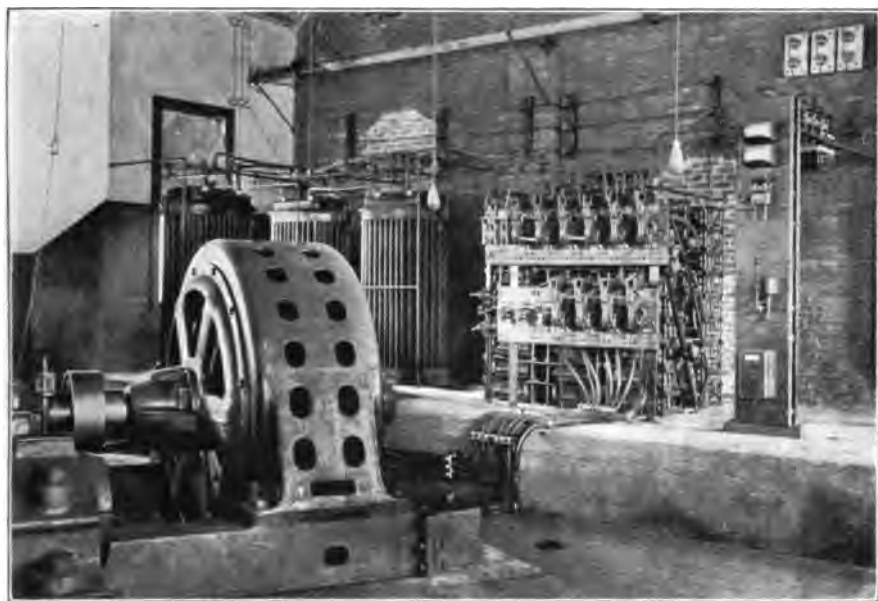


FIG. 5.—TRANSFORMERS AND CONTACT OF PANEL FOR JOKER HOIST MOTOR.

divide the hoist. The underground hoists operate on slopes from 25° to 40° from horizontal, and dump into pockets, from which the ore is reloaded into skips and hoisted up shafts which are vertical or nearly so.

These new hoists, shown in Fig. 6, a shop view, have double drums 7 ft. in diameter, 3-ft. face, with a single reduction of Wuest herringbone gears to the shaft of a 300-h.p. motor. This is arranged so that if desired to operate to capacity without working in balance, another 300-h.p. motor can be added on the opposite side. The hoists were all sectionalized to go down a 4 by 4 ft. compartment in shaft.

These hoists will be equipped with a new type of liquid rheostat

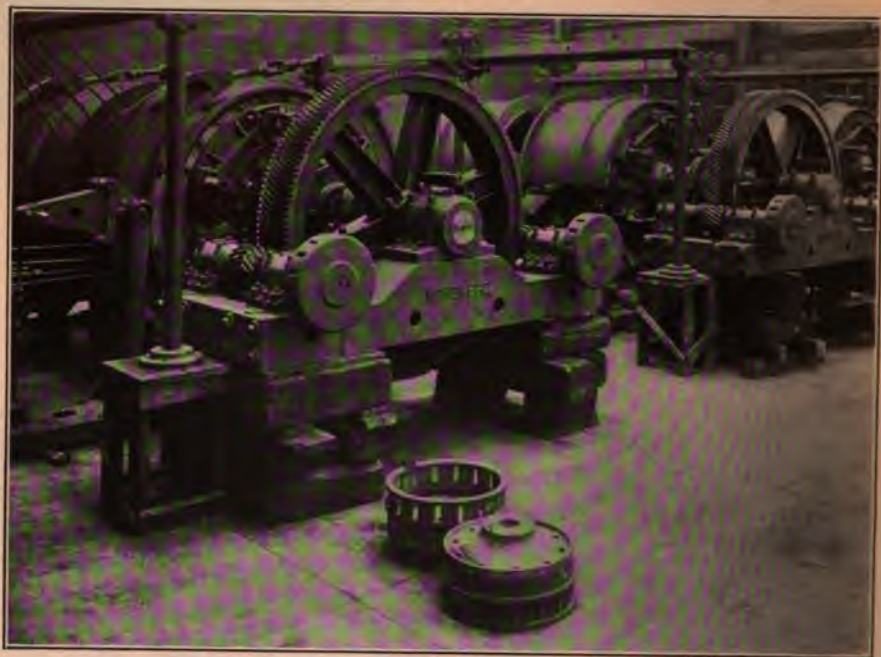


FIG. 6.—Shop View of Hoists to be Installed Underground at Mineville.

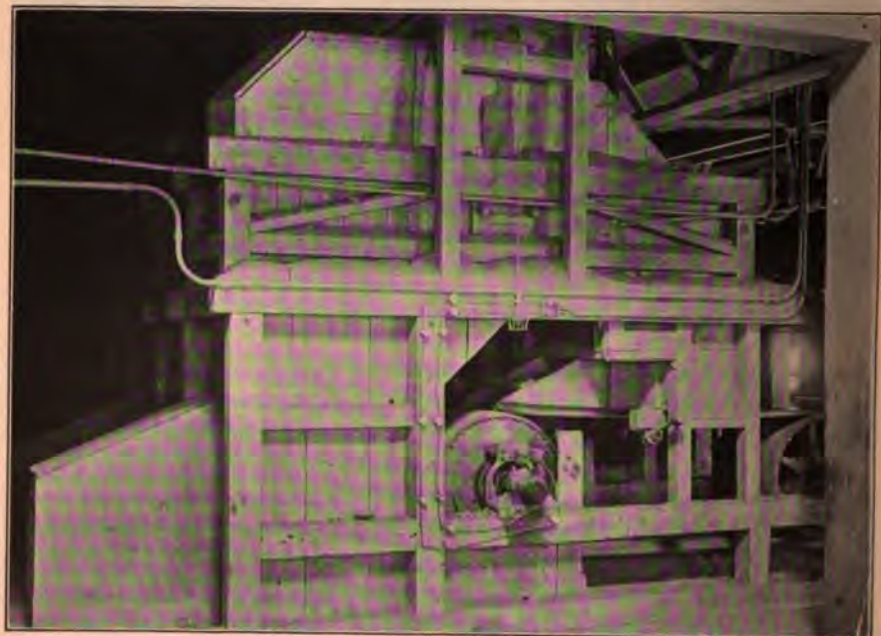


FIG. 7.—DRUM TYPE OF MAGNETIC ORE SEPARATOR IN USE AT MINEVILLE.

which has two sets of electrodes, one set of pipes which are spaced far apart and are used for starting, and another set of plates which are spaced close to give a very small slip when full speed is desired. This eliminates the danger of the rheostat arcing over if the motor is suddenly reversed, and also gives a much wider range of speed than the ordinary type of rheostat.

The separators used are the Ball-Norton type as developed by S. Norton at Mineville, N. Y., and are simple in construction and operation; they handle large quantities. Three types are in use: the drum machine (Fig. 7), the belt machine (Fig. 8), and the pulley type, or rock picker. The drum machine is used on material from 2 in. to $\frac{1}{2}$ in.,



FIG. 8.—BELT TYPE OF ORE SEPARATOR IN USE AT MINEVILLE.

although it will work on finer sizes, but the belt machine is preferred on $\frac{1}{4}$ in. or finer, as it costs less and is more open for examination and repairs.

The drum machines are usually 30 in. in diameter, 15-in. face, with brass shell revolving about the face of the magnets. These are suspended in a fixed position inside the drum, taking up about half of the periphery.

The belt machine has the magnets suspended in a horizontal position over the take-off belt, and under the return of same. The feed belt runs below the take-off belt, and when the ore comes under the first

magnet, which is only a few inches from the end of the feed belt, it is lifted up to the under side of the take-off belt and carried along until it reaches the last magnet, where it drops over a division board into the concentrates. The tailings drop into the space between the end of the feed belt and the division board.

The magnets are so connected in the wiring as to be of alternating polarity. One end of a particle of ore is attracted to a magnet, and as the belt carries it forward the other end turns to the following magnet, thus turning the stream of ore over and over, and leaving the gangue free to drop, instead of being carried along enmeshed in the ore.

The pulley type machine, or rock picker, is used in the head end of a conveyor, and has a full circle of magnets, which revolve with the pulley. This is used with a heavy current, and holds to its face anything with ore in it. The pure rock flies off over a division board set ahead of the pulley.

In making the concentrates from the Mineville ores, that from the Old Bed mine is crushed to $\frac{1}{4}$ in., but this is because the chief object is to reduce the phosphorus. For the Harmony and New Bed concentrates, the most economical practice is to pass the crude ore, after crushing to a maximum of 2 in., over a drum machine, which picks out the rich ore, as heads or concentrates, leaving the balance as middlings, which are passed over a pulley machine and by this divided into tailings and middlings. The middlings from the pulley machine are then passed through rolls, and the same process is repeated. When the material is reduced to $\frac{1}{4}$ in., belt machines are substituted for the drum machines.

The use of the pulley machines reduces materially the amount of work the rolls have to do, and also the bulk of material to be treated by the final machines. This treatment also recovers the ore in the coarsest possible state, consistent with maintaining the desired percentage of iron.

We make occasional car loads of concentrates, used for magnetite arc lights, guaranteed to contain 70 per cent. metallic iron, and have reached 71.2 per cent., leaving only 1.2 per cent. of total impurities, as pure magnetite carries 72.4 per cent. iron.

Direct current is used for the magnets at about 125 volts, and the amount of current on the individual machines runs from 3 to 7 amperes on the machines making heads, and up to 25 amperes on the pulley machines. The adjustments on individual ores to get the desired results are variations of the current, air gap, and belt speeds.

A table is appended showing the average results obtained for the year 1913 in the various mills. The recovery runs from 90 to 98 per cent. of iron in the crude, as concentrates containing between 62.5 and 66 per cent. metallic iron.

Slip-ring induction motors are used principally, and with reversible

controller, which is very desirable when putting on belts, or when crusher or rolls block. In this case a slight reverse releases the material.

We have been using 220-volt direct current for the mine lighting, and the electric haulage used current from the same circuit. In the near future a transformer will be installed to reduce the alternating current to 110 volts and this will be used for lights, which will then be taken off the trolley circuit. Clusters of incandescent lights are used in chambers instead of arc lights.

The distribution of current about Mineville is at 3,300 volts, with transformers from 3,300 to 440 near each motor or group of motors.

All of the electrical apparatus of the present equipment is of General Electric manufacture, and the plant was described in the *General Electric Review* of February, 1912, to which we are indebted for some of the illustrations used herewith. We wish to acknowledge also valuable assistance received from H. F. Pigg, Mechanical and Electrical Engineer for Witherbee, Sherman & Co.

The tables appended give the generating units, having a total capacity of 4,975 kw., and a recapitulation of the distribution of the power for the year 1913, also lists of the motors installed.

The generator capacity is about 70 per cent. of that of the motors installed, and the power used is about 60 per cent. of the motor capacity. The average operating cost of power was \$0.0091 per kilowatt-hour, and with depreciation of plants added \$0.0105.

Compressed air is the principal item of power cost and took 10.5 kw-hr. per ton of crude ore mined, or 73½ per cent. for compressed air out of a total for mining of 14.3 kw-hr. The milling took 4.1 kw-hr. and the total power for all purposes was 18.7 kw-hr. per ton of crude ore produced.

The distinct advantage which magnetic ore has over the other marketable varieties lies in the ease and certainty with which it can be separated from the gangue, after the particles of ore are crushed free from the rock. In this respect the Lake Champlain ores have a great advantage on account of coarse crystallization, so that crushing to $\frac{1}{4}$ or $\frac{1}{8}$ in. permits a recovery of from 90 to 98 per cent. of the iron in the crude in the form of a uniformly high-grade concentrate with a small percentage of objectionable fines. The quality of the concentrate can be made to stay practically constant at whatever point may be fixed as the best economic limit, without regard to the iron content of the crude ore, whether it be 25 or 55 per cent. iron, which is about the range of the ores treated.

This advantage ought to increase very materially the amount of magnetic ore used, as the demand for increased ore tonnage continues to grow. The quantity of iron ore used doubles every ten years. To provide this, even if the amount of imported ore grows largely, prepara-

tion must still be made at a rapid rate for new mines and increased outputs of present mines.

Concentration Results, Average, 1913

	Crude Fe, Per Cent.	Tails Fe, Per Cent.	Concentrates		Recov- ery, Per Cent.	Tons Crude	Tons Concs.	Ratio
			Fe, Per Cent.	Phos., Per Cent.				
<i>Old Bed concentrates.</i>								
Mill No. 1.....	54.38	5.30	63.75	0.74	98.42	87,157	73,190	1.191
Mill No. 2.....	55.21	7.88	65.15	0.67	97.53	170,161	140,576	1.21
Cobbing plant.....	53.41	6.98	60.71	1.26				
<i>New Bed concentrates.</i>								
Mill No. 4.....	34.16	6.25	65.04	0.027	90.41	192,137	91,230	2.106
<i>Harmony concentrates.</i>								
Mill No. 3.....	47.79	7.34	62.67	0.087	95.93	223,787	163,854	1.366

Equipment of Power Plants

Power Plant		Generators		Amount Pro- duced, 1913
Location	Drive	Kilowatts	Volts	Kilowatt-hours
Port Henry.....	Turbine.....	800	6,600	7,453,700
	Turbine.....	1,500	6,600	
Central.....	Engine.....	750	3,300	4,306,800
	Turbine.....	750	3,300	
Total steam.....		3,800		
Wadhams.....	Water wheel.....	375	6,600	788,300
Kingdom.....	Water wheel.....	800	13,200	1,870,700
Total.....		4,975		14,419,500

Recapitulation—Distribution, 1913

	H.P. of Motors Installed	Kilowatt-hours		Per cent. of Full-Load Capacity Used
		Total Used	Per Ton Crude Ore	
Compressors.....	2,100	7,273,365	10.5	93
Hoists.....	2,200	1,664,956	2.5	20
Hoists, new.....	1,200			
Pumps.....	131	287,554	0.4	60
Haulage.....		212,063	0.4	54
Mine lighting.....	275	333,826	0.5	
Total mining.....	5,906	9,771,784	14.3	55
Mills.....	2,094	2,756,408	4.1	76
Carpenter shop and saw mill	95	89,019	0.13	
Sherman mine development.....		98,084	0.14	
Total power used.....	8,095	12,715,295	18.7	60
Commercial power.....	1,150 } 50 }	1,704,205		
	9,295	14,419,500		

Compressors—Motor Distribution

Location	Horse Power	Volts	Service
A and B power house.	400	3,300	2,500 cu. ft. compressor, synchronous motor.
	400	3,300	2,500 cu. ft. compressor, synchronous motor.
	400	3,300	2,500 cu. ft. compressor, synchronous motor.
	200	440	1,250 cu. ft. compressor.
	200	440	1,250 cu. ft. compressor.
No. 1 power house...	200	440	1,250 cu. ft. compressor.
Smith mine.....	75	440	600 cu. ft. compressor.
	75	440	600 cu. ft. compressor.
Sherman mine.....	150	440	600 cu. ft. compressor.
Total.....	2,100		

NOTE.—Synchronous motors are operated at 80 per cent. power factor leading, which brings up the power factor at the distributing points in Mineville to about 93 to 94 per cent.

Hoists—Motor Distribution

Location	Horse Power	Volts	Service
No. 1 power house...	500	440	Double drum—Joker shaft.
	300	3,300	Double drum—Bonanza shaft.
A and B power house.	300	440	Double drum—B shaft.
	300	3,300	Double drum—A shaft.
Smith mine.....	100	440	Double drum.
Sherman mine.....	52	440	Single drum.
<i>Underground</i>			
Barton Hill.....	52	440	Single drum—Intermediate sink.
	52	440	Single drum—South pit.
	52	440	Single drum—South pit.
	35	440	Single drum—Orchard.
	22	440	Single drum—South pit raise.
	22	440	Single drum—South pit raise.
	22	440	Single drum—North pit raise.
Harmony.....	52	440	Single drum—E. side, A sink.
	52	440	Single drum—W. side, A sink.
	52	440	Single drum—B sink.
	22	440	Single drum—A pocket.
	300	440	Double drum—A shaft circle.
	52	440	Man hoist—A shaft circle.
Old Bed.....	35	440	Single drum—Miller pit sink.
	22	440	Single drum—Miller pit raise.
	52	440	Single drum—Slope No. 1—To be replaced.
	52	440	Single drum—Slope No. 2—To be replaced.
	52	440	Single drum—Slope No. 3—To be replaced.
	300	440	Double drum—Slope No. 1—Partly installed.
	300	440	Double drum—Slope No. 2—Partly installed.
	300	440	Double drum—Slope No. 3—Partly installed.
Total.....	3,452		

Pumps—Motor Distribution—all 440 Volts

Location	H.P.	Service
Smith mine.....	30	
	5	Bottom.
B shaft.....	30	Circle.
	15	Bottom.
A shaft.....	12	Circle.
	5	Bottom.
Old Bed.....	5	No. 6.
	3	No. 4.
Barton Hill.....	3	
	3	
Surface.....	15	B shaft water supply.
	5	B shaft water supply.
Total.....	131	

Power Houses—Motors

Location	H.P.	Volts	Service
Port Henry.....	50	440	35-kw. motor-generator exciter.
	6	440	Ventilating fan for 800-kw. turbine.
Central.....	50	3,300	Motor-generator exciter.
	5	440	Centrifugal pump for condenser.
	10	440	Ash conveyor and elevator.
No. 1 power house...	110	3,300	75-kw. motor generator for direct current —220 volts.
	50	440	35-kw. motor generator for direct current —220 volts.
	110	3,300	(These two for lights and haulage.) Two 37½-kw. D.-C. generators, 125 volts (Spare).
Direct-current motors	22		Hoist in Old Bed, Overland.
	22		Hoist in Old Bed, Overland.
	5		Pump at Bonanza circle.
	50		Five locomotives, two 5-h.p. motors each, trolley haulage.
	9		Two locomotives, one 4½-h.p. motor each, tailings cars.
	20		Three-car tippie, Joker shaft.

Mills—Motor Distribution—All 440 Volts A. C.

	H.P.	
Mill No. 1.....	75	Group drive.
	75	Group drive.
	75	Group drive.
	60	Group drive.
	35	Crusher—Joke shaft house.
	75	Cobbing plant.
Total.....	395	
Mill No. 2.....	100	Group drive
	100	Group drive.
	100	Group drive.
	30	Hoist for hauling cars.
	15	Conveyor to bin.
	50	35-kw. motor generator, 125 volts, magnet current, and mill lights.
Total.....	395	
Mill No. 3.....	200	Section $\frac{1}{2}$ mill.
	200	Section $\frac{1}{2}$ mill.
	50	35-kw. motor generator, 125 volts.
	35	Crusher and conveyor, B shaft.
	20	Conveyor to mill.
	35	Crusher, A shaft.
	20	Conveyor to mill.
	5	Elevator.
Total.....	565	
Mill No. 4.....	150	Group drive.
	150	Group drive.
	150	Group drive.
	150	Group drive.
	20	Tailings conveyor.
	2	Repair shop.
	2	Fan for drier.
	35	Coal-car hoist.
	5	Elevator.
	75	50-kw. motor-generator set, 125 volts, for D.C., for separators and lights.
Total.....	739	

Miscellaneous Motors

	H.P.	Volts	
Saw mill.....	60	440	Saws.
	35	440	Carpenter shop and planer.
Smith mine.....	6	440	Fan for drill forge.
Sherman mine.....	6	440	Fan for drill forge.
Fertilizer plant.....	60	440	Tube mill and conveyors, under construction.
Commercial power: Mineville Light H. & P. Co. Street lights. Residence lights.			
Cheever Iron Ore Co.	400	6,600	Compressor.
	200	6,600	Compressor, synchronous motor.
	225	440	Hoist.
	75	440	Hoist.
	37½	440	Hoist.
	37½	440	Hoist.
	37½	440	Hoist.
	15	440	Repair shop.
	5	440	Repair shop.
	75	440	Mill.
	50	440	Mill.
	25	440	Mill.
	16	440	Pumps.
	10	440	Haulage locomotive.
Total.....	1,208½		

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Mining-Law Revision: How to Obtain It

BY EDMUND B. KIRBY, ST. LOUIS, MO.

(New York Meeting, February, 1914)

THIS meeting marks the point at which the long-standing dissatisfaction with the mineral-land laws, the innumerable protests against them, and the many isolated efforts to obtain relief, have developed into a general movement for their revision.

That part of the mining industry which is affected by these laws has three general organizations which are able to speak for it: the American Institute of Mining Engineers, the Mining and Metallurgical Society of America, and the American Mining Congress; all of which are represented in this meeting.

In the series of able and thoughtful papers presented here, prominent men have discussed the evils of the present laws and some of the reforms which should be made in them. We all appreciate fully the ultimate value of these studies; but we are also painfully aware of the fact that discussion of this kind has been going on for a quarter of a century without any other effect than a steady increase in the mass of printed records stored on library shelves.

In order to get any practical result we must induce an indifferent government at Washington to act, and to do this we must unite in pushing some practical plan whereby it may act in a simple and effective way and with the least possible amount of criticism or opposition.

The practical question before us is not whether the mining laws are bad; evils and absurdities are not only beyond discussion but also beyond the resources of a temperate and dignified vocabulary. It is not particular changes ought to be made; for it is premature to make recommendations until there is some one to listen to them. But, we all know that there is wisdom and experience enough available, and awaiting that time, for the creation of a code which would be a model to the mining world.

The practical question now is, how to obtain the revision; and since the American Mining Congress has of late restricted its attention to this phase of the problem I wish as one of its representatives here to present its views upon the matter.

The American Mining Congress is engaged in the work of inducing Congress to undertake a general revision of the mineral-land laws and to this end it is urging the appointment of a commission whose members shall be selected for their recognized knowledge and experience in the mining industry. This commission shall consider the mining laws of this and other countries and shall hold public hearings in the principal mining communities of the West and Alaska; giving full opportunity for the expression of public opinion concerning the problems before it. Its recommendations shall be presented to Congress in the form of a fully drafted mining code. It is not proposed to have this include deposits of coal, oil, phosphates, or salines, because these are generally regarded as the subjects of other and special legislation.

The American Mining Congress is therefore not attempting at present to formulate and recommend specific changes in the mining laws. It is concentrating its efforts upon the creation of machinery which will do this detailed work of the revision and will do it in a way calculated to inspire the confidence of the mining communities and appeal to the common sense of all men.

The definite plan of action thus proposed is based upon the conviction that no tinkering or patchwork revision can afford any perceptible relief. The present mining laws as a whole are hopelessly at variance with the geological realities of ore-deposit structure and also with the practical operations of prospecting and mining. Moreover, their various parts are so interdependent that it is practically impossible to correct individual faults without revising the laws as a whole.

The fact is also recognized that the problems involved are very technical and difficult, and that mining men are in much fear of changes framed by legislators who do not know these technical peculiarities or the practical effects of mining laws upon the industry. This apprehension has led to the prolonged endurance of present evils until these have become disastrous to the industry. It is therefore necessary for safety and for the quieting of this fear that Congress, in making the revision, should assemble and utilize the best wisdom and experience to be found among mining men. This is, moreover, the only course which will enable it to make a revision which will be satisfactory.

It is believed that no revision can be carried through which does not gain for itself the acceptance and support of the mining communities affected by it, and a working plan, to be effective, must provide a practical way of securing this acceptance. They are already in accord as to the evils of the present laws and the fact that a revision is necessary. With respect to the details of the changes required, however, there are natural differences of opinion, and it is necessary to compose such differences by bringing them all to a focus before some authoritative body which has the power of decision. This body or commission must necessarily have a

personnel which will be known to the mining communities and inspire their confidence.

It is also believed that the commission will find the most important part of its work to be the conduct, through its public hearings and otherwise, of such a campaign of publicity and discussion as will carry the mining public with it to the same conclusions. This would be accomplished by giving the public, in one way or another, full opportunity to talk itself out upon the questions at issue.

The suggested draft for a joint resolution of the Senate and House, which is given herewith, embodies the essentials of the plan proposed, which are as follows:

1. That the revision shall be general.
2. That the membership of the commission shall be restricted to men whose qualifications for the work are known to the mining industry.
3. That the commission shall hold its public hearings within reach of the mining communities and give the public full opportunity to express its views.
4. That the commission shall have some reasonable time limit set upon its task.

These four essentials require enough expansion to be self-explanatory, but prudence dictates that the temptation to add details should be resisted. Practical experience has shown that every detail introduced into a measure of this kind tends to create differences of opinion and to develop opposition, and the draft which will meet with the most favor is that which best presents the bare essentials, unencumbered by any details.

The work already done by the American Mining Congress has made it clear that the procedure described appeals to the judgment of the great majority of men and reduces to a minimum the number of objectors. In fact, it is believed to be the only practicable plan for obtaining the end desired.

PRELIMINARY DRAFT SUGGESTED FOR A JOINT RESOLUTION OF THE SENATE AND HOUSE

That Congress shall undertake a general revision of the laws relating to mineral-bearing lands and mineral rights within the United States and Alaska, and such revision shall cover mineral deposits of every kind except those of coal, oil, phosphates, and salines, which are set aside as the subjects of other and special legislation.

In view of the technical nature of the problems presented by the work it is desired to secure first the results of the knowledge and experience which exist among those who are engaged in the mining industry.

To this end the President shall, within 60 days hereafter, appoint a

commission of five members, who shall be selected for their recognized knowledge and experience in the mining industry.

The commission shall consider the mining laws of this and other countries and shall hold public hearings in the principal mining centers of the Western States and Alaska, giving full opportunity for the expression of public opinion concerning the problems before it. Its recommendations shall be presented in the form of a fully drafted mining code.

Within six months after the appointment of the commission its report shall be delivered to the President, who shall within 30 days thereafter transmit it to Congress with his further recommendations, if there be any.

Members of the commission shall receive \$—— per diem, with expenses, and shall engage such clerical assistance as may be necessary for the work.

Clause providing for the necessary appropriation.

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The Location of Mining Claims Upon Indian Reservations

BY WILL L. CLARK, JEROME, ARIZ.

(New York Meeting, February, 1914)

HE who enters a mining claim within an Indian reservation of the United States of America acquires no rights thereby, because of the fact that the lands within such Indian reservation are not a part of the public domain, and cannot be located or acquired as mineral land. However, where such an entry is authorized as to a particular reservation by an Act of Congress, such claims as may be entered under such authorization are thereby segregated from the reservation, and the Indian title is extinguished.

Under the Choctaw Constitution, any citizen of that nation, in other words any Choctaw Indian, who discovered a coal mine, acquired an exclusive right to all coal within a radius of 1 mile.

The laws of the Chickasaw nation provided for the formation of corporations to develop coal and other mines, with authority to contract with others to develop and work the mines. Under these laws, leases were made; but Congress subsequently abrogated all existing leases, prohibited all persons from receiving royalties from such mines, and provided that all coal within the nation should remain the common property of the tribes. Such leases are now expressly prohibited by an Act of Congress (32 United States Statutes at Large, 655). All leases of mineral lands upon Indian reservations must now be made under regulations promulgated by the Secretary of the Interior, and the royalties must be paid into the United States Treasury for the benefit of the tribes.

In Canada the Indians have no rights to the royal mines and minerals, and the government can make no stipulation with the Indians which would affect the rights of the Province in which the lands are situated.

Therefore, while it is true that under our present law no rights can be acquired to a mining claim within an Indian reservation, it seems to be fully within the authority of Congress to make laws, giving such rights to mineral locators as it may deem necessary or expedient.

There is a demand for a general law that will permit mining upon Indian reservations, and I am informed that most of the members of both houses of Congress from the Western States are in favor of Congress pass-

ing a bill which will permit the working of mineral claims discovered upon Indian reservations.

The following letters from members of the Senate and House of Representatives are quoted in support of this assertion.

HOUSE OF REPRESENTATIVES,
WASHINGTON,

August 15, 1913.

"There is a great demand for a general law that will permit mining upon Indian reservations. As the law now is, the Secretary of the Interior cannot permit metal mining under any circumstances. It seems to me that a law should be passed authorizing financially responsible parties to enter the reservations for the purpose of mining upon the payment, in cash, of a small annual charge for each claim located, and whenever any bullion is produced, a royalty to be paid upon the same. This royalty ought to be equivalent to the amount that the same company would pay in State and County taxes if it were located outside of the reservation.

"Of course, no company will spend the money that is necessary in order to develop a large property upon a mere permit from the Secretary of the Interior which may be revoked at any time. The law, therefore, should specifically set forth what is required, and when the conditions are complied with, there should be no doubt about the right of the company to continue its operations within the reservation.

"CARL HAYDEN."

UNITED STATES SENATE,
December 22, 1913.

"I have an idea, without looking more closely into the matter, that reservations created by Executive Order, as distinguished from reservations created by Act of Congress, do not carry title to mining lands within such reservations. The law says that the mineral lands shall be free to exploration and purchase by citizens of the United States, and I have a doubt as to the President's power to modify that law by spreading Indian reservations over these lands. I shall bring this matter up in Congress soon after the recess. In fact, I have already introduced a resolution in the Senate inquiring into this very question, and the resolution now lies on the table to be taken up by me. After submitting such remarks as I shall deem pertinent, it will be referred to the Committee on the Judiciary for an answer to that question.

"M. A. SMITH."

UNITED STATES SENATE,
WASHINGTON, D. C.,
December 29, 1913.

"I have read Mr. Anderson's letter with much interest. He correctly states that under the present law no rights can be acquired to a mining claim within an Indian reservation, and he is of opinion that it is fully within the authority of Congress to pass a law giving such rights to mineral locators as Congress may deem expedient. I feel very certain that many, if not most, of the members of both houses of Congress from the Western States, who have had personal experience, are opposed to the policy of locking up all the mining lands, and they take the position that it would be a wholesome thing if Congress should pass a bill providing for the working of mineral claims on Indian reservations.

"HENRY F. ASHURST."

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The Initiation of Title to Mineral Lands

BY ALBERT BURCH, SAN FRANCISCO, CAL.

(New York Meeting, February, 1914)

AN analysis of this subject demands a study of the theory and practice of the present system, conclusions as to its merits, and recommendations for remedying its defects if any be found.

Theory of the Present Law

The theory of the present law with reference to lode locations contemplates the existence upon the surface of mineral-bearing veins which have clearly marked boundaries, and which can be so readily traced that a locator may immediately mark the boundaries of his 600 by 1,500 ft. claim in such a way that the vein will pass through both of its end-lines. It also supposes that, having discovered such a mineral-bearing vein, the locator will at once post a notice upon such discovery, mark the boundaries of his location in such a manner that they can be readily traced, and from that time forward have absolute title to the ground embraced within such boundaries, as against every person except the United States.

For initiating title to placer ground, the law presupposes the discovery of a valuable mineral deposit; but not necessarily in place, as in the case of lode claims; and the location must be made to conform "as nearly as may be" to the legal subdivisions of the United States public land survey. When the placer law was extended to include the acquisition of title to oil lands, no exception was made as to the necessity for the discovery of valuable mineral in advance of location!

Practice Under the Present Law

Nature did not make all veins clearly defined and readily traceable upon the surface; nor did she, in all cases, leave upon the surface which overlies important ore bodies such outcrops as would in themselves be considered valuable; and the citizen of the United States has in the past endeavored to steer his course between the mandates of the law and the limitations of nature.

Until quite recently, it has been possible to obtain a patent for any-

thing called a mining claim, regardless of the character of its discovery. The Department of the Interior is now insisting, however, upon a more strict compliance with the letter of the law; and it has become almost, if not quite, necessary to demonstrate the existence of payable ore within the limits of a mining claim in order to obtain a United States patent therefor (but not, of course, in order to make a location giving the locator a possessory title against all except the United States).

Whether this change of policy on the part of the Department of the Interior has resulted in a general attempt on the part of locators to adhere more closely to the letter of the law relating to discovery, the writer is unable to say. But, in the past, thousands of lode claims have been located, upon which there were not only no outcrops of commercially valuable ore bodies, but no outcrops of any nature whatsoever. The boundaries of claims have not been so placed that the veins found within them cross their end lines at the surface, nor have their boundaries been so marked upon the surface as to be readily traceable. The result of this laxity, in conjunction with the extralateral right granted by the law, has been costly litigation, which has made the American mining law a by-word among mining investors, and has, in many cases, actually prevented the investment of both domestic and foreign capital in American mines.

Placer claims, except for acquiring title to oil lands, have ceased to be very important in the United States; and, even in the past, comparatively few have been located without some semblance of discovery.

Faults of the Present System

The chief faults of the present system, as viewed by the writer, are but two in number:

1. If the law be not strictly complied with, the result is expensive litigation and uncertain titles.
2. If the law be strictly enforced, then it is not possible to obtain satisfactory title to such areas of mining ground as are necessary for economical operation by present-day methods.

Remedies Recommended

1. The writer suggests that "discovery" as a prerequisite to the location of all classes of mining claims be abolished in law, as it has been, for many years past, in practice.
2. That in order to avoid surface conflicts, a claimant be required to have his claim surveyed immediately after location by a salaried surveyor in the employ of the government, and that the government charge such fees for this service as are estimated to be necessary to cover the salaries and office expenses of the surveyors.

3. That the number of lode claims which an individual, corporation, or association may locate or acquire in any given mining district be limited in such a manner as to prevent monopoly on the one hand and yet admit of the economical operation on a large scale which mining methods of the present day demand.

4. The writer is not sufficiently familiar with conditions governing the development of oil lands to enable him to make any recommendations regarding the initiation of title to oil-land claims, other than to say that "discovery" as a prerequisite is manifestly improper.

5. With reference to placer locations other than of oil lands, the placer law should be made applicable only to those deposits of valuable mineral which lie above the surface of bedrock, and other provisions should be made for the initiation of title to bodies of iron ore and such stratified deposits as phosphate rock, gypsum, etc., the location of which has heretofore been considered as possibly included within the provisions of the present law governing the location of placer mining claims.

6. Buried river channels of gold-bearing gravel, which are found in California, and, to a limited extent, in other States, might be located under the law governing the location of lode mining claims, instead of that relating to placers, as at present.

Naturally, such changes as are suggested above would involve some change in the laws governing the maintenance of title to mining claims, but the discussion of that subject does not come within the province of this paper.

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An Oil-Land Law

BY GEORGE OTIS SMITH, WASHINGTON, D. C.,

(New York Meeting, February, 1914)

Introduction

THAT an oil-land law is the most needed item in the proposed program of mineral-land legislation follows from the fact that Congress has never enacted a law really applicable to petroleum and natural gas. The action of Congress in 1897 in authorizing entry of oil-lands under the placer law, enacted 27 years before for the mining of gold in surface gravels, was plainly only makeshift legislation. Naturally the provisions of this placer law with its requirement of discovery as a prerequisite to location are nothing less than absurd when applied to petroleum deposits hundreds or thousands of feet beneath the surface. Other reasons which render oil-land legislation an urgent necessity arise first from the large acreage of lands believed to be valuable for their deposits of oil and gas and therefore withdrawn by Executive order from all entry pending the enactment of an appropriate law for their disposition—a withdrawal that has been specially ratified by Congress in the case of one State, Utah—and second from the exceptional importance of this resource to the nation, the large industrial worth of petroleum, and indeed its paramount value to the navy, not being suspected 17 years ago when Congress last legislated upon this subject.

Mineral land of this type is also well adapted to serve for purposes of illustration in discussing the general principles that demand recognition in legislative reform. Most of the issues involved in the consideration of oil-land legislation are clean-cut, and essential differences of opinion will be seen to be based upon radical divergence in economic theory rather than upon conflicting views concerning unimportant details.

For the last five years or more the geologists of the Federal Survey have discussed this subject and prepared memorandums and reports on the various bills introduced in Congress. Most of the provisions suggested in the present outline have been under consideration at one time or another during that period, and many of the statements which follow can be said to represent the consensus of opinion among those of us who, both in the field and in the office, have given special attention to the public oil-lands.

Purpose

Stated concisely, the purpose of an oil-land law should be to promote development, with the aim of insuring the highest percentage of extraction at the lowest cost to the consumer, and during the longest period consistent with fully meeting market demands. This is a somewhat complex ideal, in that it includes full recovery, minimum costs, and production responsive to the market; yet these are the practical ends to be desired by operator and public alike.

Except in the special case of certain limited areas, now selected from the larger reserves which were originally set apart for the Federal purpose of insuring a future oil supply for the use of the American navy, there should be no idea of reservation of this natural resource in the sense of postponing its use. We know that the present generation needs fuel oil, gasoline, kerosene, and lubricating oils. On the other hand, there should be no artificial stimulation or encouragement of speculative activity, such as will lead to overproduction and waste; for we may expect that other generations will need, and will make good use of, whatever petroleum deposits we leave undeveloped.

An oil-land law should aim to permit full regulation of development rather than to provide large revenues. Not profit but control is the primary purpose; and the beneficiary of wise legislation is to be the public in its capacity as consumer rather than as taxpayer. With low prices as the desired end, and low costs as the necessary means, caution should be exercised not to saddle upon the development any conditions or terms of land-tenure that will involve unnecessary burdens.

To meet this purpose of public control—control only as a means, not an end in itself—the leasing system is pre-eminently the logical one; and its applicability to the oil-lands remaining in public ownership is not a matter of theory; for the traveler in the California oil-fields needs only to look in any direction from the withdrawn areas to see adjacent lands to which the government has granted patent being operated under leasehold. There is no possible difference in either theory or practice between leasing public oil-lands and leasing private oil-lands, except that in the former case the public shares in the profit as landowner, and should also write into the lease conditions that are favorable to successful development and that therefore permit prices satisfactory to the consumer. In the long run, low costs can come only through successful operation; and, to be successful, oil-production, like other business ventures, needs to be attractive to the highest type of operating skill, backed by not over-timid capital. Thus fair profits to operator and just prices to consumer are far from being antagonistic elements in the proposition, but on the contrary are essential to the ideal results sought. Some of the conditions favorable to safe and economic development will now be discussed.

Conditions Promoting Development

First, any oil-land law, whether providing for disposal of the land by sale or of the deposit by lease, should contain a provision granting exclusive occupancy for purposes of exploration. Prospecting for oil involves large expenditures; and the inherent defect of the old placer law, when applied to oil-land, was this absence of protection prior to discovery. The area necessary to encourage this expensive exploration and the period essential to its accomplishment are details to be determined, with the recognition that geographic and geologic differences exist which render advisable some latitude in administering different fields, so that a maximum area is all that should be fixed by law. There should be kept in mind also the need on the one hand of enforcing purposeful endeavor and yet on the other of avoiding any requirement of non-productive "assessment work." The prospecting-permits would terminate by forfeiture because of inadequate prosecution of development work or by failure to discover oil within the statutory period, or, in the case of discovery before its expiration, the permit would ripen into a lease.

The area to be covered by the lease must necessarily be the whole or a part of that included under the preceding permit, the granting of the larger area being possibly a wise method of rewarding a discovery of the "wildcat" type. The period of the lease could be indeterminate, the life of the pool furnishing the natural limitation upon the period of operation. Requirements of drilling should be liberal enough to provide little more than what would constitute the natural response to market conditions. A more stringent requirement, however, should be that of compliance with all waste and water regulations imposed under the police power of the State.

Terms of Lease

Recognition by the lawmaker of the existence of large geologic and commercial differences in oil-fields should lead him to fix the rate of royalty only within wide limits, or leave it wholly to executive discretion. Further, as market prices and operation costs may both be expected to change, provision needs to be made for a varying royalty rate. Thus, with the declining yield of any particular pool, unless there should be a corresponding increase in price, the point would be reached when the margin between cost and receipts would disappear, even before the wells ceased to yield oil, so that further operation would require a reduction in royalty rate. This type of a sliding royalty would permit the largest possible ultimate yield.

Even though not fixed for the purpose of revenue, the royalty rates in the developed fields would approximate those fixed by private lessors,

so that the returns from public oil-land leases might be large, and could well be divided between National and State treasuries.

Prevention of Monopoly

The primary purpose of low cost to the consumer may not be attained by simply keeping down production cost, unless some measure of safeguard is provided against the possible results of monopoly. Agreements between operators either to restrict production or to raise prices may be prevented in two ways under leasehold. Transfer of lease should be conditioned upon the approval of the Secretary of the Interior, and an anti-combination proviso in the lease should carry the penalty of immediate forfeiture of the lease, allowing, however, the opportunity of a court appeal, the government to be secured by bond for the full value of the oil produced during the period covered by the litigation.

Another condition in the lease that might be thought to possess a beneficial influence on prices would be, with the royalty under an oil-lease in theory a royalty in kind, the reservation to the lessor of an option to take the oil itself rather than its equivalent at market prices. If a large acreage of public oil-land were operated under lease, the government itself might thus be one of the larger possible factors in the market and exercise a positive influence in keeping prices expressive of true conditions of production rather than of a manipulated market. At least, if the government could not thus protect the general public, it would insure its own supply for Federal use.

Résumé

An oil-land law should provide for exploration under a permit which fully protects the prospector, and for operation under a lease which favors the operator by a reasonable royalty adjustable to actual conditions, and by provisions which permit low costs but prevent the imposition of unduly high prices on the consumer.

The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada

Discussion of the paper of JAMES JOHNSTON, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 85, January, 1914, pp. 107 to 133.

CHAIRMAN E. GYBBON SPILSBURY.—I would like to call attention to the fact shown in this paper that less than five years or perhaps seven years ago nobody had thought it possible to cyanide with success silver ores. The cyaniding of silver and gold ores was first of all started in Mexico, and gradually extended to this country in the case of the silver ores alone. The average extraction, I think, for the metals in Mexico, in the case of the silver ore, is less than 85 per cent., whereas here, in the case of a very complex ore, carrying a high percentage of sulphur and arsenic, and other objectionable elements, the ore has been treated so as to extract an average of 95 per cent. of the silver contents of the ore. It is really a phenomenal revolution in the metallurgy of silver ores which has been demonstrated and worked out at the Nipissing mines.

GEORGE A. PACKARD, Boston, Mass.—I would like to suggest a slight modification, perhaps, of the statement which is made that no one considered cyaniding of silver ores up to five, or perhaps seven years ago, because in 1897, when I was at Mercur, Utah, one mill near there was operating on silver ores. It did not operate very successfully from a commercial standpoint, because they did not have a great deal of ore, but the extraction, I was told, was very satisfactory, and I am quite confident that it was, because, as suggested by the name, Chloride Point, the ores were silver chloride ores, and of course very easily soluble. Some years before that, in 1892, I made some tests myself. I spent three months making tests on more or less complex silver sulphide ores at Virginia City, Mont., and while our tests on the plain sulphide ores, finely ground, were quite successful, on account of the presence of a small amount of copper we did not work out a satisfactory solution of the problem at that time; although since then a cyanide plant has been erected and for at least eight years these ores have been treated by cyaniding at Virginia City, and the mill is still in operation. The period of five to seven years should, therefore, be modified considerably, in justice to the pioneers in the cyanide processes.

REGINALD E. HORE, Toronto, Ont., Canada.—I hoped that there would be metallurgists from Cobalt here to present and discuss Mr. Johnston's paper. In their absence a few remarks from me may help to make clearer to you what they have accomplished.

When the mining activity at Cobalt began, it was recognized that the ore was a very difficult one to deal with. The valuable mineral is chiefly native silver occurring in particles entangled in a mixture of smaltite, niccolite, calcite and other minerals. Much of the ore is very

profitably smelted, even though little or no profit is made on the cobalt, nickel, or arsenic contained in it.

Soon it became evident that a process must be devised to treat the low-grade ore. Concentrating mills were built and were successful in treating profitably much of the low grade. There was, however, difficulty in making an economical recovery of the last 5 or 6 oz. per ton. After a little experience with milling, many who were especially familiar with cyaniding processes experimented on these silver ores. At the Buffalo mill the cyaniding treatment was used. It worked fairly well. Only the slimes were treated, most of the silver being recovered by concentration. At the O'Brien and Nova Scotia mills cyaniding treatment was introduced early. In the case of some of the ores the cyaniding of silver worked fairly well, though not remarkably well. Managers of other properties, profiting by the experiences at the Buffalo, O'Brien, and Nova Scotia mills, left it pretty much alone. Straight concentration was found in many cases to be more satisfactory.

The silver in the high-grade ore is chiefly native silver. The low-grade ore has a lot of filmy sulphides as well as native silver occurring in the crevices of the fractured rocks on either side of the high-grade veins. The silver in the sulphides and antimonides was not easily saved.

There was for some time a very decided difference of opinion as to whether it was worth while to cyanide the Cobalt silver ores. The great success of the straight concentration method was evident—there were good mills working, the Coniagas and Northern Customs, for instance.

Among the most remarkable improvements in metallurgical methods at Cobalt is the amalgamation-cyanidation of the high-grade ores. This is a distinctly new method of winning silver from ores. The high-grade ore is treated with mercury in a tube mill and a wonderful recovery is made in that way.

One of the difficulties of treating the sulphides in the low-grade ores was overcome by the process devised by J. J. Denny, which is referred to in the paper as a desulphurizing process. Another important feature is the use of aluminum dust for precipitation, a process which was introduced in Cobalt by S. F. Kirkpatrick, of the School of Mining, Kingston. Since these processes have been adopted the cyaniding of low-grade ores has been more successful. In brief, it was found that some low-grade ore could be worked and milled quite profitably without cyaniding, but low-grade ores which contained considerable sulphides were costly to treat until the new process was devised. Desulphurizing by aluminum and precipitation by aluminum are important features at the Nipissing plant, which is so creditable to the metallurgists who have been connected with it and which has been so well described by Mr. Johnston in the paper now before us.

W. A. CALDECOTT, Johannesburg, So. Africa (communication to the

Secretary*).—This detailed article on Nipissing ore-treatment methods is of special interest to the writer, both on account of his association with Messrs. Butters and Johnston in their earlier metallurgical work in South Africa, and also on account of his visit under the auspices of the Institute to the Nipissing in 1907, when the treatment of such low-grade ore was an unsolved problem, and when in fact the most prominent metallurgical appliance in Cobalt camp was a cobbing hammer.

The authors must be congratulated on the practical success of their striking innovations, such as crushing in 0.25 per cent. caustic soda solution and the use of aluminum, and these departures illustrate once more the necessity of modifying known methods to correspond with the conditions of new problems.

The use of 1,500-lb. stamps with $\frac{1}{2}$ -in. mesh screening, and a ratio of ten stamps per tube mill, is in accordance with the latest South African practice, although the weight of the stamps is a mean between those installed in recent Rand mills and those in general use in the United States. No doubt the low stamp duty of 6.7 tons is due to the hard and tough nature of the ore. The water ratio, however, of 7 to 1 in the battery screen pulp and of 11 to 1 in the final slime pulp overflow seems high as compared with the 5 to 1 practicable here, and entails additional classifier capacity and pumping.

In view of the fact that the -200-mesh sand requires regrinding, it is possible that 260 mesh (0.0023 in. aperture) screening, now commercially obtainable and in use here, could be used with advantage in the grading analysis, as a sample containing, say, 2.6 per cent. +200 may show 13.1 per cent. +260.

It is probable that the crushing efficiency of the large tube mills employed would be materially increased if fed with double or treble the present tonnage of sand per 24 hr., so as to provide sufficient coarse particles to fully utilize the impact of the falling pebbles near the discharge end.

It is not clear from the paper why the high-grade ore is subjected to the treatment detailed in the paper, since the total residue, assaying 26.484 oz. per ton, is sold and presumably smelted, which treatment might be applied to the ore in the first instance.

Safeguarding the Use of Electricity in Mines

Discussion of the paper of H. H. CLARK, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 88, April, 1914, pp. 715 to 719.

GEORGE S. RICE, Pittsburg, Pa.—On behalf of the mining division of the Bureau of Mines, I want to heartily concur in what Mr. Clark has

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said, and to emphasize that in the introduction of the miner's portable electric lamps of permissible type there will be a very great gain for safety, not alone in coal mining, where there is a possibility of the ignition of fire damp, but also in metal mines, where there is a possibility of causing fires through open lights. As the matter stands at the present time, it may be some time before we can expect to see the open lamps in metal mines supplanted, but I hope it is not so far off in the case of either gaseous or non-gaseous coal mines. Safety lamps, as compared with the electric cap-lamp, are cumbersome and present elements of danger in careless hands. The proper place in coal mines for safety lamps is, as I see it, only for the testing of gas, and it may be that in the near future we will have better means for testing for gas than this.

The number of accidents that occur from open lights in coal mines is surprising. Unfortunately, the mining statistics which are gathered by the various States do not bring this out clearly, but in Great Britain in certain districts where they gather figures giving the cause of accidents from explosions and fires in open-lamp mines and safety-lamp mines, 90 per cent. of such accidents have arisen from the use of the open light.

In the matter of the introduction of electricity in mines, although it has proved of great commercial benefit, I think it must be admitted that it has increased the hazards of mining, in the case of coal mines, from the ignition of fire damp, by the trolley or grounding of power lines, and in both the coal and metal mines, as a source of ignition of fires. But electricity is so convenient we would not like to be without it, and the best thing to do is to make it as safe as we can.

I think that one of the points which will need the attention of the electrical engineers is as to the use of insulation for underground power wires. Abroad that is required. In this country it is not, and it has been constantly brought to my attention, from the reports of accidents, that if the wires had been insulated perhaps the accident would not have occurred. I have in mind one recent investigation which developed that in all probability an explosion of fire damp was caused by a wire grounding from a piece of slate having fallen on the bare power line, and carried it down so that it grounded against the rail.

There is one point I might speak of in Mr. Clark's paper, which I think, perhaps, might give an impression different from what he meant. He referred to the fact that the Bureau would take up tests of the ignition of coal dust when we had funds. He undoubtedly meant the study of different electrical means of igniting coal dust, because for the last three or four years the Bureau has been gathering samples of coal dust in coal mines throughout the country and testing them in laboratory apparatus which employs electricity for ignition, for determination of their relative inflammability. Regarding the latter we have not yet reached conclusions, but do find very marked variations in inflammability in different

coal dusts, apart from the ash contents. Sometimes coal dusts from the same district and giving about the same approximate analysis differ very markedly. The physical characteristics, the grouping of the molecules of the coal, and the relative presence of spores all may be factors. The interrelation of the chemical constituents is very complicated. The nearest parallel to the comparative inflammability I can observe is in the relative heat value of the coal, rather than in the presence of oxygen as a constituent of the coal or in the different proportions of its hydrocarbons, with the exception that Pennsylvania anthracite dust is not explosive. The study is a very useful one in following up the propagation of mine explosions. For example, in the case of a recent disaster in New Mexico, we had samples gathered at all the points where the explosion died away, and it is very interesting to see that the results in these laboratory inflammability tests compare very closely with relative violence or checking of the explosion in the mine in the respective districts where the samples were gathered. On a large scale we make somewhat similar tests at the experimental coal mine near Pittsburg, in explosion experiments. Different mixtures of dust are tried, an inert dust being mixed with the coal dust, to observe how much the inflammability is reduced.

E. GYBBON SPILSBURY, New York, N. Y.—I would like to ask Mr. Clark how far the development of the electric lamp has been carried, and also if the weight has been reduced in comparison with the weight of the ordinary safety lamp used in the coal-mining district.

H. H. CLARK.—In reply to Mr. Spilsbury's question, with reference to the development and the weight of the electric lamp, I will say that the development of the electric lamp is now in its infancy. We have been at work in collaboration, so to speak, with the manufacturers something over a year and a half. I do not know that we can say that any lamp is completely developed, although some are very nearly so, and I believe that there will shortly be on the market a practical, efficient, and safe lamp, that will meet the requirements of the authorities, that will weigh in the vicinity of $3\frac{1}{2}$ lb., and that will give a good deal more light than the safety lamp will. The light will also be disposed in a better way, so that a man can easily see the roof or the sides or the floor of the mine, or concentrate the light by being able to put his head closer to the part he wishes to see. The development is proceeding rapidly, when everything is considered.

There are a number of different manufacturers working on the problem, and all of them seem to be making headway; they will finally develop an article that will be strong, substantial and give a satisfactory amount of light, a lamp which will stand up under service.

E. GYBBON SPILSBURY.—What is the weight of the ordinary safety lamp?

H. H. CLARK.—I am not positive, I think it is about 3 or $3\frac{1}{2}$ lb.

G. S. RICE.—When the safety lamps are made of aluminum, they weigh from $3\frac{1}{2}$ to 4 lb., but the ordinary miners' safety lamp is made of brass and this weighs from 5 to $5\frac{1}{2}$ lb.

The fact might be brought out that a type of electric lamp which seems to be favored in this country is, not the hand lamp, but a cap type of lamp, with a cord going down at the back to the battery, which is carried by a waist belt, thus presenting advantages over the safety lamp carried by hand, which makes a man one-handed, or, if he hangs the lamp from his clothing or belt, there is danger of breaking the glass chimney of the safety lamp.

B. F. TILLSON, Franklin Furnace, N. J.—I would ask Mr. Clark whether the light in the lamp to which he refers is supplied by a storage battery, or is the lamp of a type which was more recently developed, in which electrodes of carbon, zinc or other metals are used in primary batteries. There has come to my attention a lamp which uses different electrodes, which electrodes require replacement each shift. That type of lamp seems to be lighter than the storage-battery lamps which we have tried.

We have tried lamps in metal mining chiefly for the use of men in the powder house and of the men who are priming the sticks of powder, so as to avoid any naked lights or lanterns under such conditions. The lamps of a storage-battery type which we have tried have been too heavy for cap lamps or workmen's lanterns. We have seen some in which they used electrodes, and changed the electrodes each shift, which were lighter. I do not know what would be the limit in the reduction of weight in future designs of a primary-cell type.

H. H. CLARK.—The lamp I had more particular reference to was a lamp of the storage-battery type. There have been attempts made to develop a primary-battery lamp, and I hope they may be successful eventually. The primary-battery lamp will probably be a little lighter than the storage-battery lamp.

The lamp I had in mind when speaking a few moments ago was the cap type of lamp which is used by most of the miners in the Western Pennsylvania field. The hand type of lamp will probably weigh more than the cap type of lamp, but will give more light than the cap type of lamp. The casing will be made heavier. Probably an electric hand lamp will weigh from 4 to 5 lb., depending on the amount of light the lamp is to give, or the length of time that the lamp is required to produce that light.

The cap type of lamp I refer to as weighing $3\frac{1}{2}$ lb. was a storage-battery equipment, although, as Mr. Tillson suggests, there are one, or possibly two, primary-battery lamps which are now in the midst of their development.

The Application of Electric Motors to Shovels

Discussion of the paper of H. W. ROGERS, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 86, February, 1914, pp. 299 to 309.

WILLIAM KELLY, Vulcan, Mich.—This is certainly a very interesting paper, for the application of electricity to steam-shovel work is perhaps one of the most difficult of its applications. The full load comes on at once, is of very brief duration, and then the operation must come to rest. The control must be positive, for if the moving parts go too far there is disaster. To control such a machine by electricity is exceedingly difficult. I would like to ask where electric shovels are in operation.

H. W. ROGERS.—There are three four-motor direct-current shovels at the Empire Limestone Co.'s quarry, Sanborn, N. Y.; one four-motor alternating-current shovel operated by the Acton Rock Co., Los Angeles, Cal.; one three-motor alternating-current shovel owned by the United States Reclamation Service, near Fallon, Nev.; one three-motor alternating-current shovel at the Granite Quarries, Ltd., Vancouver, B. C., and several smaller friction electric shovels owned by the Ontario Power Co., the Los Angeles Aqueduct Co., the Milwaukee Electric Railway Co., and the Chautauqua Traction Co.

E. GYBBON SPILSBURY, New York, N. Y.—I would like to ask whether the adoption of the motor-generator system, with heavy fly wheel, has been thought of in connection with this work. It is used in hoisting work in mines where the same conditions of stoppage and starting of heavy loads obtain. If a motor-generator set was put on the shovel, would not the intense peaks be more or less cut down?

H. W. ROGERS.—The line peaks would undoubtedly be reduced by using a fly-wheel motor-generator set with Ward-Leonard control, but calculations which I have recently made on a 120-ton shovel equipped with a four-unit fly-wheel set show that the space occupied by the boiler and water tanks (approximately 15 by 9 ft.) is not sufficient, as the motor-generator set requires approximately 25 ft. In addition to this such an equipment would be too expensive to consider.

THOMAS T. READ, New York, N. Y.—What are the chief advantages and drawbacks of the electric equipment as compared with steam-shovel work? One of the chief items of cost in shovel work in mining is lost time, and I was wondering what effect the electric equipment would have on that as compared with steam.

H. W. ROGERS.—The chief advantages of the electric shovel are lower operating cost and reduction in lost time due to fewer repairs and breakdowns. The only disadvantage is the increased initial cost, which should not be considered apart from the operating expense. Whether a shovel is steam or electrically operated should not affect the spotting

of cars and although the shovel may be idle during this process it is not using any power if it is electrically operated.

WILLIAM KELLY.—There would be a saving in power, and there would also be a saving in the number of men used for operating the shovel. Are any of these shovels operated with one man?

H. W. ROGERS.—So far as I know, all existing electric shovels require two operators, although the service of the fireman, the watchman, the coal passer, a team for one-half day, and considerable oil and waste are eliminated. In addition to this there may be a further saving in power consumption, but this will depend entirely on the comparative cost of coal and power.

F. H. ARMSTRONG, Vulcan, Mich.—I gather from your paper that you prefer the direct-current shovel. How would you advise a company equipped only with alternating current to handle this electric-shovel proposition?

H. W. ROGERS.—That question is treated on the last two pages of the paper. I recommend the use of direct-current series motors with a motor generator, as the characteristics of the series motor very nearly approach those of the steam engine. I do not claim that it is impossible to operate a shovel with alternating-current motors, but that it requires motors of from 50 to 100 per cent. larger capacity than the direct current to accomplish the same results and with an increase in power consumption.

F. H. ARMSTRONG.—I understood you to say there was not room on the shovel for a motor-generator set?

H. W. ROGERS.—My answer to Mr. Spilbury's question was in reference to Ward-Leonard control with a four-unit, fly-wheel, motor-generator set, which cannot be mounted on a standard shovel on account of insufficient space.

F. H. ARMSTRONG.—What I will have to say now really is not in discussion of the paper under consideration, because it is a somewhat different shovel, but in a sense it is an indirect discussion of it. I refer to an electric shovel which our company in Michigan is now constructing.

It has only one motor, and is a combination of an electric and a hydraulic operation. The motor drives a centrifugal pump, which pumps water to a pressure of 250 lb. into a pressure tank, above which is an air tank, to allow storage of power, and the water is carried to cylinders, one large plunger cylinder for the hoist, a smaller double-acting cylinder for the swing, with a rope from either end up to each side of the swing circle, and cylinders on the sides of the boom, double acting, for the thrust. We have only just built this shovel, and have not had it in the ore pile, and cannot say how it will work, but we have tried it out otherwise with fairly satisfactory results, as far as we can tell.

WALTER FERRIS,* So. Milwaukee, Wis. (communication to the Secretary†).—Mr. Rogers's article seems to the present writer to give a very incomplete idea of the actual present state of development of the electric excavator. In brief, after classifying electric shovels into various types he proceeds with an argument to show the advantages of direct-current machinery over alternating-current machinery without calling attention to the large preponderance of alternating-current machines now in actual service. I have not the exact figures of the number of electric excavators put out by other concerns, but it seems a proper statement to make here that during the last two and one-half years the Bucyrus Co. alone has built 17 electric shovels and drag lines, all of which have been equipped with alternating-current motors with the exception of one small revolving shovel and one medium-sized drag line. The only reason for these exceptions is that alternating current was not available where these machines were to be used and we do not consider the advantage of alternating current over direct current sufficient to justify putting in a motor-generating set to obtain alternating current.

We do, however, believe that alternating-current equipment is distinctly superior to direct, not only because of the great advantage in simplicity of mechanism, but because the characteristics of wound rotor induction motors are better adapted to shovel service than are the characteristics of series motor. The ability of a series motor to run up to high speed under small loads has scarcely any application in shovel operation. The only part of a cycle where the load is small enough to take advantage of this point is just after the dipper is dropped into the pit, when the operator starts the hoisting motor. At this point the work to be done is to accelerate the rotor, take up slack chain, and move the dipper over any space which may intervene before it reaches the material to be dug. A little observation of shovels in actual operation will show that a good operator makes it a point not to have much slack chain to take up, and to lower the dipper into the pit close to the point where digging commences.

The digging stroke usually begins with quite an acute angle between the chain and the dipper handle, so that winding in sufficient chain to make 1 ft. of bail travel will move the dipper teeth forward 2 ft. or more in the direction of digging. For this reason it is not usually necessary to accelerate the hoisting motor very quickly. A good operator, either with electric motor or engine, starts his digging stroke quite gently, and there is no opportunity for the series motor to save time by running fast under light load at this part of the cycle. After the digging part of the cycle is completed, the full dipper frequently has to be raised several feet before it is

* Non-member.

† Received May 1, 1914.

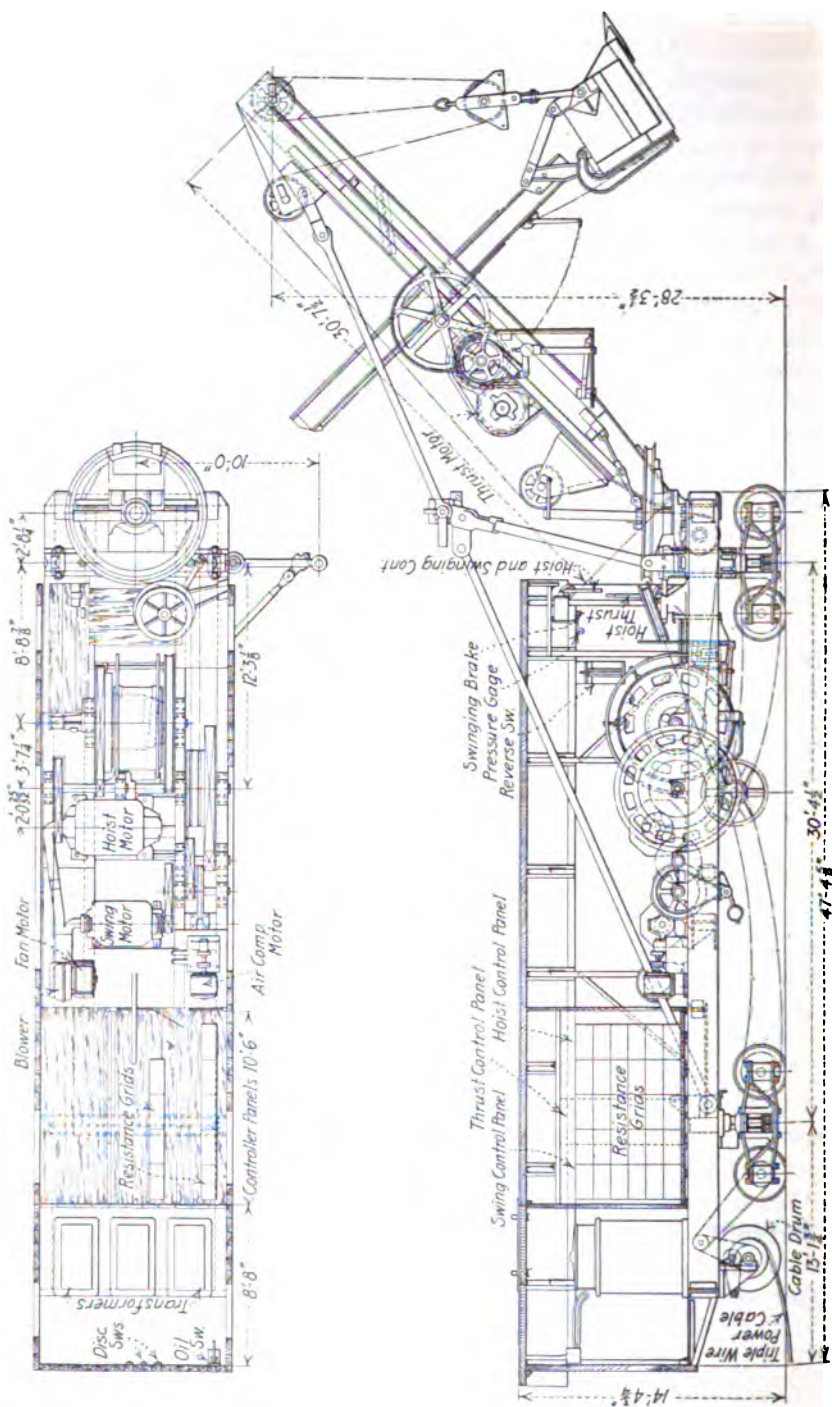


FIG. 1.—SHOVEL EQUIPPED WITH ALTERNATING-CURRENT MOTORS.

high enough to dump into the car. In nine cases out of ten there is plenty of time to raise the dipper slowly before the swinging movement has brought it into dumping position. If this were not the case the series motor would not be able to raise the dipper fast any way, as the bail pull with a loaded dipper held out at dumping radius will require approximately the rated torque of the hoisting motor.

Let us now consider the action of the series motor during the heavy digging part of a stroke. At this time the motor is usually working considerably above its rated load and the speed will therefore be pulled down. A certain amount of speed reduction under heavy load is desirable. This amount should be enough to inform the operator about how hard the machine is pulling, but any further reduction in speed tends to reduce the output unnecessarily.

The wound rotor induction motor, provided with a proper amount of resistance in the rotor circuit, will very nearly approach the performance of the steam engine. It will start promptly, accelerating to full speed, if this should be desired, in less than 1 sec. After the dipper strikes the dirt it will pull through and load the dipper with a proper reduction of speed. After the dipper is full and is pulling out of the bank the motor will promptly accelerate to a speed very slightly above full-load speed, handling the dipper with all desirable promptness.

In some places it is advisable to place some permanent resistance in the rotor circuit, but the resistance is usually automatically cut in and out by solenoid switch control. The operator has a master controller which enables him to run the motor on any desired step of resistance, while the automatic solenoids will promptly cut in additional resistance should the current exceed a proper maximum. With this arrangement the author's statement that the maximum torque cannot be obtained on starting is quite incorrect. The rotor will either start or stall at maximum torque, and automatic jamming relays prevent the current from reaching the breakdown point.

While it is advisable to keep the rotor inertia small, we consider that neither any saving of power nor any saving of time is ever a gain such as to justify using two motors where only one is necessary. As in the case of steam shovels, small power economies do not usually govern in the matter of selection of proper equipment. In most of the electric installations now being made, coal is so expensive as to leave no doubt as to the desirability of electric equipment, and the cost of electric power is quite moderate, being frequently as low as $\frac{3}{4}$ c. per kilowatt-hour. On the basis of actual results from shovels long in operation we find that the average current consumption per yard of material varies from 0.3 to 0.4 kw-hr. per yard excavated, possibly 0.75 kw-hr. for each cycle of a $2\frac{1}{2}$ -yard shovel. Taking the author's example in which he shows a saving of about 50 h.p. for 1 sec., due to small loads, I find this amount to be considerably less

than 1 per cent. of the power used by the entire shovel during a single cycle.

When trifling savings of this kind are compared with the additional complications necessary to produce them the apparent economy usually disappears. In the present case the author shows an arrangement of main hoisting machinery involving two hoist motors with spur gear reduction to two intermediate shafts, with two bevel or miter gear transmissions to a transverse pinion shaft. On this shaft is a spur pinion meshing with the hoisting-drum gear. For comparison with this we submit the arrangement of shovel machinery shown in Fig. 1, where the hoisting drum is driven by a single motor with two pairs of spur gears and one intermediate shaft as against the two motors, three pairs of spur gears, two pairs of bevel gears, and three intermediate shafts described by the author.

Pursuing the matter of simplicity, it appears that the alternating-current shovel involves three motors and three sets of slip rings as the only electrical rotating parts to be maintained. In many cases three stationary transformers are also used. As against this the author proposed for use with alternating supply a motor-generator set and four series motors with commutators. This makes a total of six revolving electrical machines, having five commutators and one set of slip rings. It is typical of shovel service that this electrical machinery will have to work in considerable grit and dust, which is particularly unhealthy for commutators. Slip rings are much less delicate and will stand a good deal of grinding down without giving trouble.

To justify such an excess of parts to be maintained we think a very strong case must be made out showing important advantages in favor of direct-current machines. The author claims a large excess in motor capacity which is required for alternating current. This is true to a certain extent, which we believe is much nearer 25 than 45 per cent. If the additional cost of maintenance, and especially the cost of delays which will undoubtedly be charged against commutator maintenance, could be figured, the writer believes that the amount would be largely in favor of the simpler alternating motors even at a higher cost.

As a matter of fact a large majority—probably at least three-fourths—of all of the large excavators (three-motor and four-motor machines) built during the past two years, have been equipped with alternating motors, and it is of record that the machines built by the Bucyrus Company have given uniform satisfaction and are all in successful operation at the present time.

H. W. ROGERS (communication to the Secretary*).—It is not the purpose of my paper to deal with the present state of development of the

* Received May 14, 1914.

electric shovel, except to show that it is a possibility, but rather to lend aid in settling the discussion as to whether direct-current series motors or alternating-current slip-ring motors should properly be used. The various types of electric shovels outlined will cover practically every one that is in operation at the present time, and the fact that several are equipped with alternating-current motors does not alter the situation.

When referring to shovels, I mean one particular type of excavator, which does not include the so-called drag line excavator, which has a very much slower cycle and for which alternating-current motors are the best adapted when alternating current is available. Mr. Ferris mentions 17 excavators which he has equipped with alternating-current motors during the past two and one-half years, and in addition to these there have probably been a dozen more put out by other concerns.

There is no doubt but that the steam engine possesses ideal characteristics for shovel service and the motor that most nearly approaches these characteristics, whether it be direct or alternating current, is the most suitable motor to use. The series direct-current motor possesses these characteristics and requires a considerably smaller capacity, to do the same work, than the alternating-current motor, and although the motor may not be called upon to come to speed rapidly during every cycle, the increased inertia, due to the larger motor, will always call for an increased power.

Mr. Ferris, in referring to the saving gained by using two 75-h.p. motors instead of one 150-h.p. motor, states that this amounts to considerably less than 1 per cent., whereas it actually amounts to $1\frac{3}{4}$ per cent. of the power required during the complete cycle.

In comparing this drive with the one submitted by Mr. Ferris, from the efficiency standpoint, the gearing and shafting from one motor only should be considered, as the motors are operating in multiple. The comparison resolves itself into one extra pair of bevel gears and one intermediate shaft for the two-motor drive, which does not look as unfavorable as the comparison drawn.

So far as the operation is concerned, the series motor will give smaller variation in speed under heavy loads, on rheostatic control, and slightly greater variations in speed under light loads, without resistance in the circuit, than the induction motor, and will come to speed in a given time with considerably less power. It can be made to run at speeds above normal full-load speed while delivering full-load torque, whereas the induction motor cannot run above synchronous speed without additional equipment, which is too complicated and expensive to be considered for shovel service. As a matter of fact, the comparison made between direct- and alternating-current equipments in the paper is based on the series motor delivering full-load torque at the high speeds; otherwise the power consumption shown on this equipment would be about 8 per cent. less,

and, taking into consideration the inertia, the power required by the alternating-current equipment would be slightly over 50 per cent. in excess of that required by the direct-current equipment. It would seem that such a saving is worthy of consideration.

While it need not necessarily be considered a disadvantage to the induction motor, it is a fact that the maximum torque is not obtainable at standstill, but inasmuch as the maximum bail pull is not based on the maximum running torque this may be disregarded. As far as simplicity is concerned, the direct-current control requires fewer contactors, is less complicated, and less expensive than the alternating-current control and the motors recommended are of the mill type, totally inclosed construction, which eliminates any commutator trouble due to grit or dust.

The motor of the motor-generator set would not be of the slip-ring type and consequently would give no trouble due to collector rings. There is no doubt whatever about the possibility of operating a shovel with alternating-current motors, but when taking all things into consideration it is more advantageous to use the direct-current equipment, even though it makes the use of a motor-generator set necessary.

Mining and Mining Methods in the Southeast Missouri Disseminated-Lead District

Discussion of the paper of H. A. GUESS, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 84, December, 1913, pp. 2749 to 2768.

WILLIAM WHALEY,* Knoxville, Tenn.—Reference is made in Mr. Guess's paper to the Myers-Whaley Co. shoveling machine, and the opinion is expressed that "it is improbable that it will ever be found profitable to use this mechanical shovel except in stopes 20 ft. or more in height, as in the lower stopes the small size of the piles of broken rock would render necessary the frequent moving of the shovel." It is also stated that it is doubtful if average per ton cost of shoveling is any lower than by hand.

I think that further data should be given, as I do not agree with either of Mr. Guess's conclusions above. The shoveling machine is very portable and moves under its own power from one place to another in a few minutes, usually between 5 and 10 min., and to get profitable operation in low mines it is only necessary to provide more places for the machine. If there are enough working places averaging, say, only 20 tons per place within a distance of 600 ft. of each other to make an output of 150 tons per shift, the operation is profitable. For instance, a 20-ton place would be loaded out in half an hour, and moving to a second place 600 ft. distant

* Non-member.

would consume 15 min., or a place and a move in 45 min. Assuming a loss of 2 hr. per day from incidental causes, car delay, track, etc., the number of places worked per day would be eight, or 160 tons. The machine is run by one man, two men shift the cars, and a fourth cleans up and helps generally; total labor cost is \$9.80, or 6.1c. per ton, which certainly should show a reasonable profit compared with 13c. per ton by hand.

The tracks should be solid and well supported and the cost per ton of laying track is more in low stopes than in higher, increasing in inverse ratio to height of stope, but that is the only item of cost which is increased with moderately low workings. This track cost increases for hand shoveling as well as for machine shoveling. At the Flat River mines, the gauge of track is 24 in. It is necessary to use outside rails for the machine at the working face, making four rails for each track for a distance back of about 30 ft. If wider gauge track were used the cost of track laying for the machine would be materially reduced, and a marked increase in efficiency would follow. In coal, iron-ore, and rock-salt mines it is usual to find track gauges of 36 to 44 in. In these mines it is easier to get a solid track, for the ties are not so high above the solid bottom as is the case in the Flat River mines.

The cost of track laying at Flat River is possibly such as to make the use of machines unprofitable to the Federal Lead Co. in stopes less than 20 ft. high, as the company pays a contract price for the material loaded by machine, which price was intended to nearly approach hand cost. The machine at Federal is operated by a contracting company which owns the machine and loads ore at a price of 10c. per ton for the machine and one operator per shift. This company is making a profit of \$250 to \$300 per month above cost of upkeep and operation, despite the fact that the cost of operation is high because the company's headquarters is several hundred miles away and it is necessary to have a high-priced man in charge of the machine, one who is competent to attend to any repairs that may be needed. It costs the contracting company \$9 per two-shift day for operators, whereas if the machine were owned by the mining company the operators would be paid about \$2.75 each, or \$5.50 per day of two shifts, the mine master mechanic then carrying the responsibility of mechanical attention.

As an example of the use of machine in comparatively low headroom, I give some data of a machine at work in the rock-salt mines at Retsof, N. Y. This seam is 8 ft. thick, and the machine has averaged a loading of 250 to 300 tons per shift (about twice the output at Flat River). The cost is about 4½c. per ton. This tonnage is only possible where a large quantity of material can be shot down, which of course is very much simpler in coal or salt than in rock. The upkeep of the machine at Retsof is 0.7c. per ton of salt loaded. The mining conditions are as follows:

Width of working places, 30 ft. Thickness of seam, minimum, 5 ft.;

maximum, 10 ft.; average, 8 ft. No impurities. Track gauge, 36 in. Roof, salt and shale. Working places dry. No timber is used. Drilling is done with rotary electric drills. Power, 250 volts, D. C. Salt is shot loose in fine and blocks. Cars hold about $3\frac{1}{2}$ tons.

The points to be noted are that, generally speaking, the machines are applicable to comparatively low mine workings, and can be profitably employed wherever sufficient tonnage can be shot loose, either in big piles or in a number of smaller piles in adjacent territory.

Recent Developments in the Design of Electric Locomotives and Coal-Cutting Machines

Discussion of the paper of SANFORD B. BELDEN, presented at the New York meeting, February, 1914, and printed in this (June, 1914,) *Bulletin*, pp. 977 to 1013.

E. W. PARKER, Washington, D. C.—I should like to ask whether or not with this new combination cutting and loading machine, you make only one cut in the face. How do you make the second cut? There seem to be three series of cutting operations—two shearing and one undercutting. Do you make a second cut, and shoot off to the side in a case of that kind, or do you run a shear from the center of the room and shoot off the side?

R. R. DUNLOP, Columbus, Ohio.—The machine makes as many cuts in the place as the width of the place will permit. In order to understand the operation of the machine, it is necessary to keep in mind the general design. The machine complete with undercutting chain, conveyors, pick and swinging conveyor, slides in a pan or stationary frame similar to the operation of a breast machine.

When driving narrow places the machine is set up in the center of the place. Consider the point on the machine where the rear conveyor is pivoted, as a point about which to swing the machine; the front end, or the cutting end, is pulled toward the left-hand rib by power. With the machine in this position, the first (or "tight") cut is made, the undercutting chain making the undercut, and the shearing chain making a narrow cut, 4 ft. apart, but perpendicular to the undercut. The shearing cut and undercut are both made at identically the same time. As the machine feeds forward into the coal, the machine runner operates the pick, knocking down the coal into the front conveyor in the cutter head (see Fig. 42), where it is carried back and dumped into the pivoted conveyor, which in turn loads it into the car.

In a 7-ft. seam this machine when fed in 6 ft. will load 7 tons of coal. If the cars are large, it will require two cars for such a cut.

After the tight cut is made, the machine is pulled back in the pan and the front end, or cutting end, is swung to the right the width of the cut, or 4 ft. When the second cut is made the left-hand shearing chain does not cut the coal, as that side has already been relieved by the first cut. The number of such cuts depends upon the width of the place. When wide places are to be cut, the machine is taken into the place and set up near the left-hand rib, and as new cuts are made, the whole machine, including the pan, is moved toward the right-hand rib the width of the cut. There is no limit to the width of the places which this machine will cut. The narrowest place which the machine can cut and provide ample room for the machine operator is 8 ft.

G. S. RICE, Pittsburg, Pa.—Can you shift sidewise?

R. R. DUNLOP.—The machine is provided with power-driven handling drum fitted with steel rope. By setting suitable jacks provided with sheave wheels at the right-hand rib, the machine can be moved to any desired position, by its own power.

The machine when taken into a working place remains there until that place is finished. When the machine is taken into the working place, the height of the shearing-chain frames is made suitable for the thickness of the vein. In any reasonable mine, the coal in any particular place does not vary in thickness to such an amount as to interfere with the operation of the machine. In mines where the coal is faulty and varies in height for short distances, the machine would not be practical, due to having to change the height of the shearing-chain frames too often.

Use of Electricity at the Penn and Republic Iron Mines, Michigan

Discussion of the paper of WILLIAM KELLY and F. H. ARMSTRONG, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 86, February, 1914, pp. 317 to 347.

K. A. PAULY, Schenectady, N. Y.—I feel that this paper is especially valuable, because it contains so completely the results of actual experience by two men who have been connected with mining ore for many years.

There are two or three questions which I would like to ask. First, under the subject of "pumping," I notice that by the substitution of electric pumps for steam pumps the cost of supplies has been reduced from a little over \$2,000 to a little over \$500. I would ask whether that reduction is due to the substitution of electric motors for the engines, or is due to the substitution of centrifugal pumps for the reciprocating pumps?

F. H. ARMSTRONG, Vulcan, Mich.—That reduction is due to the water end of the pump. The high cost of the steam supplies is largely due to

the replacement of the brass valves and brass valve seats in the steam pump, in the steam end or motor end.

K. A. PAULY.—With reference to the water rheostat, which differs materially from the standard rheostat used for mining hoist work, what resistance ranges are obtainable, and what is the minimum slip of the motor when running with the rheostat in the minimum resistance position?

F. H. ARMSTRONG.—You can get any resistance, up to so large a point that the motor will not start, by simply reducing the amount of the area of the plates which is in the water, and the slip can be varied from 20 per cent. down to 6 per cent. by the small slip plates which are illustrated at the top of the main plate. We are running at between 7 and 8 per cent. slip.

K. A. PAULY.—The efficiency, as determined from the power consumption and work done with the hoist, seemed to vary from 57 per cent., over all, to 49 per cent. I would like to ask how much of this 43 per cent. and 51 per cent. of loss is due to shaft loss and mechanical parts, and how much from the hoist coupled to the main motor, if that is known.

F. H. ARMSTRONG.—That is not known.

BENJAMIN F. TILLSON, Franklin Furnace, N. J.—Speaking of the signaling system, I notice it is spoken of as a grounded system. I would ask if any trouble has occurred through what you might call "spook signals," from inadvertent grounding in other parts of the system, or other parts of the power plant. In our alternating-current system at Franklin Furnace, we have occasionally had some "spook signals," although our system is not supposed to be a grounded system, due to the grounding of a line, which was unnoted until such signals were given. Our voltage will run somewhere around 25 or 30 volts through the bell magnets.

F. H. ARMSTRONG.—Our voltage is about the same. We have never noticed anything of the kind.

J. E. JOHNSON, JR., New York, N. Y.—I would ask how the efficiency of these electric hoists, of which we have been hearing, compares with the efficiency of the compressed-air hoist system which was applied at Anaconda. The Anaconda Co. put that job up to the electric companies of the country, and they could not come up to the specifications. Then Mr. Nordberg put in the compressed-air system, which gives an efficiency of somewhere upward of 50 per cent., and is giving the most absolute satisfaction, with a minimum of danger and a maximum of convenience. If we could get hold of some information as to the relative efficiencies of these electrical installations and the compressed-air system put in by Mr. Nordberg, of which there is given a full description in a recent issue of the *Bulletin*, it would be very illuminating.

WILLIAM KELLY, Vulcan, Mich.—That information can be had in large part now by all who are interested. In addition to the paper by B. V. Nordberg in the *Bulletin*, September, 1913, there have been other papers on the subject, one by D. B. Rushmore and K. A. Pauly, presented at a meeting of the American Institute of Electrical Engineers on Mar. 11, 1910. We expect the papers now under discussion will be followed by another paper, to be presented a little later, on the results of electric hoisting at a Lake Superior mine, and when all these papers are brought together and compared, the comparative costs by the different methods of hoisting can be arrived at quite closely.

K. A. PAULY.—I would like to ask the gentleman who mentioned the 50 per cent. efficiency for the air hoist where he obtained that figure. In the first place, I do not believe that, theoretically, such a figure is possible. In the second place, I have seen the results of the tests made at Butte, and they do not give over 29.5 per cent.; and, on top of all of that, efficiency in an air system means nothing anyway, because you have to pay for reheating the air. You have a much higher maintenance expense, both on the distributing system and the actual operating part of the equipment, the air engine.

Unquestionably the Butte system has been a complete success from an operating standpoint during the period that it has been installed, but I think that all of those who are connected with mining operations will admit that one of the most troublesome pieces of machinery about a mine is the air end of the air compressor. I think we may look for the same trouble on the engine end of the air compressor, after it has been installed some few years.

J. E. JOHNSON, JR.—The figures which were published were given by Mr. Nordberg. They include the power coming in on the line to the transmission line, and the load hoisted on the shaft, which is the over-all efficiency, if there is any such thing, and these figures are about 50 per cent.

I do not know where this gentleman gets his data of 29 per cent., but I do know that papers have been published, concerning the electrification of mines and central hoisting systems using electricity, that have described travesties on good engineering. They have spent twice as much money in the installation of these systems as a compressed-air system would have cost, and have given no better result than the compressed-air system, in some cases not as good. When the gentleman says that the air end of a good air compressor will create trouble, as compared with the electric motor, I can only say that he has had an unusual experience. I have had a good deal of experience with both, and will back the air end of the air compressor to let a man sleep a good many more nights in a year than the electric motor will.

The Injection of Cement Grout into Water-Bearing Fissures

Discussion of the paper of FRANCIS DONALDSON, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 85, January, 1914, pp. 41 to 43.

FRANK FIRMSTONE, Easton, Pa.—The European experience referred to by the speaker has been fully described in the *Bulletin de la Société de l'Industrie Minérale*.¹

There is another very obvious application of the plan of grouting a fissure that I do not think has been made. In the limestone region around Easton there are many layers of limestone which have been shattered in the course of the flexure of the rocks, which have been filled with pieces of limestone, varying from pieces the size of your head and fist to very small stones. A seam full of loose boulder stones is something an artesian well driller does not care to encounter. It is a very obvious expedient to grout such seams shut. I do not think any one has done it.

G. S. WRIGHT.—In 1908 I visited the Pas-de-Calais district in France and observed there the methods which were being used for cementation. At that time they had been pretty well worked out, so that one of the large companies, the largest company, had in sinking its shafts put aside its freezing-plant methods, and adopted the cementation method. The plan that they were then following, and I think are still following, is to take about 12 to 16 holes, put down bits about 15 or 20 ft. deep, and plug these up with concrete around the pipe in order to get an anchorage, then use a grout pump, drill down, and continue to pump in under high pressure. The first step may have been by gravity, but in 1908 they adopted a pressure of about 600 lb. per square foot. They recommend that in certain cases it may be as high as 1,000 lb. per square foot. The system worked out more cheaply than the cementation process. As many of you may know, the carbonates are very lean in the Pas-de-Calais district and adjoining the Beltin district, and the marls and chalks are very fragmentary, and filled with air, so that it is impossible in places to sink by any ordinary system, and this method is in vogue in those locations.

ALFRED C. LANE, Tufts College, Mass.—The author of this paper, in saying that cementing fissures by grouting was first tried in this country, evidently was not aware that the process was successfully used by E. F. Bradt, of the Lake Superior Mining Institute, under very great difficulties, in putting down the first salt shaft in Michigan, at Oakwood, just outside South Detroit, in 1904.² There the Devonian Dundee for-

¹ R. Fagniez 4th ser., vol. ix, p. 81 (1908). J. B. Forge: 5th ser., vol. iii, p. 625 (1913).

² E. F. Bradt: Crevices in a Salt Shaft, *Michigan Engineer*, 1907, p. 16. *Mining World*, vol. xxx, No. 12, p. 525 (Mar. 20, 1909). *Engineering and Mining Journal*, vol. xci, No. 11, p. 565 (Mar. 18, 1911).

mation which overlies the Silurian salt beds was full of crevices, regular caves, heavily charged with strong H_2S water.

Mr. Bradt has a patent on the process, which is briefly: the "displacement of the water by forcing it back into the rock or ground by injecting 'neat' cement grout under pressure," No. 849043, dated Apr. 2, 1907.

E. GYBBON SPILSBURY, New York, N. Y.—I think that a somewhat similar process to that described by Mr. Donaldson was used in the sinking of the Franz-Frederic shaft in Mechlenburg about the year 1901-02. After passing through the alluvials consisting of gravels, sands, and clays, the shaft, which was being sunk by the Kindt-Chaudron method, was footed in compact anhydrite at, I think, 1,100 ft. in depth. While continuing through the anhydrite with the regular cutting tools, a fissure was encountered with a considerable pressure of water, and the cutting tools were lost at this point. It required very nearly a year's hard work to break up the cutting ring and remove the particles. The shaft was 21 ft. in diameter. I understand that a shield, having a central opening, was lowered to the bottom of the shaft, and that grout, under hydraulic pressure, was then forced through the central pipe over the whole surface of the bottom of the shaft, so as to fill up the fissures and hold the water back. When this had been successfully accomplished new cutting tools were introduced and the grout in the bottom of the shaft was passed through with practically no leakage showing from the fissures.

Notes on the Plastic Deformation of Steel during Overstrain

Discussion of the paper of HENRY M. HOWE and ARTHUR G. LEVY, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 88, April, 1914, pp. 585 to 600.

H. C. H. CARPENTER,* London, England.—There are one or two points of perhaps general interest on which I should like to touch. I notice that Professor Howe refers to the work of Ewing and Rosenhain in connection with the plastic deformation of metals. That, of course, is quite a classical work. Their theory was that in that formation you get the metal slipping along its gliding planes in a series of steps. Now, it is very curious that that work, although quite right up to a certain point, has been comparatively little used, and, indeed, Professor Howe himself does not make any very great use of it in this paper.

One of the main reasons why the work has not been more used, I believe, is this: Professor Ewing and Dr. Rosenhain showed, in the plastic deformation of metals with which they experimented, that these undoubtedly began to slip by a series of steps, but they were not able to

* Non-member.

give a reason showing why, when once a metal has started to deform, it should ever stop deforming until it was broken.

Now, of course, we know that it does; the metal can be deformed up to a certain point with a given stress, and then, after that, the stress has to be increased if any further deformation is to take place. No explanation of that increased stress was given by the authors.

That brings me to the work of another researcher in our country, Dr. Beilby, whose work is not so well known as it deserves to be, particularly because his work has been overlooked, for the reason that he chose metals which are simpler than iron—gold, silver, and copper, which can be easily obtained in a state of purity, and, therefore, the results can be interpreted more easily. Dr. Beilby furnished an interesting interpretation of where these begin and end. Dr. Beilby showed, and I think conclusively, that in addition to the actual slip by steps, or under these gliding planes, you get an actual transformation from the crystalline solid metal into what is virtually a liquid—what are called amorphous vitreous phases. This phase is brittle, and what happens is that when the stress is applied, and the metal begins to move along the gliding planes, then the rubbing of the crystal surfaces on one another produces the vitreous brittle phases, and ultimately the crystal units which are broken up are incased in an enamel of this vitreous material, and after that rupture can only take place by breaking across these vitreous phases. All of this is very fully indicated and brought out in Dr. Beilby's paper published in 1910 or 1911. I imagine that Dr. Howe is familiar with Dr. Beilby's work, as he quoted his name in connection with the paper, but it is surprising that more use has not been made of Dr. Beilby's work in this connection.

I should like to ask the authors whether they had any difficulty in developing the structure of their pearlite and other steels by etching. The effect, of course, of the cutting work is to give us ultimately this mixture of crystalline and vitreous materials, this etching, rather differently.

Some years ago I was consulted by a firm which was having difficulty with the stamping of steel links for chains. The carbon percentage of these links was about 0.01 per cent., and I found great difficulty in developing the structure of that steel in the neighborhood of the punch. That again, I think, is explained by the theory of the amorphous vitreous phases which are formed in the neighborhood of the fracture. We will have, undoubtedly, different properties in the etched material from the crystalline material.

In this paper there is a certain amount of evidence. Turning to Figs. 3 and 4, I notice that on the extreme right we have an area which is apparently hardly etched at all. I do not know whether that is because the material is decarburized.

PROFESSOR HOWE.—It is decarburized.

H. C. H. CARPENTER.—That is a point I wanted to be clear about. I do find that difficulty with regard to etching, and I found I was only able to get comparable results by giving these severely strained links a low-temperature annealing in order to reconvert that amorphous phase into the crystalline phase. That has to be done at a temperature below the melting conversion, but I found a temperature of 500° to 550° was insufficient.

I would like to ask a question of the author. These photographs, as I gather, are of materials just in the strained conditions, without any other treatment. I would like to ask if they have had any difficulties in interpreting their stresses? I should like to express, in conclusion, my great admiration for the paper by Professor Howe and Mr. Levy.

HENRY D. HIBBARD, Plainfield, N. J.—These views showing the surface of the distorted steel remind me of some compression tests of manganese steel I made a great many years ago, the test pieces being cylinders 1 in. in diameter and 4 in. long. They were cast test pieces, ground to dimensions. After compression it was possible from the appearance of the surface, which looked much like these photographs, to tell about the coarseness of the crystallization or granulation of the metal within, the test piece having been shortened so that its diameter was increased by bulging perhaps a fourth.

WILLIAM R. WEBSTER, Philadelphia, Pa.—Several years ago, in order to show the injurious effect on steel by punching and shearing, even after the rough edges had been removed by reaming or planing, I took some pieces of steel plate and punched two lines of parallel holes in them about 3 in. apart, then cut out the longitudinal strips between these punched holes. These strips were pulled in the tension testing machine and in all cases showed hard spots opposite the punched holes, and at other points, where the metal had not been hardened by punching, it necked down. In all cases the fractures occurred at one of these points where the metal had not been hardened, but the hardening effect of the punching extended in much further than I had expected.

RALPH H. SWEETSER, Easton, Pa.—I would ask Dr. Howe if temperature has anything to do with the final rupture of the steel, if it is under stress; that is, will intense cold help the final rupture, when the shock comes, or will the higher temperature, say of superheated steam, have anything to do with the rupture of steel under stress?

C. P. LINVILLE, South Amboy, N. J.—I should like to ask Dr. Howe whether the surface hardening caused by wire drawing, cold rolling, and such processes may not to a certain extent be explained by this paper. It would seem to me that in the case of steels having some cementite, even though it be pearlitic, such surface treatment would tend to drive, or cause the softer ferrite to flow away from the surface, thus leaving a shell

much richer in cementite and accordingly harder. Another instance which I have observed is the case of rails that have been a considerable length of time in service, so that the top has been worn down. I have found a thin skin, extremely hard, which was apparently altogether cementite. On a polished section the skin would not etch when treated with acid and under the microscope was white and in its behavior like cementite. It was also noted that in this skin numerous small surface cracks were visible, which might eventually be extended deep enough into the rail to cause weakness. I realize that there is a certain amount of surface hardening, even in pure metals, which has been explained as being due to the formation of an amorphous layer, but it seems to me that in the case of the carbon steels, the driving out of the soft ferrite, leaving the hard cementite on the surface, may also have an important effect.

HENRY M. HOWE.—I am sure it is a great gratification to us to listen to the successor of Dr. John Percy and Sir William Roberts-Austen. Dr. Beilby's wonderful theory of a temporary liquid state between the natural crystalline state and the final amorphous state, such as we find in hard-drawn and cold-worked metals generally, certainly does explain an enormous amount of phenomena that before that was perfectly unknown and not understood. Why sliding should start a step, of course, was very difficult to understand. If the thing started to break, why did not it keep on breaking; if the stress was able to start, what should stop the started crack?

Dr. Beilby's theory does explain that in the most satisfactory way, so far as the phenomena of simple overstrain is concerned, but it seems to me that it breaks down at the phenomena of reversing overstrain; that is, if you strain a piece of metal past the elastic limit in tension, and reverse the strain past the elastic limit in compression, the thing seems so far to have brought about a set of results which were very different from those you got on simple overstrain.

The hardening effect of simple overstrain cannot be due to bringing the cementite to the surface before the development of the amorphous state, because that is the case in pure metals—pure copper is hardened the same way by overstrain—it hardens pure metal just the same as it hardens impure ones. The purity of the metal does not decrease the degree of hardening at all, so that it is a perfectly satisfactory explanation, but the generation of the harder amorphous metal, why that metal should be harder, is extremely reasonable and natural when you consider the natural conditions of the case.

If simple overstrain makes the metal lighter, harder, etc., the tests made so far do not indicate that reversing overstrain either hardens the metal or lightens it. Effects of simple overstrain may be removed absolutely and completely by heating to 600° C., or, as Professor Carpenter says, an even lower temperature. The effects of reversing strain do not

seem to be the same. The evidence so far as it goes does not indicate clearly that reversing overstrain is removed by such heating, but it is only fair to say that the bulk of the annealings which have been made were not carried to a high enough temperature to establish that point. That is a point which is still under debate.

The tensile strength, depending on the temperature, certainly rises with the temperature up to a moderate degree. There is a very steep rise of the tensile strength curve with rising temperature, followed by a drop again, if you get to about 200° C. There is a reversal, but from that temperature down, as the temperature falls, the tensile strength diminishes. And that may be bound up with this phenomena of the play between the cementite and the ferrite during deformation. That, however, I do not think we can feel sure about yet.

PROFESSOR HENRY LE CHATELIER, Paris, France (communication to the Secretary*).—I have read with interest the paper of Messrs. H. M. Howe and A. Levy on the plastic deformation of steel, and was particularly struck by the observations concerning the cementite lamellæ of the pearlite, which were bent without fracture. Must they not have been, then, in elastic tension? Their solubility in ferrite would be increased as a consequence. If then a steel in which the cementite is thus deformed be heated to a temperature near the point of recalescence, the curved lamellæ ought to disappear and recrystallize out; but under what form? As lamellæ or as rounded grains? A crystal soluble in water, if subjected to mechanical stress has its solubility increased. The same is necessarily true in metals. This may explain in part the facility with which metals partially hardened, and thus out of equilibrium, can dissolve and recrystallize in the presence of very small quantities of solvents.

The Work of Crushing

Discussion of the paper of ARTHUR F. TAGGART, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 85, January, 1914, pp. 143 to 161.

A. O. GATES, LaFayette, Ind. (communication to the Secretary†).—I happen to be one of those who believe that Rittinger probably meant what he said when he wrote what Stadler has quoted, "the increase of the surfaces exposed is directly proportional to the force required for reducing" "*and therefore also to the work absorbed in effecting the separations.*" (The italics are now mine instead of Rittinger's or Stadler's.)

Our difficulty lies in not understanding just what occurs at the so-called "elastic limit." Up to this point the energy applied to the body is absorbed by it uniformly in proportion to volume. At the so-called elas-

* Received Mar. 26, 1914.

† Received Mar. 3, 1914.

tic limit the first break occurs some place within the body, releasing the energy locally absorbed at the point and allowing a further deformation without a proportional amount of energy being absorbed. The cause of the first break within the body is that the ultimate strength of a single crystal, or of the bonding material between crystals, was exceeded. A series of these "local" breaks occur in sequence throughout the body until at some point several of these breaks lying close together so weaken it that the remaining crystals and bonding material are unable to resist further and the section fails, a fracture being the result. And except as energy has been absorbed in making these breaks and any heating of the material has caused radiation losses, the individual crystals of the rock are allowed upon release of the pressure to return to their original form (in conformity to Kick's law, if I understand it aright), and in so returning to this original form they return the energy they contained.

How is this energy returned? Either in pushing (or pulling) back the crushing faces, or, in accordance with a more universal law of nature, attacking the weaker part of the organization and completing the breaks upon the parts that have increased load put upon them by breaks alongside them.

I can conceive of a condition in a body in which a few individual crystals or groups of crystals are so interlaced that when the body is deformed by external pressure, and that pressure is released, these few crystals are so held that they are unable to return to their original shape. And Kick's law would apply to these, but the effect of the energy absorbed by these few crystals would have no appreciable effect upon the total of energy absorbed when actual crushing takes place.

I consider that the above reasoning applies to bodies in tension, shearing, or compression; it can be shown, I believe, that these three phenomena reduce to the same thing, the breaking of crystal or molecular bonds. I am quoting no authorities in support of my arguments, as it seems we have to be our own authority at times.

Mr. Taggart's paper is a disappointment to me in that he gives no experimental data in support of his argument. His use of Eads's experiment is not convincing; he has made an assumption that is unwarranted—namely, that all the deformation remains in the rock after fracture.

Von Reytt's experiments, which Mr. Taggart cites, were made with working crushing machines, in which there are big friction losses, in bearings mostly, the actual efficiency of which I consider to be less than 25 per cent., disregarding the energy spent upon the unmeasured fines, —0.1 mm. By efficiency I mean the ratio of energy absorbed by the rock to energy applied to the machine.

In his criticism of my arguments in *The Crushing-Surface Diagram*,¹

¹ *Engineering and Mining Journal*, vol. xcv, No. 21, p. 1040 (May 24, 1913).

he might have analyzed by Stadler's method the two crushing-surface diagrams given as Figs. 4 and 5 in my paper. (The two cuts were transposed in making up the forms, and certain box areas were not marked thereon, although referred to on the cut.) A statement in my original copy that was edited out suggested such analysis by the members of the Stadler school. Provided my work was honest and reasonably accurate, a comparison of "E. U.'s" per foot-pound applied would be illuminating.

Answering Mr. Taggart, I do not presume to say that crushing takes place in commercial machines between offset faces as shown in the theoretical diagram I used to show the application of the law of Rittinger. I might have amplified it further by showing how the cracks actually occur in the breaking rock and how the broken pieces assume their original volume *in accordance with Kick's law*. But my paper was written with the purpose of explaining a new tool for mill operators, and I was hardly prepared to take at that time the full burden of the defense of Mr. Rittinger. I did what I could, and as I look over that argument now, it looks pretty good to me.

But why "argify"? The place to prove whether either law applies to crushing operations is in the laboratory, and there upon machines which will enable us to determine the energy absorbed by the rock, not in commercial crushing machines.

Some data of this kind I have at hand. I may say that it is all favorable to the law of Rittinger, and that it will be submitted to the Institute shortly.

HALLET R. ROBBINS, Pullman, Wash. (communication to the Secretary*).—It is with some trepidation that I venture to comment upon Mr. Taggart's excellent paper, which has done a great deal to clarify a subject heretofore somewhat obscure. The arguments brought forward in support of Kick's law are logical and convincing.

Mr. Taggart seems to have gone astray on those of his formulæ which involve F , the "area of fracture with reference to unit," or else he has not made his definition sufficiently clear. To my mind, F means the total area of all the planes of fracture which are assumed to pass through the unit cube, when this is broken into a number of smaller cubes of equal size. Using the notation of the original paper, the number of these planes of fracture is represented by the expression $3\left(\frac{U}{S} - 1\right)$. The area of each plane of fracture is U^2 . The total area of fracture is then equal to $3U^2\left(\frac{U}{S} - 1\right)$ which may be written $\frac{3U^2(U - S)}{S}$. The results given by this formula may be checked by inspection and a slide rule for those cases where the unit cube is reduced in linear size one-half, one-quarter, etc. For example, to break into 64 cubes of one-quarter the linear size of the

* Received Mar. 31, 1914.

unit, it is readily seen there must be nine planes of fracture, each one $(0.896)^2$ or 0.8028 sq. in. in area; $9 \times 0.8028 = 7.225$, checking the figure in my table of constants.

Based upon the preceding formula for F, I offer the accompanying revised table of formulæ for determining Stadler's constants.

Revised Table of Formulæ for Determining Stadler's Constants

	P	S	V	F	N
P		$\frac{U^2}{S^2}$	$\frac{U^2}{V}$	$\frac{(F + 3U^2)^2}{27U^6}$	R^N
S	$\frac{U}{P^{\frac{1}{2}}}$		$V^{\frac{1}{2}}$	$\frac{3U^2}{F + 3U^2}$	$\frac{U}{R^{\frac{N}{2}}}$
V	$\frac{U^2}{P}$	S^2		$\frac{27U^6}{(F + 3U^2)^2}$	$\frac{U^2}{R^N}$
F	$3U^2(P^{\frac{1}{2}} - 1)$	$\frac{3U^2(U - S)}{S}$	$\frac{3U^2(U - V^{\frac{1}{2}})}{V^{\frac{1}{2}}}$		$3U^2(R^{\frac{N}{2}} - 1)$
N	$\frac{\log P}{\log R}$	$\frac{3(\log U - \log S)}{\log R}$	$\frac{3 \log U - \log V}{\log R}$	$\frac{3[\log(F + 3U^2) - 2 \log U - \log^2]}{\log R}$	

I have also calculated a table of constants for the sieves of the Tyler standard screen scale, based, not upon the aperture of the coarsest screen as a unit, but upon the average size of the product through the coarsest sieve on the next finer. The reason for this is that any product in a screen analysis consists of particles ranging in size between the apertures of the limiting screens. For all practical purposes it is proper to take as the average size of this product the mean of the apertures of the limiting screens. This is equivalent to assuming that the curve of crushing is a straight line between the points represented by two consecutive screens. A table calculated on the basis of the average sizes of the successive screened products should be more generally useful than one based on the apertures. For the sake of uniformity, I have followed Mr. Taggart's example in calculating the table to five significant figures, although three are all that is warranted, inasmuch as the screen apertures are accurate only to three significant figures, and in some instances even the third figure is inaccurate. The Energy Numbers in my table will not all be integers, if carried beyond three significant figures, for this reason.

For the Tyler standard screen scale sieves, $R = 2^{\frac{1}{2}} = 2.8284$, $\log R = 0.4515450$, and $\log(\log R) = 9.6547010$; $r = \frac{1}{2^{\frac{1}{2}}} = 0.35355$, $\log r = 9.5484500$.

W. J. SHARWOOD, Lead, S. D. (communication to the Secretary*):—The author of this paper has done good service in attempting to clear up, for the average mill man, some of the difficulties involved in an understanding of the rival "laws" proposed for comparing the work of crushing. It is regrettable that more exact experimental data are not available, and the hope may be expressed that such data may soon be forthcoming.

Hydrometallurgists must regret to see Rittinger's law disproved, for surface exposed is, in general, a measure of the accessibility of solutions to ore; and this rule also ceases to approximate the truth only when the finest sizes are reached. If the promises are sound the author has evidently proved the case for Stadler's Energy Units.

The fact that F is not absolutely equal to $3U^3/S$ (as assumed on p. 158), but to $3U^3/S - 3U^2$, does not apparently invalidate the proof in any way.

There appears, however, to be a flaw in the supplementary reasoning on p. 160, for if

$$W = 3 \frac{K C U^6}{V} \quad (8\text{th line from foot})$$

and
$$P = \frac{U^3}{V} \quad (\text{from (2), p. 158})$$

we would have
$$W = 3 K C U^3 \times P$$

where K , C , and U have all been assumed as having constant values.

Hence
$$W = \text{constant} \times P$$

That is to say, the work done, in crushing a given weight of ore from any one stage to the next, would, from this reasoning, be proportional to the *number of particles* at the first of these two stages. If this were actually the case, the law deduced would give, with the finer sizes, extremely large work values, even larger than Rittinger's increase of surface principle.

Apparently errors have been introduced in lines 12 and 13 from foot (p. 160), by assuming that the distance through which the force acts is $C S$; that is, that it is a constant product of the final dimension, rather than of the dimension before the breakage takes place. Again, in the next line we have: $K F C S = \text{work to rupture one particle}$, instead of *all particles*, although F has been previously defined as the fracture surface of the *total number of particles* derived from the original unit.

Crushing from one stage to another, several grades finer, may be effected in a number of ways. For instance, consider an 8-in. cube which is to be reduced to cubes of 1-in. edge. (1) It is conceivable that this might be done at one step, say by pressure applied at two opposite corners, breaking it, like Oliver Wendell Holmes's one-hoss shay, "all at once and

*Received April 20, 1914.

all o'er," into 512 1-in. cubes (Fig. 1). Or (2) by a similar application of force it might break at the first stage into eight 4-in. cubes, each of which might again be similarly broken, *per saltum*, into 2-in., and these again into 1-in. cubes. (3) A process of repeated bisection might be applied, the cube being first cleft into two 4-in. slabs, as in Fig. 2. These again

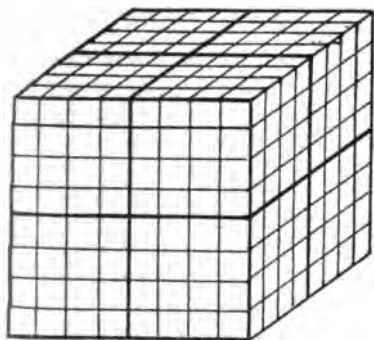


FIG. 1.

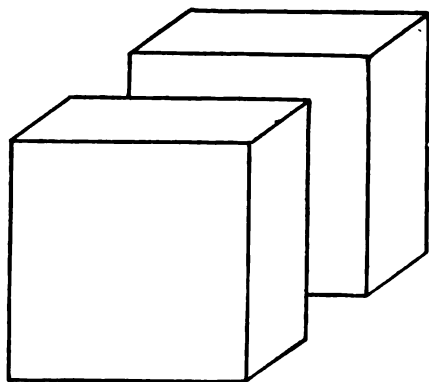


FIG. 2.

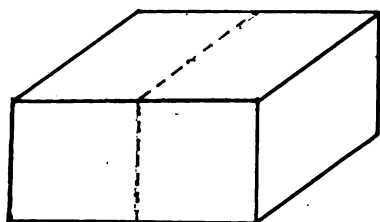


FIG. 3A.

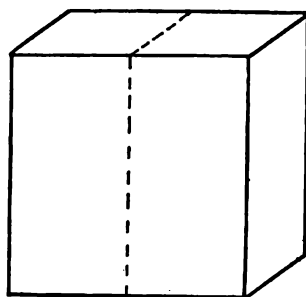


FIG. 3B.

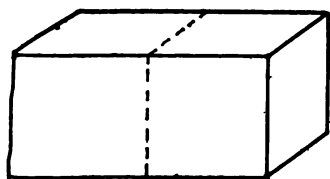


FIG. 4.

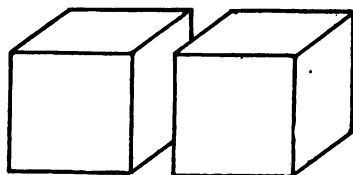


FIG. 5.

may be bisected, either by a cut across a narrow face, as in Fig. 3A, or across a square face, as in Fig. 3B, each forming two "bricks" 4 by 4 by 8 in., which are broken crosswise into 4-in. cubes. The cycle is repeated to give 2-in., and again for 1-in. cubes.

From reasoning on the lines followed by Mr. Taggart, it would seem

that the first method requires the greatest, and the repeated bisection the smallest, expenditure of energy. This indicates that, from the standpoint of work, fine crushing is best affected in several stages.

One source of difficulty in following the reasoning is due, I think, to the choice of the particular ratio of diameters adopted for the successive stages. In this connection the following quotations may be cited from Mr. Stadler's reply to the discussion on his paper, as it originally appeared in the *Bulletin of the Institution of Mining and Metallurgy*:

"Just as coordinates, in analytical geometry, are a mere matter of convenience, so the determination of the sizes and steps in grading is simply a question of convention. Once, however, we have adopted a particular standard of measurement, we must stick to it. . . . I adopted the cube for the sake of mathematical simplicity. The sphere would have been equally suitable from a mathematical standpoint, but the reduction of a sphere to smaller spheres is not easily pictured in the mind, while the reduction of a cube to smaller cubes is easily imagined."

"By reducing the cube of the unit successively to *one-half of its volume*, and assuming these fractions to be again of cubical shape, each size of this series of theoretical cubes obtained represents a grade of a reduction scale of the ratio 2."

The successive diameters adopted by Mr. Stadler have, therefore, the ratio $\frac{1}{\sqrt[3]{2}}$.

When breakage of a cube is assumed to take place in a number of stages, at each of which the ratio of successive average diameters is the same; and if, as in the series considered by Mr. Taggart and by Mr. Stadler, this ratio is an irrational number such as $\frac{1}{\sqrt{2}}$ or $\frac{1}{\sqrt[3]{2}}$, it is a physical impossibility to carry out the comminution in practice.

If, for example, we assume the ratio of diameters to be $\frac{1}{\sqrt{2}}$ or 0.707, and we attempt to carry out one stage of crushing on a single unit cube, we find that we get only one resulting cube of the theoretical dimension, and the same for each consecutive stage; the remainder of the mass is necessarily broken up into pieces of indefinite size. The same is true of the ratio $\frac{1}{\sqrt[3]{2}}$.

A clear mental picture of the process, as suggested by Mr. Stadler, is undoubtedly desirable, but many people seem to find it difficult to conjure up a mental picture of one cube of rock being subdivided exactly into two cubes. It is, however, easy to understand the subdivision when the successive linear dimensions have a ratio corresponding to some simple fraction, preferably $\frac{1}{2}$, thus giving stages of crushing which can actually be realized, as for instance by the breaking of a single unit cube into eight cubes—either at one motion or by three successive bisections, as illustrated in Figs. 1 to 5.

On account of this fact that most people, whose mathematical faculties are only moderately developed, find it easier to follow the reasoning where the reductions in size are physically possible, it would seem desirable to make the Energy Units conform to some integral ratio of diameters, preferably 2:1 rather than $\sqrt{2}$:1, and to make the intermediate values fractional.

This system has the advantage of simplicity, as it is easily remembered that each halving of the average diameter means an addition of one unit to the ordinal number. Thus in the case of Mr. Taggart's table (p. 161) the only change involved would be the division of all the numbers in the last column by two, each screen grade in the Tyler series corresponding to an ordinal number of $\frac{1}{2}$. If Hoover's $\sqrt[3]{2}$ series of screens were adopted, each successive screen would mean an addition of $\frac{1}{3}$ to the ordinal number. In a $\sqrt[4]{2}$ series the addition would be $\frac{1}{4}$, and so on.

Research with Regard to the Non-Magnetic and Magnetic Conditions of Manganese Steel

Discussion of the paper of PROF. B. HOPKINSON and SIR ROBERT HADFIELD, presented at the New York Meeting, February, 1914, and printed in *Bulletin* No. 87, March, 1914, pp. 513 to 530.

W. S. POTTER, Pittsburg, Pa.—The writer of this paper speaks of heating specimens of rolled manganese steel containing carbon 1.21 per cent., manganese 12.36 per cent., in a cementation furnace, and of obtaining various degrees of magnetic increase in the different specimens.

Under the heading "A," completely or practically non-magnetic, the writer mentions specimens Nos. 5972 and 5973, each heated 15 hr. at 530° C., while under the heading "C," very magnetic, he lists specimens 5748 and 5750, heated at 550° and 500° C., each for 6 hr. (in all four cases the metal was cooled with the furnace). A lower magnetic effect was observed in the one case after 15 hr. heating at 530° C. than in the other two specimens heated for the lesser time, one at 550° C. and the other at 500° C.

I would ask the writer what analyses, particularly as to carbon, were found in the various specimens in the lots "A," "B," and "C" after the heat treatment. In July, 1911, about two carloads of manganese-steel sheets were rolled at the works of the American Sheet & Tin Plate Co. at Vandergrift, Pa. As an experiment, one sheet was annealed in the box annealing furnace. This sheet was hard, brittle, and highly magnetic. It went into the furnace with carbon 1.17 per cent. It came out with a carbon content of 0.65 per cent.

In my paper presented to this Institute at this meeting on the subject of manganese steel, the various heat treatments as per Series 2, with the exception of heat treatments 1, 2, and 3, caused the bars to suffer a

very notable change in respect to magnetic qualities; that is to say, manganese-steel bars heat treated for 1 hr. at temperatures of from 371° C. to $1,204^{\circ}$ C. were much more magnetic than in the original rolled and water-quenched state of the steel, but this was confined to the skin of the bars, except as follows: A magnetic increase was just noticeable in the interior portions of the bars heated to 400° C. to 700° C., but not in those heated to higher temperatures. The sorbitic and martensitic conditions of the steel have perhaps a slightly greater magnetic quality than the austenite state. The strong, unmistakable magnetic effects so far as I have observed them, are, however, accompanied with loss of carbon, and a general break up, which might be expected to liberate iron from a double carbide of iron and manganese.

I repeat my question, What were the analyses of Sir Robert Hadfield's specimens after the various long heatings which he mentions?

ALBERT SAUVEUR, Cambridge, Mass.—This paper has interested me very much, but apparently I am not looking at the question from the standpoint of the authors, because while they show in their paper a very strong argument against the allotropic theory, I find it one of the best supports of the allotropic theory ever offered. In my opinion, the behavior of manganese steel under heat treatment can be accounted for on terms of the allotropic theory, and, so far as I can see, in no other way.

HENRY M. HOWE, New York, N. Y.—I think the Chairman has spoken very justly in saying that the experiments of Sir Robert Hadfield are a very strong support, not of the allotropic theory—that does not require any support, that is as firmly established as the dynamic theory of heat compared with the phlogistic theory of heat. I do not think it is open to discussion. Phlogistic theory of heat is a little longer dead. The theory that we have no allotropy in iron is equally dead with the phlogistic theory. It simply is not quite so long dead.

The case of manganese steel is greatly complicated from the fact that from it the cementite precipitates with great readiness, and that makes the phenomena much more difficult to interpret, but this evidence that you pass to the magnetic state so readily, it seems to me, is fully in accord with the idea which I expressed this morning earlier, that you have this beta state intermediate between the gamma and the alpha state. In the case of nickel steel, to which I referred, that is now to a large extent paralleled by that of manganese steel, that you could, by proper heat treatment, transfer from the gamma state to this magnetic state.

B. HOPKINSON,* Cambridge, England (communication to the Secretary†).—I am very glad that Sir Robert Hadfield has been able to present a paper to the American Institute of Mining Engineers on manganese steel and to include in it some account of the research on which we have recently been engaged together. I need hardly say that I entirely agree

* Non-member.

† Received Mar. 7, 1914.

with the statement which he has given of the general results of our joint work. I should like to add one point which has arisen since Sir Robert Hadfield left this country for the United States. He refers in his paper to the hardening or stiffening of water-toughened manganese steel which occurs when a test piece is pulled out and broken; the parts near the fracture have 2 or $2\frac{1}{2}$ times the Brinell hardness number that the metal possessed before the fracture. An accurate magnetic test which I have made recently on one such bar which Sir Robert Hadfield sent to me just before he left shows that the hardened portion near the fracture is very slightly, but quite distinctly, magnetic. The magnetism in *intense* fields is about 0.8 per cent. of that of pure iron¹, whereas a piece cut from the unstrained ends of the same bar would show less than 0.3 per cent. This fact does not, of course, affect Sir Robert Hadfield's general conclusion that profound changes in mechanical properties may be associated with very slight changes in magnetic properties and presumably therefore with but little change in the molecular condition (whether by the formation of allotropic modifications or otherwise) of the iron or other elements present.

The fact that manganese steel which has been hardened by pulling is, though only slightly magnetic, still many times as magnetic as the same steel before the work has been done on it, has some significance, however. Taken together with much other evidence which cannot be summarized here but which will be given in detail in the paper which Sir Robert Hadfield and I presented last October to the Iron and Steel Institute, to be read in May, it seems to me to show that the hardness of this material when in the magnetic condition is due to the presence of a hard constituent, which is produced by heating the water-toughened steel to any temperature between 400° and 600° C. or by mechanical work. As the amount of the constituent is increased the magnetism increases with it, but whether this is because it is itself magnetic or because the remainder is made magnetic by the abstraction of carbon or manganese into the new constituent, the evidence is as yet insufficient to decide. Stead, who proved the existence of such a constituent in 1894, was of the latter opinion. It is apparently formed as a continuous network inclosing separate grains of (presumably) still soft metal. In consequence of this distribution even small quantities of it such as are associated with trifling changes of magnetism can greatly affect the mechanical properties of the material, just as a trace of lead added to gold entirely destroys its ductility by forming a brittle eutectic inclosing the grains of pure metal. Facts such as these suggest that no invariable relation ought to be expected between the mechanical properties of a steel and the molecular condition of its constituents, because so much depends upon the arrangement of these constituents. The production by heat treatment of a hard constituent (say the beta iron of the

¹ In ordinary fields it, of course, remains practically non-magnetic.

allotropists, assuming such to exist), even in considerable quantities, might leave the material perfectly ductile if it were formed in separate nodules inclosed in a matrix of ductile metal. On the other hand, quite a small quantity of the same material arranged as a continuous network might confer hardness. It seems to me that any theory which fails to take full account of this important factor of arrangement is likely to lead to wrong conclusions.

I ought perhaps to add that the views expressed in the last paragraph must be taken as the expression of my own opinion only, as I have not yet had the opportunity of discussing this interpretation of the facts fully with Sir Robert Hadfield.

SIR ROBERT HADFIELD (communication to the Secretary*):—I note with much interest Mr. Potter's remarks regarding the decarbonization of a manganese-steel sheet. A thin sheet may be readily decarbonized, so that a reduction from 1.17 to 0.65 per cent. carbon might be expected. In our own practice in Sheffield we have had similar experience.

As regards the latter portion of Mr. Potter's remarks, he says that "manganese-steel bars heat-treated for 1 hr. at temperatures of from 371° C. to 1,204° C. were much more magnetic than in the original rolled and water-quenched state of the steel." If by this he means that upon heating to 371° C. the material becomes magnetic, then it is quite certain he would not be able to obtain magnetic qualities. The temperatures referred to in our paper must be reached before this change from non-magnetic to magnetic condition occurs. It was one of the objects of presenting this paper to identify and make clear the exact temperatures, or as near as is possible, at which this extraordinary change occurs; that is, when manganese steel passes from the ordinarily non-magnetic into the magnetic condition.

With regard to the various specimens referred to in our paper, and the question asked by Mr. Potter, I can assure him most positively that there was practically no alteration in the percentages of carbon, whether by short, medium, or long heat treatment. We have repeatedly tested and proved this. If Mr. Potter wishes, I would be very pleased to send him specimens of our manganese steel before and after the treatment referred to in the paper, so that he himself could verify the statement.

I would again state most positively that the magnetic effect produced has been obtained entirely apart from any loss of carbon in the steel. Mr. Potter speaks of "a general break up." This term seems somewhat vague, and it is difficult to answer his question without more definite information as to what he has in mind by the use of this term.

I note with interest Professor Sauveur's remarks, and in reply would say that as I have dealt with the points with regard to allotropy in my discussion on his paper entitled Notes on Some Heating and Cooling

* Received April 13, 1914.

Curves of Professor Carpenter's Electrolytic Iron, it would be better to refer the reader to this for fuller comments.

In the same way as regards Professor Howe's remarks. I do not, however, quite understand what Professor Howe means by the allotropic theory, which in the language of the founder (M. Osmond) was: "I conclude that hardened steel owes its properties principally to the presence of beta iron, which is hard and brittle by itself at the ordinary temperature." If Professor Howe includes this as one of the basic points and claims that there is a hard beta adamantine form of iron, then not only is this seriously open to discussion, but such theory has now been proved to be untenable. Surely it is the allotropic theory, if it is intended in this sense, which is dead.

I would suggest that those interested in this matter, before making any further positive statements with regard to the allotropic theory, should read the recent important papers by A. McCance on The Theory of Hardening, Professors Edwards and Carpenter on The Hardening of Metals, and Professor Honda on Transformations of Iron and Steel at High Temperatures. These three authors clearly prove that the much-disputed A2 point represents no allotropic change at all; also that beta iron does not exist.

I quite agree with Professor Howe's remarks regarding cementite being precipitated with great readiness in manganese steel.

The Heat Treatment of Steel Castings

Discussion of the paper of C. D. YOUNG, O. D. A. PEASE, and C. H. STRAND, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 86, February, 1914, pp. 227 to 233.

C. H. STRAND.—I would like to supplement the description of the photomicrographs by an interpretation of them by some figures which I have made which will perhaps be rather a departure from what has been done in the past.

Since this paper was written, there has been considerable agitation concerning the question of grain size in steel. As a result, Mr. Young, Engineer of Tests, P. R. R., has asked if it is not possible to devise a metallographic specification of necessarily wide tolerances. The results of my work in this connection have convinced me that the size and amount of the ferrite areas is the shortest and best means of judging the grain size in annealed and heat-treated steels. Ferrite has a tensile strength of 50,000 lb. per square inch; pearlite, 110,000 lb.; and sorbite, somewhat higher. Obviously, as the percentage of ferrite and size of individual ferrite areas decrease, the strength of the steel increases. However, there must be a maximum and minimum limit for this condition. In a steel with large ferrite areas fracture easily originates and

progresses. Finding the method of measuring areas too tedious, I decided simply to measure the length of the longest axis or dimension of several of the largest grains in the field, average them, and calculate the theoretical number of these in a row per inch. I have estimated the grain size of the steel from the eight photomicrographs from this paper, which are as follows:

Annealed:	Ferrite Cells per Inch
Fig. 1, No. B43.....	14,000
Fig. 2, No. B18.....	13,000
Fig. 3, No. B2	15,000
Fig. 4, No. B29.....	27,000
Average, annealed.....	17,200
Heat Treated:	
Fig. 5, No. B10.....	52,000
Fig. 6, No. B39.....	100,000 ?
Fig. 7, No. B15....	100,000
Fig. 8, No. BB.....	60,000
Average, heat treated.....	78,000

The average of annealed samples is 17,200 and of heat treated is 78,000, showing, as is to be expected, a much purer grain in the heat-treated steels. The results of B39 are not reliable as the amount of ferrite is small and indistinguishable.

In railroad work, the sections are for the most part large, few alloy steels are used, and the carbon content is fairly low, resulting in the presence of only the constituents ferrite, pearlite, sorbite, with some troostite. I believe it possible to devise some roughly accurate short and very useful metallographic specification for annealed and heat-treated steel. I would be pleased to have opinions from those present on this matter and to exchange specimens and views on methods of carrying on this work, with the object of the adoption of a standard metallographic specification.

EDWIN F. CONE,* New York, N. Y.—I want to call particular attention to the results which are given here of commercially annealed steel. I have had several years' experience in the past in producing steel castings for railroad work, especially locomotive frames, and the point to be made at the outset, it seems to me, is that everything depends, in the case of large steel castings in particular, upon the location of the test bar or the piece of steel from which the tests are to be made. In the case of a bolster, the conditions under which these are annealed differ from the test bar that might occur on a locomotive engine frame. Also, the original microstructure is totally different in each case, rendering comparisons unfair.

*Non-member.

I have never seen results in commercially annealed steel which will compare as to inferiority with the results given in the paper. Here is a tensile strength of 78,000 lb. per square inch with an elongation in 2 in. of only 6 per cent. Here is a tensile strength of 57,000 lb. per square inch, with carbon 0.31 per cent., with an elongation of only 14.8 per cent. Ordinarily, in ninety-nine cases out of a hundred, cast steel with a carbon of 0.31 will give a much higher tensile strength, and an elongation at least twice 14.8; or even two and a half times that. The 0.49 carbon is given here with a tensile strength of only 69,783 lb. per square inch, which is absolutely contrary to the results obtained from commercially annealed castings. Of course, a great deal depends on the location from which these tests were taken.

The heat-treated tests are very interesting, and especially the method of heat treatment. I have known for some time that experiments were being made, and have been very anxious to hear of the results. I feel somewhat surprised that the elongation and the reduction of area are so poor as compared with the tensile strength, and also that the elastic ratio is not higher than is shown here; that is, I mean by the elastic ratio the quotient derived by dividing the tensile strength into the elastic limit.

I want to add, particularly, that the same results and much better ones have been obtained experimentally, and also commercially, by what is called air tempering; that is, heating the castings to just about the recalescence point, and pulling them out in the air to cool. By this method one secures a higher elastic limit, as compared with the tensile strength and elongation, far superior to those shown in this table.

I would like to ask the authors from what location in the castings these tests were taken, and whether the heat-treated tests as compared with the commercially annealed tests were from the same class of steel castings, locomotive frames, or whether they were taken from other pieces.

JOHN H. HALL, New York.—Mr. Young's most interesting and valuable paper is exceedingly welcome at this time, and to me at least has a strong personal interest. Some five years ago, when I was engaged in seeking to produce a cast steel that should be capable of resisting severe and suddenly applied stresses in service, I advocated the use of a steel of about 0.25 per cent. carbon, quenched in water or oil, and reheated. My superiors in the company by which I was then employed were startled, to put it mildly, by my proposal to subject ordinary steel castings to so drastic a treatment as quenching in cold water. I was fortunate enough, however, to be in the employ of men who could be "shown," and the treatment was adopted and has been used ever since.

Last June I presented a paper on the Heat Treatment of Cast Steel before the American Society for Testing Materials, and endeavored to set forth the advantages of heat treatment, properly so called, for cast steel.

It was politely pointed out to me in discussion that though, of course, water quenching might be all very well to give fine microstructures to laboratory test bars, yet such treatment of real castings would be out of the question, on account of the danger of setting up serious stresses and cracking the castings.

My paper presented before this Institute at the last October meeting, on Shock Tests of Cast Steel, was read largely to show the advantages of heat treatment for cast steel. In the discussion of the paper, no criticism was offered of the methods of treatments suggested.

It is most gratifying, therefore, to find at this time that other shops have taken up the heat treatment of castings, and especially to read in Mr. Young's paper (p. 228) "The quenching agent employed is generally water or oil, preferably the former, because of its cheapness and drastic cooling effect, more readily breaking up the coarse cast-steel grain. With intricate castings it is generally best to use oil." Evidently others have found that the danger of cracking castings by quenching them in water has been greatly exaggerated, as I found five years ago; and that the fine microstructure and accompanying strength and toughness produced by quenching are advantages well worth securing.

Mr. Young states that 1 hr. at about 816° to 871° C. is sufficient for the first heating. I myself am inclined to consider this a little too short a heating, when the very best results are to be attained. Again, he gives temperatures for the second heating varying from 482° to 593° C., for a 0.30-carbon steel. It is not stated how long the castings were subjected to this temperature, and I should like to ask Mr. Young whether he has investigated the influence of the time of second heating upon the physical properties of the steel. My own experience has been that several hours are needed to remove the quenching stresses and draw the hardness to the desired degree.

The reheating temperatures given seem to me rather low. Personally, I have preferred to reheat the castings for several hours to a temperature closely approaching A_{r1} , some 660° to 680° C., in order to obtain the greatest possible toughness, without sacrificing strength to too great an extent. Mr. Young presents no figures for shock toughness of his steels, nor, in the plot at least, does he mention contraction of area—a most important property. I am inclined to think that with such low drawing temperatures as Mr. Young gives in his paper, a shock test would show that the quenching hardness had not been sufficiently drawn. I am not quite clear on this point, however, in that Mr. Young's paper gives but four tests of 0.30-carbon steel, with an average elongation of but 11.6 per cent., and an average contraction of area of 17.3 per cent., while in the plot he gives an average elongation for 0.30-carbon steel, heat treated, of 25 per cent., and does not state the contraction of area.

The following 17 tests (Table I) were made upon coupons attached by their long sides to castings of an average weight of about 1,500 lb. These

results were selected at random from some 2,000 tests. These castings averaged from 0.20 to 0.30 per cent. carbon, from 1.10 to 1.30 per cent. manganese, and about 0.30 per cent. silicon. Phosphorus and sulphur were about 0.05 per cent. They were heated 3 or 4 hr. at 900° C., quenched in water, and reheated for from 5 to 8 hr. to from 650° to 680° C. The very high tensile strength and elastic limit, coupled with high elongation, very high contraction of area, a nearly perfect bending test, and an excellent shock toughness, are very noticeable. In Table II, I

TABLE I

Tensile Strength	Elastic Limit	Extension, Per Cent. in 2 in.	Contraction, Per cent.	Férmont Kgm.	Bend † by 1 in. on 1-in. Mandrel
88,850	63,500	26.60	52.90	19.0	180
88,300	61,240	28.11	54.75	14.0	180
89,600	60,400	26.19	47.30	15.0	180
85,950	60,400	28.36	59.05	22.0	180
89,200	61,020	26.19	51.00	11.0	180
100,000	70,820	23.80	49.65	23.5	180
92,850	62,300	24.66	55.60	15.0	180
90,100	60,200	26.95	58.75	13.0	180
98,800	66,680	22.80	48.75	24.0	145
88,550	60,600	26.60	54.40	11.0	180
96,150	65,070	25.57	57.15	15.0	180
90,000	55,100	31.05	56.30	15.0	180
85,100	55,970	29.14	58.80	30.0	180
80,800	51,510	29.14	62.50	28.5	180
90,700	62,780	27.20	56.60	25.5	180
85,950	51,040	25.26	58.60	30.0	180
83,450	51,700	29.69	60.45	27.5	180

TABLE II

Source	Tensile Strength	Elastic Limit	Elongation in 2 in., per cent.	Contraction of area, per cent.	Férmont Kgm.	Bend † by 1 in. on 1-in. Mandrel.
J. H. H., Table I.						
Average	83,780	60,020	26.9	55.44	19.9	177.6
Lowest	80,800	51,040	22.8	47.3	11.0	145.
Highest	100,000	70,820	31.05	62.5	30.0	180.
Young, Tests						
Average	86,330	55,730	11.6	17.3		
Lowest	80,393	44,863	4.3	7.7		
Highest	92,750	61,940	17.3	27.3		
Young, Plot 0.30 C. steel						
	89,000	55,000	24.0			

have summarized Mr. Young's results and my own, in order to show that by the use of higher drawing temperatures (and a high percentage of manganese) it is possible to secure castings as strong as those of Mr. Young, but much tougher than those he shows in full. It would be most interesting, as I said before, if Mr. Young would supplement his excellent paper by giving us more detailed information on the tests upon which his plot is based.

O. D. A. PEASE.—Relative to the test specimens, as brought out earlier in the discussion, these test specimens were cut from the bolster itself, from that part of the bolster which was not strained during the time of the transverse test of the full size bolster, and were taken outside of the supporting knife edges. It is interesting to note in connection with the manufacture of steel castings, such as bolsters, that the sections are rather thin, and I might remark, in the way of getting an elongation in our tests, which is considered a little low as compared with the other properties, that it is due to the fact that the chances of getting a sound test piece out of the casting are about one out of ten.

The reason why the tensile test results as shown in the table do not check well with those shown on the plate is that the figures in the table are from light castings, or bolsters, as stated, and the plate shows the figures as taken from heavy castings. We did get some good results from heavy castings. I might mention here one which I have noted. It happens to be a part of a locomotive frame. (By the way, we have some heat-treated locomotive frames in service, and they were quenched in water.) It is about 0.30-carbon steel, double water quenched at 1,550° F., and the temperature drawn to 1,110°. This casting gave tensile results as follows: 51,000 lb. elastic limit, 90,000 lb. ultimate strength, a ratio of about 57, an elongation of 23.3 per cent., and a reduction of area of 41.1 per cent. We have duplicated such results as these on a number of occasions, and consider that they were very good.

In the matter of heating for 1 hr. for a foot thickness, I will say that this is considered by most people rather short, and I rather agree with Mr. Hall on that point. A good many of the mills take light castings, for instance, and let them soak for a period of about 4 hr. after the temperature is apparently up. For heavy castings some people allow 6 hr. The most vital thing in connection with heat-treating castings, especially those of irregular shape, weight, size, and form, is the time during which you have them in the water. This depends on the cross-sectional thickness and the irregularities of the shape. Four minutes are probably sufficient for water quenching of a casting of from 4 to 6 in., square or round, in cross-section. This gives sufficient chilling effect, and as long as the glow has absolutely disappeared, and the casting has steamed off and dried quickly when removed from the water, it is chilled and quenched. It should at that time be removed directly to the drawing

furnace, before it is allowed to become cold. This eliminates the possibility of the introduction of any quenching stresses, and will also eliminate the chances of cracking.

The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements

Discussion of the paper of J. E. JOHNSON, JR., presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 85, January, 1914, pp. 1 to 40.

RICHARD MOLDENKE, Watchung, N. J.—I am very deeply interested in Mr. Johnson's paper, as I have been working on the same subject for over 20 years. The paper is a difficult one to discuss, as there is so much in it that agrees with our experience, and again certain things that are diametrically opposed to what good foundry practice has indicated as being the case.

I must thank Mr. Johnson for proving so clearly the actual presence of oxygen in cast iron, as I have been claiming this for years as the cause of most of the foundryman's troubles on the iron end. It looks somewhat dubious to the chemist to claim the presence of even small percentages of oxygen where the carbon runs up to, say, 4 per cent., and hence steel metallurgists cannot see it. Now, however, that the analyses given have corroborated my own results along defective malleable practice, the presence of oxygen in high-carbon irons will be more readily believed. My own analyses and investigations make me believe the oxygen to be present in Fe_2O_3 dissolved or in some way present in the bulk of the metal. Mr. Johnson describes flakes of iron oxide coming out of the molten metal, to the top, and that explains a lot to the foundryman who deals only with the remelted pig iron.

I made some 14,000 analyses of low-silicon, high-carbon cast irons, pouring the samples into shot over a pail of water, picking out good pieces, about pea size, and perfectly bright and clean. When crushed and dissolved in the mixed acids for the silicon determination, no graphite being present, I noticed many years ago that whenever we had a bad run of metal—even with the proper chemical composition—there would appear on the bottom of the evaporating dish small black particles of what turned out to be the above-mentioned oxide of iron. Sometimes these periods of poor metal ran for months, and the study of the problem finally developed the fact that a radical reform in the melting of the malleable mixtures was required, so that an oxidation of the metal could not become too serious. Short heats, careful piling of the charges to insure this, and arranging the bottom to prevent feather-edged baths, accomplished the result, gave far stronger and more perfect castings than were ever obtained before; and hence oxygen was kept out by every means possible.

Mr. Johnson, however, is perfectly right in stating that oxygen gives a harder iron. The chill becomes deeper. It makes the metal set quicker, and hence from the standpoint of hard castings the presence of high percentages—comparatively speaking—of oxygen in the iron is desirable. Unfortunately, however, the presence of oxygen allows the opportunity for a reaction with carbon, forming gas, which means pinholes and gas pockets in the castings. These will be found just under the cope, or upper part of the castings, and, when in castings that can be machined, present a sorry spectacle after an eighth of an inch of the skin is removed. The raising of the freezing point of the metal, making what is called a "dead," or lifeless, iron, means that when poured the metal in the gates sets so fast that before the casting has time to be properly fed full of metal the gates are frozen, and the casting is full of shrinkage spots. Occasionally in very highly oxidized iron the metal sets so fast with the slight lowering of temperature that molds are short-poured, and ladles are skulled up badly. These are the objections to what looks like an advantage at first glance.

I further think that Mr. Johnson's tables of analyses have a weakness in the carbon determinations that leads to defective conclusions. The sample for combined carbon has been taken by drilling into the center of the test piece, thus getting the core only. The right way would have been to drill from the skin to the center and use all the drillings for the analysis. By getting the core only, the possibly heavier percentages of combined carbon near the surface, when comparing high with low oxygen samples, with identical core results, would be lost. Hence I cannot agree to his conclusions, for the stronger metal shown as due to higher oxygen content might just as readily have come from other causes, such as lower silicon with higher combined carbon nearer the skin, etc.

I would take exception further to the statement that the cupola process is not an oxidizing process, or a meaning to that effect. In the cupola we oxidize almost as much silicon and manganese as in the air furnace or the open-hearth furnace. Ordinarily there is lost in the cupola 0.25 silicon, and in the other furnaces mentioned 0.30.

Then again, what about the deoxidizers, such as titanium, vanadium, aluminum? They strengthen the metal by removal of oxygen, or the very opposite of what is shown by some of the analyses given. Hence, as stated before, I think that the man who actually uses the pig iron by remelting it, while agreeing with Mr. Johnson that oxygen hardens the metal, will have to watch his work very carefully when using high-oxygen pig irons, to avoid other casting troubles which would make the work difficult to dispose of to a consumer.

HENRY D. HIBBARD, Plainfield, N. J.—Others besides foundrymen find much in this most interesting paper, but there are several points which may need further attention. The change in the method of oc-

currence of the graphite in the iron due to the oxide of iron contained seems to amply explain the increased strength obtained by the addition of the oxide.

Referring to p. 24, it would be well to know more about the iron of faultless analysis from which good steel cannot be made, as such iron would be a novelty to the writer and perhaps to some others.

Melting cast iron in a crucible may be done with a wide range in the final temperature, say from 1,250° to 1,550° C., and therefore one should know the temperature contemplated in the statement on p. 26 relative to remelting normal iron with the spongy No. 6 in order to better understand what happens. There is possibly a more or less clearly defined temperature below which carbon dissolved in iron will not reduce iron oxide, and if the temperature of the crucible melt is below that the result may be quite different from that when the temperature is above. This, if so, may explain the possible existence of carbon and iron oxide side by side in iron. What temperature does the author have in mind?

Regarding the question of the combination of sulphur and oxygen, temperature is also important. At steel-melting temperature, or say 1,550° C., the affinity of sulphur for iron is greater than for oxygen. At lower temperatures, such as those of the puddling furnace and refinery, sulphur is oxidized and eliminated to an important degree from the iron which is worked therein.

The reference on p. 39 to chilled cast-iron car wheels and conclusion No. 7 on p. 40 need some qualification. Manganese, in pig iron, which has been reduced in the blast-furnace with the iron, seems to increase the tendency of the carbon to remain in solution when the iron freezes, as the author states. But manganese added, after it has left the melting furnace, to molten iron intended for car wheels has the effect of decreasing the depth of chill by tending to throw the carbon out of solution as graphite. This has not so far been explained, I believe, but is true, and ferro-manganese, rather finely ground, is used regularly in car-wheel foundries to cut down the depths of chill of iron whose test shows too great chilling qualities.

HENRY M. HOWE, New York, N. Y.—I do not think we will keep to our belief that oxygen is *malum in se*. Mr. Johnson does not look at the oxygen as the cause of the matter—he looks at the oxygen as reducing the size of the graphite flakes, so in spite of the harm the oxygen itself does, there is a net benefit in decreasing the size of the graphite flakes to a great extent.

C. H. STRAND, Altoona, Pa.—I would like to ask Mr. Johnson what an uneven distribution of graphite, as shown in the accompanying photomicrographs, Figs. 1 and 2, indicates as to the quality of the cast iron. These two photomicrographs were taken on spots about $\frac{1}{4}$ in. apart, the nests of graphite occurring periodically throughout. Other



FIG. 1.—UNIFORM DISTRIBUTION OF GRAPHITE. $\times 100$.

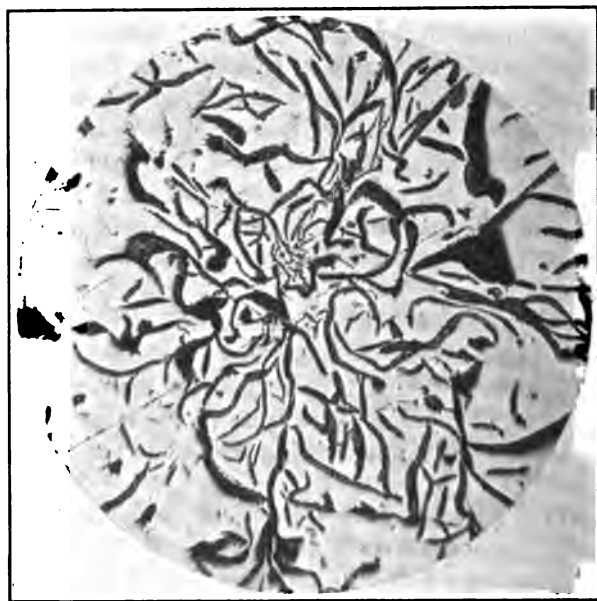


FIG. 2.—SHOWS NESTING OF GRAPHITE. $\times 100$.

cast irons show an absolutely uniform distribution of graphite. The opinion has been offered that this condition indicates a too low casting temperature. I would like to have Mr. Johnson's views on the matter, and whether he considers this condition to indicate a poor-quality iron.

WILLIAM R. WEBSTER, Philadelphia, Pa.—I will ask Mr. Johnson if he will make clear the difference between the so-called semi-steels produced from using slab in the pile of wrought iron or steel, and his steel. I believe that one is a low-carbon proposition and the other is a high-carbon proposition.

JOHN H. HALL, New York, N. Y.—On reading Mr. Johnson's most interesting paper, it at first appears that the oxygen content of the iron is wholly or chiefly responsible for the differences in strength between the good and the bad irons of similar analyses. More careful consideration, however, leads one to modify that verdict very considerably.

For convenience, the more important tests given are summarized in the two tables herewith. From Table I we see that the differences in strength between the good and bad irons of the typical six given, are more or less in proportion to the differences in oxygen content. Table II, however, shows us that differences in oxygen content between original and deoxidized irons, much larger than the differences in all but one case in Table I, not only do not cause the reductions in strength that we should expect if oxygen alone causes the variations, but even result in considerable strengthening of the iron in many cases. Deoxidation with either aluminum or titanium appears to strengthen the No. 2 irons.

The only iron in Table II that shows reduction of strength after remelting with a deoxidizer, in enough cases to lead us to believe reduction of strength to be the rule, is the iron labeled "Medium 3." Why should this iron be so weakened by the elimination of oxygen, while the others are strengthened?

At first glance one is lead to say it is because the amount of oxygen removed is the greatest in this iron. Yet comparing the decreases in strength resulting from deoxidizing this strong "Medium 3," with the differences between a weak and a strong "Medium 3" (Table I), we see that the reduction is insufficient to allow us to believe the oxygen is the only variable responsible for the weakness of the low-oxygen irons. The weakening of this iron on remelting with deoxidizers, removing some 0.044 per cent. oxygen, is of the order of magnitude of the difference between the good and bad "Soft 1" (Table I), with an oxygen difference of but 0.018 per cent.

Thus the removal of oxygen by aluminum or titanium weakens the strong irons only in the case of the "Medium 3," and does not weaken this iron as much as we should expect, even allowing for any strengthening effect of titanium. We must look further for the complete explanation of the facts observed, and to my mind we find it in the total carbon

content of the irons. All are near the probable location of the eutectic point, some 4.10 per cent. carbon according to Wittorff, and inspection of Table I shows that the greatest differences in strength correspond with the greatest variations in total carbon. This is especially the case with the "Medium 3" iron. The weak "Medium 3" contains 4.49 per cent. of carbon, and hence is well above the eutectic ratio.

TABLE I.—*Comparison of Weak and Strong Irons of Similar Analyses*

Iron	O ₂ Difference	Total C Difference	Excess Strength, Good over Bad, Pounds		
			Original	Remelted 1.25-in. Bars	2-in. Bars
Soft 1	0.018	0.06	475	525	2,500
High 2	0.018	0.11	800	850	2,050
Medium 3	0.056	0.39	1,350	1,400	8,150

TABLE II.—*Effect of Remelting Strong Irons with Deoxidizers*

Remelt No.	Iron	Deoxidiser	O ₂ Difference	Excess Strength, Pounds			
				Deoxidising Remelt over Original		Deoxidising Remelt over Plain Remelt	
				1.25-in. Bars	2-in. Bars	1.25-in. Bars	2-in. Bars
278 } 277 } 279 } 277 } 305 } 297 } 303 } 301 } 306 } 301 } 304 } 301 }	Low 2	Al	0.036	+ 50	-4,500	0	-3,700
	Low 2	Ti.	0.03	+500		+450	+3,900
	High 2	Al	0.02	+250		- 50	+1,050
	Medium 3	Al	?	-350		-650	+1,600
	Medium 3	Al	0.044	+150		-150	-1,250
	Medium 3	Ti	?	- 50		-350	-3,000

Of the irons remelted with deoxidizers, "Medium 3" contains the most total carbon, 4.10 per cent., and is weakened the most by deoxidation. The irons strengthened by deoxidation contain carbon below the eutectic ratio. The conclusion is almost forced upon us that the effect of oxygen is marked only when the iron is near the eutectic ratio. A shifting of this ratio by increased oxygen content may throw a normally hyper-eutectic iron into the hypo-eutectic class. In this connection the fact that the weak irons of the typical six contain in every case more

total carbon than the strong irons, is suggestive. Whether this shifting of the iron to the hypo-eutectic class is capable of producing the fine graphite formation shown, I am not prepared to state. I should suppose, however, that it would be.

The improvement in the strength of weak irons produced by remelting them with an admixture of "spongy No. 6" may also be explained in this manner. Mr. Johnson, judging by his remarks in paragraph 3, p. 14, seems to have seen that the influence of oxygen and that of high total carbon are jointly responsible for the facts he observed, but to have laid the heaviest emphasis on the influence of the oxygen.

It would be of great interest if Mr. Johnson would give us more data, as, for instance, the microstructures and more complete analyses of the irons remelted with deoxidizers, complete analyses, tests, and microstructures to show the effect of remelting "weak spotted irons" with "spongy No. 6," complete analyses, tests, and microstructures to show the status of the cold-blast charcoal irons referred to, analyses and tests of the electric-furnace irons, etc. The subject matter of the paper is so important and the opportunity for contributing greatly to existing knowledge so clear, that more ample data will inevitably be of the utmost value. As the paper now stands one is constantly headed off in following a line of reasoning by the dilemma which confronted Hosea Bigelow's campaign orator: "Tries pump again—no facts!"

ALBERT SAUVEUR, Cambridge, Mass.—Mr. Johnson has again placed those interested in cast iron and its properties under very great obligation by his well-planned and carefully conducted investigation of the puzzling fact that some irons—the good ones, as he terms them—always give strong castings, even after repeated remelting, while others, of *apparently* the same chemical composition—the poor irons—always yield weak castings. Mr. Johnson claims that the presence of a larger proportion of oxygen in the good irons is responsible for this great strength and he contends that his experiments prove his claims beyond any possible doubt. While at first sight it may appear that he has demonstrated his contention, upon closer scrutiny and thoughtful reflection doubt arises as to his having discovered the real cause of the phenomenon under investigation. The author shows a difference of a few hundredths of 1 per cent. of oxygen between the weakest and the strongest irons, otherwise of similar composition, so far as analysis of cast iron generally goes, and we are asked to believe that so very slight a difference in oxygen content accounts satisfactorily for the much greater strength of the good irons. He contends, for instance, that increasing the percentage of oxygen from some 0.02 to 0.1 per cent. is responsible for an increase in transverse strength from 3,000 to 5,000 lb. Turning to the photomicrographs illustrating the structure of the various irons, we see clearly why the good irons are stronger; it is evidently because their graphite occurs in

small, and often rounded, particles, instead of occurring, as it does in the bad irons, in the form of long, curved and sharp plates of sheets. It is for a similar reason that malleable cast iron is always stronger than gray iron of like chemical composition. We are naturally reluctant to believe, however, that so marked a difference in the mechanical behavior of graphite is caused by a difference of a few hundredths of 1 per cent. of oxygen between the two classes of cast iron. To accept this explanation, we certainly must have more conclusive evidences than those offered by the author.

Granting that Mr. Johnson has satisfactorily shown that good irons contain more oxygen than poor irons, and that the furnace conditions necessary to produce strong irons are generally, also, those which lead to less perfect deoxidation, the inference that the greater strength of the good irons is actually due to the presence of a slightly greater percentage of oxygen is very far from conclusively demonstrated. Were it so, it should be a comparatively easy matter to increase the strength of poor irons by oxidizing them, and to decrease the strength of the good irons by deoxidizing them, and it is a significant fact that the author's attempts in both of these directions have failed.

The results published in Tables I and II are not convincing, or even always consistent. The average strength of the C High 2 iron, for instance, containing 0.038 per cent. oxygen, is 3,800 lb., whereas the average strength of the E Med. 3 iron, with 0.065 per cent. oxygen, is but slightly greater, namely, 3,950 lb. In Table II, the irons low in oxygen appear to be as strong as, if not stronger than, those containing more oxygen.

Summing up, the utmost that may be granted is that the author has shown satisfactorily that good irons generally contain more oxygen than poor irons. He has not demonstrated that poor irons can be made stronger through the introduction of oxygen, or that good irons are made weaker through the removal of some of their oxygen. The fact that normal iron (containing little oxygen) is made stronger by melting it with "spongy No. 6" (containing more oxygen) does not prove that oxygen is actually the cause of the increased strength. It merely points to some qualities, inherent to the spongy No. 6 and making for strength, being imparted to the resulting mixture.

BRADLEY STOUGHTON, New York, N. Y.—I really think that Mr. Johnson's analyses show definitely more oxygen in some cases than in others, and this offers much more evidence than the absence or deficiency of oxygen by inference. Of course, I do not know but that Dr. Moldenke has made these analyses, too, but I was simply taking what he said to be limited by as much as he said.

In regard to Mr. Hibbard's remark about faultless iron, as far as the analysis is concerned, not making good steel, I have had that experience

a number of times. There is a sort of iron which comes from the blast-furnace out of which you cannot make good steel. It has been my misfortune to note that many times. We have never been able to find out, in the analyses, anything that is wrong with that steel, but I have seen the percentage of second-class rails go up from 3 to 22, and it was an unhappy time for me, because I was supposed to be responsible in any event.

It finally got so bad in a large works with which I was connected that a practical man there could tell this iron as it ran from the furnace—it had a peculiar appearance—and in order to prove that it was the iron which was not satisfactory he would take a ladle of iron every once in a while and send it directly to the converter to convert it into steel, without its passing through the mixer, the analysis, of course, being correct, so that we were able to do that. In every case we had very bad results with the steel. Furthermore, I had to convince the men having charge of this steel that if a certain number of ladles of this kind of iron went into the mixture we would have unsatisfactory results, and in each case we would find the percentage of second-quality rails went up.

The analysis of that iron was all right, but it had a peculiar appearance to it, which none of us could tell except the one man to whom I have referred, and finally we used to keep him at the blast-furnace a greater part of his time trying to spot that iron and prevent it from going into the mixer and being made into steel, much to the disgust of the blast-furnace man, who was positive, of course, that it was the fault of the steel mill.

It seems to be admitted by all of us, from what I have heard said, and it seems very evident to me, that these irons high in oxygen, but otherwise practically the same, will have a better structure than the irons which do not have oxygen in them. It seems to me, even if the oxygen in itself is harmful, provided the net result is good, there should be no objection to the oxygen. Certainly we agree that the iron with graphite in large flakes is likely to be, as Mr. Johnson certainly indicates, very strong, and is stronger than the iron in which the graphite flakes break up easily.

The fact that the amount of oxygen is very small does not seem to me to be material at all. We do not know what form the oxygen is in. One-thousandth of 1 per cent. of oxygen may mean quite a good deal of iron oxides or some other oxides, and we have always had that augmented with a small amount of phosphorus. If you remember, the small amount of moisture in the blast-furnace was judged to be ridiculously small to produce any beneficial effect on the work of the furnace. Consequently our German friend patented the idea and decided that the Americans did not know what they were talking about in blast-furnace practice until they were obliged to admit that their theory was defective in some way.

It has occurred to me, and I find in conversation with others it has occurred to several persons, that possibly the oxygen may have an

influence in throwing the eutectic point to the right in the diagram, and several have mentioned that particular thing to me. I do not see any evidence to confirm that view, of course, except by inference, and if this occurred it would have an effect, perhaps, on the oxygen, and might have a marked effect on the quality of the iron.

WILLIAM CAMPBELL, New York, N.Y.—From looking over Mr. Johnson's figures and photographs we must be led to the conclusion that the difference in strength depends mainly on the forms of graphite. In examining quite a number of samples of cast iron, we find that there is certainly a difference in the way in which the graphite comes out. Some of it comes out directly from the liquid, whereas other forms of it appear after the metal is solidified. Looking over the different iron-carbon diagrams, of which there are many, we can get some evidence of this; if the later determinations of Wittorff and similar workers are right we have the formation of cementite just above the freezing point, which afterward breaks down into the graphite. The natural conclusion is that the graphite which forms from decomposition of cementite will be finer than that which comes out of the melt. It may be that the small amount of oxygen present in the iron may be the factor which swings over from the stable diagram on the one hand to the metastable on the other, and in one case we get mainly the primary graphite and in the other case we get the secondary graphite.

RALPH H. SWEETSER, Easton, Pa.—Possibly some experience I had in Canada may throw some light on this "good iron that makes bad steel." I happened to be at the blast-furnace end, and my position was not any more cheerful than that of Mr. Stoughton. At the Lake Superior Corporation we made iron for the Bessemer department, and it was made wholly with Pocahontas coke. There were times when we had the very best of iron, it would be what we would call in the foundry No. 2, or 2X iron, but the graphite was very high, and the kish that would come off from the iron would collect everywhere, yet they claimed at the steel plant that the iron did not make good steel. The analysis showed just exactly what they asked for in silicon, sulphur, and manganese, but they had trouble in the mill, and they blamed it, of course, on the blast-furnace. We found large flakes of this kish, that were almost transparent, and when they were analyzed we found that they were a combination of silicon and carbon. I think there was just exactly one atom of silicon and one atom of carbon in some of the flakes we separated.

Possibly the making of iron with a very soft coke makes what we call a "rich iron."—the steel men said "Your iron is too rich." The air would be full of kish, and when that iron was cooled down in the ladles the top of the ladle would be covered with a thick layer of kish.

HENRY D. HIBBARD.—I am not satisfied to let the matter rest so. Mr. Stoughton's experience with those irons was when he was much

younger than he is now. It is noticeable that his trouble with the apparently good iron of which good steel could not be made was in connection with the Bessemer process, which of course does not have as wide a scope in the materials used nor admit of variations in anything like the degree that the open-hearth process does.

Another thing is that in plants where only one product, like rail steel, is made they do not have much variety of methods or experience with other steels. Then when the regular practice with the regular stock does not yield the required result they are more or less at sea and may blame the stock when some change in manipulation might have made all right. Therefore, while I do not consider that Mr. Stoughton's experience proves the author's statement, I think it would be well if he would give us some analyses of those irons and say what he was making of them.

F. L. GRAMMER, Leesburg, Va.—I think Mr. Johnson has correctly attributed to temperature the position of chief factor in determining whether the hearth has the maximum reducing tendency and I also am inclined to believe that the chief source of oxygen is to be found in the descending ore. Nevertheless, there are other influences than temperature and there are other possible sources of oxygen than the ore. Perhaps the following may be of interest, even though the writer believes the first two influences are paramount.

Under Carbon Absorption, p. 29, I would like to add two more known facts:

3. The larger the amount of surface offered by a unit weight of carbon the shorter the time required for its solution in a given weight of molten iron. This is a relation governing all solution if interstices are not too minute for liquid to enter.

4. Solution as it approaches saturation proceeds more slowly.

While in discussion of oxygen contents we are not so much concerned with the first two postulates of Mr. Johnson as with the last two, these added factors if considered will be seen to be the product of conditions assisting deoxidation.

Let us assume arbitrarily for conciseness that the metal as it drops into the bath of iron through the slag contains 2 per cent. carbon in solution and that it issues from the furnace with 4 per cent. carbon content. The 2 per cent. increase would be obtained by the bath dissolving carbon from incandescent fuel which penetrates below the floating molten slag into the molten iron. At casting times the solid fuel goes to the hearth bottom. That it does this after the first flush of cinder (as a rule) I am uncertain.

We are told that recently some of the large steel companies are giving their blast-furnaces crushed coke, rather than as it formerly came from beehive ovens. The motive probably is better distribution and regular

working. Such coke, offering more surface per unit of weight, should produce a metal with lower oxygen content, other conditions being constant, than one with larger lumps. The intermixture in the bath is more complete.

The Scandinavian mystic and metallurgist Swedenborg, I am told, first announced that spheres of equi-diameter tetrahedrally piled exhibited a ratio of voids of 29 per cent. and aggregated 71 per cent. So the amount of space in the hearth occupied by the fuel (if of equal spherical size) is practically independent of the size of the lumps. The amount of surface offered for attack by the molten material (occupying the voids) increases as the diameter decreases.

The incandescent fuel may not penetrate the slag and iron bath because of the pressure of the superincumbent stack contents; usually we believe this pressure on both is due chiefly to the weight of a dome-capped cylinder of incandescent fuel extending from a hemisphere, at the flare of the bosh near the tuyères, downward. The weight of material in the stack is borne by the bosh (on which stack momentarily arches) and by friction and ascending gases.

This dome-capped cylinder of fuel is taller after casting than before casting.

Taking the bath of iron and slag at cinder-notch level on the first flush, it might be fair as furnaces are built to say that the maximum fuel heights of dome-cylinder would be in coke furnaces using Lake ores about 9 to 13 ft. and in charcoal furnaces about 6 to 10 ft. Disregarding penetration below the molten surface, we may say for anthracite, 10 ft.

We will call the specific gravity of anthracite 1.6; coke, 1.0; and charcoal, 0.4.

Then this dome-cylinder will give a fuel pressure on top of the molten surface as follows:

Anthracite.....	$10 \times 1.6 = 16$
Coke.....	$(9 \text{ to } 13) \times 1.0 = 9 \text{ to } 13$
Charcoal.....	$6 \text{ to } 10 \times 0.4 = 2.4 \text{ to } 4$

If this is the pressure which controls penetration into metal, it will be seen that charcoal has nothing like so long a contact nor so deep a penetration into the metal bath as coke has, not to mention anthracite. Charcoal just before casting or before the first flush may hardly penetrate the slag. Oxygen in the bath thus is less likely to be removed.

One may protest that we are not dealing with spheres nor even with sized particles, and that our 2 per cent. carbon is arbitrary. Many will question this method of stock settling. They will say it is due to the underpinning by solution and combustion of a fuel column resting on the hearth bottom and not by constant renewal of a fuel arch springing from the bosh batter commencing above the tuyères.

Our picture is probably more closely followed by narrow crucibles using charcoal than by coke crucibles of greater width and weaker dome. The truth is somewhere between. We have all seen coke at the bottom iron notch: it should always be in front of the tuyères. But we have heard of metal being partly Bessemerized in the blast-furnace crucible owing to fuel descent being delayed. It is certain that a very large percentage of the weight is borne by the bosh and the stock.

We think most would concede that charcoal would dissolve in iron more rapidly than coke or anthracite, not to mention graphite. It is true that in recarburizing steel the solution of added coke is very rapid, but at blast-furnace temperatures the surface of fuel exposed and the time of contact rather than the nature of occurrence govern contents. The final carbon increments are slow of growth—certainly if these oxide products are with such difficulty removed; time and full surface exposure where greatest in crucible should be associated with lowest oxygen.

This picture is timidly sketched in the hope that some may see more than a cubist scramble, exhibiting additional possible explanations besides these offered by Mr. Johnson.

CO_2 reacts on C at certain temperatures depending on preponderance of fuel and its nature and velocity of passage of gases. We would expect this temperature to be lower with a soft, cellular fuel like charcoal, which occludes, than with a dense, hard fuel (resembling incombustible graphite) like anthracite. The zones of combustion per unit of blast will be larger in the case of hard and dense fuels than with porous fuels. This seems to be an additional reason other than slag composition why coke and anthracite furnaces need high temperatures of blast compared to charcoal practice.

Magnetites enter the hottest zone with more oxygen than other iron ores save a few specular ores and so it seems reasonable to expect more oxygen in metal bath from such mixtures.

Owing to low ash in charcoal, in such a furnace the slag is practically all complete as it drips into the crucible. In a coke furnace, a large percentage of the slag comes from ash which is released as combustion in the hearth proceeds. It unites with the excessively basic product descending from the bosh.

A coke furnace requires reserve heat to fuse a belated slag completion, and to take care of a slag whose volume is excessive or basicity and temperature are high, in order to prevent sulphur entering the metal.

A unit weight of oxygen in entering a hot-blast furnace occupies thrice the volume of a similar weight furnished the charcoal furnace. Besides the increased *velocity* of gases incident to excess heat in coke furnaces, the velocity is also larger because of the larger weights of air (rapid driving) employed.

The cold-blast charcoal furnaces offer converse conditions to those

cited as obtaining in coke practice. Whether the oxygen in metal has its origin in descending charge or entering blast, the two furnace operations are so diverse as to lead one to expect differing products.

Thinking of occlusion, one would have expected the higher-pressure coke furnaces to show higher gas contents in metal.

Throttling the downtake increases the gas tension in the furnace very markedly, preserving at the same time quantities and velocities of blast. It might be interesting to note whether metal made under such conditions exhibits change as regards gaseous contents. One should expect by this throttling, if fine ores are used, somewhat increased hanging. Probably carbon deposition is accentuated under tension.

Antimony, unlike phosphorus, with which it is often grouped chemically, seems to increase total carbon contents in pig.

I analyzed some kish scooped up from the ground in 1894 at the Edgar Thomson Furnaces and found it contained a large per cent. of silicon, as Mr. Sweetser cites.

We had a case of "dead iron" at Duquesne in 1897 in which a lowering of the temperature of the blast accompanied by the reduction of the burden was followed several hours later by a reduction in the per cent. of seconds of the rails.

There is quite a widespread belief that iron made from magnetites is better than that made from hematites, and we rather expect those who have had the misfortune to use these irreducible ores to investigate the possibilities of the superior value of their product as compared with hematite metal.

In addition to investigations pursued to ascertain the influence on oxygen content of gas throttling, magnetites, crushed coke, and moisture elimination, we also think it would be of interest to ascertain whether metal made from furnaces with 20 tuyères as compared to furnaces with 10 tuyères has a different oxygen content.

J. E. JOHNSON, JR.—Replying in part to Dr. Moldenke's criticism, I would call particular attention to an analysis on p. 3 of my paper in which it is shown that the variation of combined and graphitic carbon in these irons is exactly the opposite of what it should be if the accepted theory were correct; that the strength increases primarily with the quantity of combined carbon and decreases with the quantity of graphite. This latter is a perfectly logical and obvious conclusion if we disregard variations in the size and shape of the graphite particles. I have therefore taken some pains to analyze these results so as to show that the strength varied in these six irons in the opposite direction from what these considerations would indicate, and this seems to me to disprove absolutely any contention based on the idea that the variations in the combined carbon are responsible for these results. This absolute contradiction of this obvious and natural conclusion indicates that there must be some cause for the

results totally independent of this consideration. This cause, I believe, will be found by any one who examines the photomicrographs critically, in the size and shape of the graphite particles, for there is no important cohesion between graphite and iron, and each flake of graphite is therefore neither more nor less than a cut in the iron of area equal to that of the graphite plate. To carry this idea to its ultimate conclusion we need only to turn to the salamanders which all furnacemen have taken from the hearths of their furnaces and which contain flakes of graphite a square inch or more in area and can be broken apart along these flakes with the fingers. What a flake of this size would do to a $1\frac{1}{2}$ -in. test bar is obvious and is an index of the importance of the size of the graphite. I have tried to make it clear in the paper that it was through this control over the size of the graphite particles that oxygen exerted its principal influence.

In regard to Dr. Moldenke's statement that he had made many hundreds of oxygen determinations, I understand that these were made by a wet or acid method, and I do not consider these methods reliable for the determination of oxygen, since almost every action of solution is an oxidizing one in one form or another, and for that reason oxygen may be introduced into the very substance whose oxygen content is sought to be determined.

The determinations on which my paper is based were made by the Ledebur hydrogen method as modified by Cushman and by ourselves to suit the conditions arising from the presence of so much more carbon than is present in steel. So great an amount of care and time was put upon these determinations, such satisfactory blanks could be run with them and results could be so carefully checked, that it is impossible to escape the conclusion that the oxygen which we found in the form of water came from the iron and from the iron alone.

I would like to call attention here to the fact that much of what Dr. Moldenke himself has written on this subject can be as well explained on my hypothesis as on his, for he has repeatedly spoken of getting the bath too hot and burning it. What really happens, in my judgment, is that when the bath becomes too hot the oxygen reacts with the carbon and silicon present, so passing out of the metal and lowering its quality in the way which Dr. Moldenke attributes to "burning."

In regard to the allusion in the paper to the reducing tendency of the cupola as compared with the oxidizing tendency of the air furnace, these terms are relative and were used by me solely in a relative sense, and I think it will be admitted by Dr. Moldenke that this is justifiable, because the oxidizing influence of the blast in the cupola is partly offset by the fuel bed below, whereas in the air furnace the oxidizing tendency of the flame is much greater and is not offset by any circumstances whatever. Therefore, I think it will be admitted that the likelihood of the intro-

duction of oxygen in the air furnace is much greater than it is in the cupola—the idea which I intended to convey.

In reply to Mr. Hibbard, I can only say that if he will take a trip around among the steel works and obtain the confidence of the metallurgist or the steel mill superintendent, either of them will not only tell him that the quality of the steel is seriously affected by an unknown quality of the iron from which it is made (not shown by the analysis of the latter), but that the situation so produced is a serious one against which they have constantly to be on their guard. This, as far as I can find out, has passed from the realm of dispute into the realm of accepted facts among those who know.

In answer to W. R. Webster, I would say briefly that the difference between high-oxygen cast iron and semi-steel is briefly that the high oxygen content reduces the graphite particles to a small fraction of their area in ordinary cast iron, while semi-steel lowers the graphite present as a whole by diluting the relatively high carbon cast iron with relatively low carbon steel. This makes a less quantity of graphite to exert an injurious influence and there may be a certain oxidation of the steel scrap, and consequent introduction of oxygen into the bath, with beneficial result, but this does not compare with the results to be obtained in an iron with normal or slightly subnormal carbon which contains a sufficient quantity of oxygen to suppress the formation of flat flakes of graphite. The iron produced by the latter method is stronger and capable of more satisfactory handling both in the foundry and in the machine shop.

In reply to C. H. Strand, I will freely confess that I cannot answer his question positively. In a general way I should be inclined to connect the formation of graphite nests, such as shown by his photomicrographs, with the formation of windrows of graphite, the formation and cause of which were pointed out in my paper, *The Effect of High Carbon on the Quality of Charcoal-Iron*.¹ I should expect that these graphite nests indicated an iron rather high in carbon and very low in oxygen, consequently, a weak iron, but not being familiar with the phenomenon he points out in the shape in which he shows it, I cannot answer positively.

In answer to John Howe Hall, I admit freely that the data submitted in the paper are insufficient to enable us to attribute a given increase in strength to a given increase in oxygen, and it is true that some of the results on deoxidation were contradictory. At the same time the impression formed as a consequence of the more careful and more reliable of these results, that is, the ones later made, was that they did result in a considerable weakening of the iron. This is particularly true with aluminum, whose effect we understand much better than we do that of titanium.

¹ *Trans.*, xlv, 314 (1912).

In regard to Mr. Hall's attempt to connect up the quantity of carbon with the strength of the iron, I tried to make it clear in my paper that the conditions which led to high carbonization were largely similar to those which led to thorough deoxidation, and referring back to my previous paper I intended to make clear my opinion that both of these conditions contributed to weaken some irons. At the same time, a careful analysis of the carbons and oxygens shown in the original paper, and also in the additional data put on the blackboard at the meeting and given below, will show that there is no unvarying connection between total carbon and strength, but that almost as much evidence can be obtained for one side as for the other. This, however, is very far from being the case with the oxygen content, since this in every case is high with the strong iron and low with the weak iron. It is very doubtful if, even with the additional data here presented, any quantitative relation between oxygen content and strength, in the presence of so many other variables, can be established, but the qualitative effect is so clearly shown as to be obvious to all save those who do not desire to see.

			Mn	P	S	C	Si	O	Breaking Strength Pounds
A	4/20/13	10 p. m....	0.76	0.134	0.016	4.06	2.56	0.050	3,450
	11/1/13	10 a. m....	0.70	0.160	0.016	3.80	2.58	0.009	2,400
B	10/20/12	4 p. m....	0.30	0.150	0.020	3.75	1.17	0.068	3,700
	2/10/13	10 p. m....	0.46	0.119	0.015	3.90	1.00	0.065	3,400
									3,500
	10/14/13	4 a. m....	0.26	0.156	0.022	3.82	1.10	0.012	2,350
C	2/10/13	10 p. m....	0.40	0.119	0.015	3.90	0.70	0.055	3,400
									3,500
	3/11/13	4 a. m....	0.70	0.112	0.025	3.98	0.61	0.064	3,700
									3,825
	3/12/13	10 a. m....	0.72	0.106	0.020	3.76	0.60	0.061	3,950
									3,200
	8/10/13	4 a. m....	0.37	0.134	0.021	4.01	0.65	0.010	2,500
	11/9/13	4 a. m....	0.38	0.160	0.019	4.14	0.60	0.009	2,500

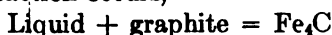
In reply to the remarks of Mr. Stoughton, I have only to express my thanks for the confirmation of my statements and views which he so clearly and comprehensively furnished.

In regard to Professor Campbell's remarks, it is most unfortunate that lack of time prevented his explaining in detail certain possibilities based on the latest developments of the iron-carbon diagram which he was good enough to explain to me privately. Through his kindness I am able to present here three possibilities based on the work of Guertler, who founds

his conclusions upon the experiments of Wittorff and Ruff, and one based on Upton's iron-carbon diagram. According to Guertler we have:

I. In the stable condition above $1,360^{\circ}$ C. we can have liquid + graphite.

At $1,360^{\circ}$ C. the reaction occurs,



and in alloys with 4.4 per cent. carbon down to 3.8 per cent. we find Fe_4C crystallizing out from the melt.

At $1,180^{\circ}$ C. we have the eutectic of austenite + Fe_4C .

At $1,110^{\circ}$ C. Fe_4C breaks up into austenite + graphite.

At 750° C. the eutectoid of ferrite and graphite forms.

These are equilibrium conditions which are reached only with extremely slow cooling.

II. With more rapid cooling the reaction at $1,360^{\circ}$ does not occur. Graphite separates from the melt and at $1,155^{\circ}$ we get the eutectic of austenite + graphite.

III. With more rapid cooling Fe_3C separates out from the melt and at $1,135^{\circ}$ C. and 4.3 per cent. carbon forms a eutectic with austenite; then at 710° C. we have the eutectoid Fe_3C + ferrite, commonly known as pearlite.

According to Upton's diagram white irons are supersaturated solid solutions which are breaking down; and the stable constituents are austenite + graphite above $1,095^{\circ}$ C.; austenite + Fe_3C or Fe_4C + graphite below $1,095^{\circ}$; and at 800° Fe_3C appears, while Fe_4C disappears. Fe_3C and Fe_4C have yet to be distinguished under the microscope.

From the point of view of practical value two possibilities according to these diagrams stand forth prominently:

1. Graphite can separate directly from the melt.

This is contrary to the claims of Professor Howe and Professor Sauveur, but is abundantly supported by the facts, while their contention that this action can only take place through the prior formation of cementite is completely negated by facts of practice which are matters of common knowledge to furnacemen.

2. After solidification various possibilities are presented according to both of these investigators. These possibilities indicate that in certain areas of the diagram a very slight change in conditions may cause a complete change in the resulting product. Professor Campbell's valuable suggestion is, as I understand it, that a small quantity of oxygen may be the determining factor in controlling which one of these conditions arises, and as a matter of great practical and industrial importance I know it to be true as the result of abundant investigation that not only the shape of the graphite is greatly influenced by a change in the oxygen content of the iron, but that the structure of the iron itself appears to be fundamentally altered by the presence or absence of this element. When it is

absent certain irons tend to form the iron-carbon eutectic, whose effect is absolutely detrimental from every point of view, while the presence of oxygen appears to suppress this structure altogether and forms instead a network structure concerning which the microscope furnishes the presumption that it should possess superior strength, a presumption abundantly borne out by physical tests. It would seem that scientific metallurgists interested in this subject may find interesting work for years to come along this line.

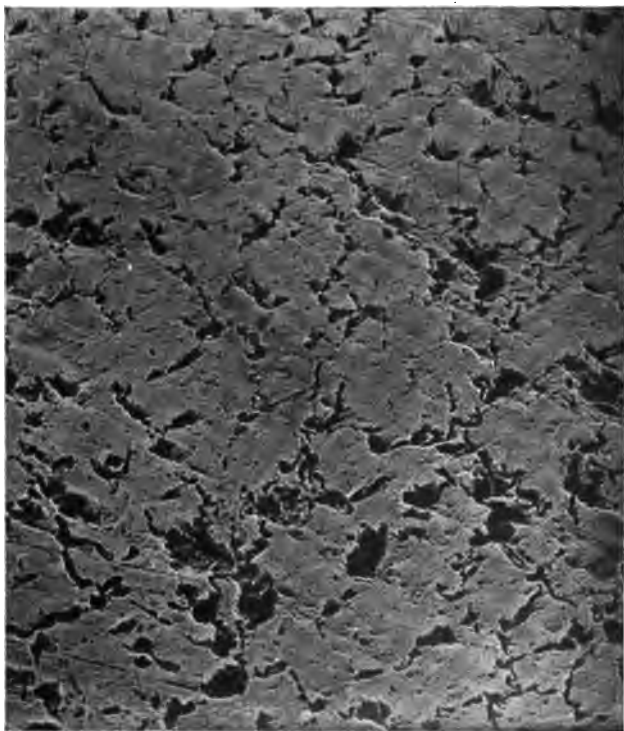


FIG. 3.—COKE IRON + OXYGEN. UNETCHED. $\times 100$.

Turning now to the practical point of view and in particular answering the criticism of Professor Sauveur that our tests do not carry conviction that the oxygen is the cause of the change because, he says, remelts with spongy No. 6 do not prove our case, I will furnish additional evidence developed since this paper was written.

A short time ago I took a heat of coke iron containing about 1 per cent. silicon, from which test bars were made as it came from the cupola. This heat was put into a Stoughton side-blow converter and had the silicon blown out of it at a temperature as low as we could safely hold it. The blow was stopped before much of the carbon had gone, and the

resulting metal, which was a wild white iron full of blow-holes, utterly rotten and worthless, as shown by samples which we poured, was mixed with an equal weight of 2 per cent. silicon iron from the cupola, from which test bars were made previous to mixing. From the mixture additional test bars were made. These included $1\frac{1}{4}$ -in. rounds, as with the two kinds of iron from the cupola, and also 2-in. square bars. These were broken by Tinius Olsen & Co. in Philadelphia. The results on the cupola irons of both heats ran from 2,000 to 2,500 lb., while the bars made from the mixture of converter metal and cupola metal broke at from 3,900 to 4,200 lb. The 2-in. square bars of the latter mixture broke at



FIG. 4.—COKE IRON + OXYGEN. ETCHED HNO_3 . $\times 100$.

from 19,000 to 24,000 lb., the latter figure considerably in excess of the most rigid requirements of the purchasers of Southern and Eastern charcoal irons. It will be seen that by mixing the siliconless converter metal with 2 per cent. silicon metal we restored the iron practically to the analysis of the original melt in everything save oxygen, and this seems to me to furnish as conclusive a proof as the most exacting can require that it is oxygen and oxygen alone which produces these results. This iron not only had the strength, but also had the close grain, the high density, and

the great chilling power of the best grades of charcoal iron, from which it could not be distinguished by the most careful examination. Photomicrographs of this iron, etched and unetched, Figs. 3 and 4, are presented herewith on the same scale as those in the original paper. The similarity of this iron to those charcoal irons will not, I believe, be denied by any one, and I believe those familiar with the microstructure of malleable castings will be struck with the similarity of the unetched specimen to the structure of malleable iron.

It is suggested in the paper that the explanation for the action of oxygen is that it reverses the points of graphite formation and solidification, which ordinarily occur in the order stated, but which in these irons containing oxygen, in my opinion, occur in the opposite order, as shown by the suppressed condition of the graphite in the photomicrographs. This opinion receives confirmation from a direction which did not occur to me at the time the paper was written. It is a matter of common knowledge among users of high-grade irons that strong irons of the kind they desire have low shrinkage, while poor irons have high shrinkage, and the shrinkage test is actually used as one of the determining factors in the purchase of iron.

Now, if the foregoing explanation be correct it explains this fact also, because if graphite formation be delayed until after the partial solidification of the iron matrix, then obviously when the graphite does form the increase in volume which its formation involves will expand the partly solidified iron, and this expansion will partly offset the subsequent contraction due to cooling, so that the net shrinkage of the iron will be small. If, on the other hand, the graphite separates out while the iron is still perfectly liquid, this action obviously does not occur. The contraction due to cooling is not offset by this expansion due to the formation of graphite, and the resulting shrinkage is greater than in the other iron.

Thus, this explanation agrees with another important factor, well established by practice, which seems to me to increase the probability of the correctness of the explanation very materially.

In the commercial aspect: While producers of coke iron ignore the fact, the manufacturers of high-grade castings pay premiums of from \$10 to \$25 per ton for irons having these qualities. They do this not through inertia, not through ignorance, but only after thorough and convincing trials of coke iron as a substitute have demonstrated beyond doubt that they cannot obtain the results that can be obtained with these other irons. There is not only one such foundryman in the country, there are scores.

These expensive irons do not differ chemically in any way whatever from coke irons of similar analysis except in regard to oxygen.

Admitting that we have all, myself most strenuously with the others, until recently, believed that oxygen was a serious detriment to the quality

of iron, have we any objective evidence whatever to support this view? Admitting the old idea that a difference in oxygen is accountable for these differences in quality, does not the view that its presence is responsible for the improvement agree far better with the actual facts than the view that its absence causes these differences? Are we not, therefore, entitled to lay aside our preconceived ideas, our prejudices, our beliefs and our theories, in order to accept the concordant results of a vast mass of uncontroverted and, I believe, uncontrovertible evidence of which this paper furnishes only a small part, and against which the only objection urged is that it is new?

Manganese Steel, with Especial Reference to the Relation of Physical Properties to Microstructure and Critical Ranges

Discussion of the paper of W. S. POTTER, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 88, April, 1914, pp. 601 to 634.

ALBERT SAUVEUR, Cambridge, Mass.—While I do not think that Mr. Potter's conclusions in regard to the existence of several critical points in heating and cooling manganese steel are fully supported by the appearance of the curves exhibited, the results of his heat treatment and physical testing are most important and instructive. It seems to me that Mr. Potter has shown in a conclusive manner the breaking up of austenite on heating manganese steel, and that he has located accurately the limits of the range in which such breaking up takes place. I see in his results a confirmation of a view I have held for many years: namely, that the reheating of quenched manganese steel is, in reality, a tempering operation, even if it implies an increase rather than a decrease of hardness. Undoubtedly it represents a return to a more stable condition—the essential feature of the tempering operation.

Quenched manganese steel is austenitic; that is, its condition is that of a solid solution in which the iron is in the gamma form. This is proven by (1) its structure, (2) the absence of magnetism, (3) the absence of critical point during the cooling of such steel, (4) its moderate hardness, high tensile strength, high ductility, and relatively low elastic limit. On reheating the metal to about 500° C., the austenite begins to break up through the formation of some beta iron, as evidenced by increased hardness and greatly decreased ductility. At some 500° C. some alpha iron also forms, since the metal becomes magnetic. The behavior of manganese steel when subjected to heat treatment appears to me to fall in line, in a remarkable manner, with the allotropic theory of the hardening of steel as first outlined by Osmond. The fact that, on tempering austenitic manganese steel, its hardness *increases* affords, in my opinion, a very strong evidence of the existence of a hard variety

of iron between the soft alpha variety and the moderately hard gamma variety; in other words, of beta iron.

SIR ROBERT HADFIELD, London, England (communication to the Secretary.*)—I have read with much interest Mr. Potter's paper on Manganese Steel, and compliment him upon his excellent photomicrographs.

In view of his opening remarks, in which, no doubt unintentionally, he states that manganese steel has only been known and introduced for about a decade, it seems to me he hardly does justice to the many willing helpers who have worked long before the decade of which he speaks, on manganese steel, which I discovered now about 30 years ago. When, therefore, he speaks as if this material, with its many wonderful qualities, has only been in use for the short space of 10 years, he evidently does so without being aware of the real facts. Manganese steel, both in its cast and forged forms, has been commercially used for more than 20 years. Therefore the statement referred to hardly seems to do justice to those in England and America who have been working so patiently and for so long a period and trying to master the intricacies of this interesting metallurgical product.

As a matter of interest to those of our younger members who are coming to the front in metallurgy, it may be interesting to make a brief mention of those to whom credit has been due.

The first in this country to take an interest in manganese steel was the late Joseph D. Weeks, of Pittsburg, a very able writer and well acquainted with much of the practical side of metallurgy. He read the first paper in this, or in fact in any country, on my material, before this Institute in 1884. He met my father and myself in Europe just at the time the development of this steel was being made, and was so much interested that he asked if he might say something about it in America; hence the paper in question.

Not long after, Prof. H. M. Howe came over to Europe, and was so much struck that he also asked if he could speak of the research work we were then carrying on. As is well known, he gave a considerable and lengthy account of manganese steel in his great work, *The Metallurgy of Steel*.

Then came my friends the late W. J. Taylor and Maunsel White, also H. D. Hibbard, each of whom did much valuable work on manganese steel now some 23 years ago. Professor Howe read several papers regarding my discoveries, including those before the American Society of Mechanical Engineers in 1891, and the Franklin Institute in 1892. As I have just said, he also devoted considerable space to describing and commenting upon the material in his well-known *Metallurgy of Steel*,

* Received Mar. 27, 1914.

published 25 years ago. Since then, younger members of the Taylor Co., including Knox Taylor and others, have also worked indefatigably at the problem of developing and improving manganese steel during the last 15 years.

Among others in this country who did pioneer work with regard to the material are C. W. Hubbard, Pittsburg; the late respected William Metcalf, of the Crescent Steel Works, Pittsburg; also his firm, Miller, Metcalf & Parkin, all of whom took a practical interest in the subject; P. H. Dudley; Mr. Harrington, mechanical engineer of the Pittsburg, Cincinnati & St. Louis Railway; S. T. Wellman, of the Otis Steel & Iron Co., who made some of the earliest mechanical tests; E. G. Spilsbury, one of the first to experiment with manganese-steel wire; T. T. Morrell, chief chemist of the Cambria Iron Works; J. P. Shinn, also his brother; the Lehigh & Wilkes-Barre Coal Co.; and many other individuals and corporations.

It may be of interest to also add that among those who took an interest in developing manganese steel on the continent of Europe were: in France, M. Saladin, of Messrs. Schneider; Chatillon & Commentry Co.; Messrs. Gautier, Osmond, Pourcel, Dumas, H. and A. le Chatelier, and many others. In Germany, Messrs. Krupp and their Directors, Dr. Schmidt and Dr. Ehrensberger; Professor Ledebur, Dr. Muller, and others.

I have only mentioned a few of those who have been interested in doing research and practical work on manganese steel. There are, of course, many others, and I have expressly avoided referring to those concerned in this matter in Great Britain, in this reference to what has now become a world industry.

As a landmark and date for reference, it may be mentioned that the first manganese steel I ever produced, as duly registered in my experimental records, was in March, 1882.

The work done and the interest taken by those mentioned—all men of well-known metallurgical reputation—took place not a decade ago, but in most cases from 15 to 20, or even 25 years ago. It is only fair that their work should not be forgotten. While we are glad to have new investigators in the field of research, those who have done good work in the past—alas, a number of them not now with us—should not be overlooked.

With reference to the remarks about grain adhesion,—we would prefer to use the term “grain cohesion”—our experience, going back over experimental and practical commercial work for 25 years, has been somewhat in the opposite direction to that of the author, and we believe that the well-defined etching crystalline structure is evidence of good-quality steel.

As regards “crackling,” this we have often noticed in test bars, and



FIG. 1.—NEARLY PURE AUSTENITE OF QUENCHED MANGANESE STEEL. $\times 50$.



FIG. 2.—AUSTENITE CHAINED BY HAMMER BLOWS. SHOWS NEUMANN LINES IN THE CRYSTALS (Slip bands). $\times 150$.

we consider it to be due to the cracking of the decarbonized surface skin. It will be noticed this chiefly occurs in bars heated to the higher temperatures, which supports our theory.

The author refers to three temperature regions where the metal is uniformly austenitic. We consider that above a certain temperature, probably about $1,000^{\circ}\text{C.}$, uniform austenitic structure exists. In other words, above $1,000^{\circ}\text{C.}$ the cementite is all dissolved, resulting in a single uniform constituent of austenitic structure.

As regards the remarks on p. 617, the author states that in region three the temperatures of $1,125^{\circ}$ and $1,225^{\circ}\text{C.}$ have been found useful for the heating of ingots for rolling. A research was made regarding the temperatures used for rolling ordinary carbon-steel rails, and it was found that the regular practice ran in some cases to as high as $1,170^{\circ}\text{C.}$ or more.

Some of the photographs shown by the author appear to be covered with parallel lines. It is probable that these parallel markings are Neumann lines (slip bands), and are due to the fact that manganese-steel sections are somewhat difficult to prepare. In fact, owing to their toughness, if they are knocked or hammered off a larger piece, this disturbs the cleavage planes of the austenitic crystals. We have produced such a result, as shown by the accompanying photomicrographs. Fig. 1 represents pure austenitic structure, and Fig. 2 is taken from a part of the same section after having been knocked and hammered off. The latter shows the parallel lines due to the mechanical disturbance of the austenitic cleavage planes caused by hammering. They appear more numerous in this photograph than those shown by the author because the magnification is less, but on the other hand a larger field is shown, and it is interesting to note that some of the crystals have stood the treatment quite unmarked.

As regards the effect of heat treatment, it may be of interest to give the result obtained from one of my earliest specimens of manganese steel.

These tests were made in 1887, and are of interest as showing the effect of heat treatment, namely, (A) as forged; (B) afterward heated to a high yellow and cooled in air; (C) cooled in oil; (D) cooled in water. It will be noticed the elongation varied exactly in accordance with the relative conductivity of the medium in which the specimen was cooled.

It is also of interest to note that this was the first bar of steel ever produced in the world's history which gave the exceedingly high tenacity of 67.13 tons per square inch accompanied with 44.44 per cent. elongation, *and yet the specimen still remained unbroken.* These tests were carried out by the British Admiralty Inspector, in the year 1887, details being given in the following table:

Tests on Hadfield's Manganese Steel

Composition: C, 1.26; Si, 0.28; S, 0.05; P, 0.11; Mn, 14.01 per cent.

Mark	Treatment	Breaking Load Tons per Square Inch	Elonga- tion on 8 in., Per Cent.	Appearance of Fracture
A	Forged into test bar and no further treatment	36.43	1.56	Fine grain
B	Heated to high yellow and cooled in air	47.90	14.06	Granular; fine grain
C	Heated to high yellow and cooled in oil	55.04	26.56	Fibrous
D	Heated to high yellow and cooled in water	67.13	44.44	Fibrous*

* The Admiralty Inspector who carried out tests on this bar remarked: "Notwithstanding 44.44 per cent. elongation, and a tensile load of 67 tons, this bar is still *unbroken*." It is probable that the final elongation would have been 51 to 52 per cent., with a breaking strength of a little over 70 tons per square inch.

The author of the paper states that the maximum results are obtained at 840° C., but we have often obtained both superior tenacity, and equal if not superior elongation at temperatures in some cases as high as 940° C., or over 100° C. higher than those mentioned by the author.

Manganese-Steel Rails

Discussion of the paper of SIR ROBERT HADFIELD, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 86, February, 1914, pp. 259 to 265.

W. S. POTTER, Pittsburg, Pa.—The second page of this paper contains the repetition of the substance occurring at the bottom of the first page, i.e., "There never has been any real difficulty in obtaining forged, rolled, or pressed manganese steel."

The writer refers to manganese-steel rails made by the Schneider Co. for the Metropolitan Electric Railway of Paris. In the *Revue de Métallurgie*, Paris, 1909, vol. vi, p. 550, in a letter dated Feb. 6, 1909, the Schneider Co. makes the following statements concerning the history of its efforts in the manufacture of manganese-steel rails:

"Our first tests made in this direction go back to March, 1904, and consisted in the rolling of some rails of 7.5 kg. per meter, obtained by the transformation of billets forged at the drop hammer. These tests were continued in February, 1905, by rolling, under the same conditions, two rails of 18 kg. to the meter.

"In August, 1906, and in anticipation of a furnishing of rails to the Metropolitan of Paris, we proceeded to a preliminary rolling of rails of 20.35 kg., which permitted us to recognize in advance the difficulties of manufacture which we would encounter, and

to take all suitable measures for surmounting them. Six rails of 7 m. were thus rolled. The operation presented this peculiarity, that the transformation of the ingots into rails was effected exclusively by rolling (without preliminary forging of the ingots). Finally, from March to July, 1907, we proceeded under excellent conditions, to the rolling of 60 tons of Vignole rails, of 52 kg. to the meter, from 6 to 12 m. in length, at present in service at the Metropolitan Railroad of Paris."

On p. 560 of this journal the Schneider Co. makes the following statement:

"Manganese steel may be forged and rolled." Nevertheless, the quite special pains which its heating demands, as well as the very narrow limits of temperature between which one is restricted in working it, render these operations, the rolling of rails of great lengths especially, particularly difficult.

"We have, however, succeeded in rolling, in July, 1907, an important tonnage of Vignole rails, at present in service on the Metropolitan Lines of Paris."

The makers of the rails referred to by Sir Robert Hadfield differ radically from him as to their experience in the heating of manganese steel for rolling or forging. Until 1906, and after two years' experimenting, they were still unable to roll the manganese ingots without first forging them. Thereafter they found that the heating required "quite special pains" and that the metal could only be heated within "very narrow limits of temperature between which one is restricted in working it."

Personally, I have had 14 years' experience in the manufacture of cast manganese steel, rolled and forged manganese steel, and it would certainly be absurd for me to say that I had found "no difficulty," particularly in the heating of the cast metal for its conversion into rolled or forged forms.

Nine important steel companies in the United States, one in France, and two in England have wrestled with the heating of the cast manganese steel in their efforts to devise a practical method for its conversion into the wrought state. Hundreds, if not thousands, of men on both sides of the water will agree with me when I say that the preparation of the manganese-steel ingot for rolling is the very name of trouble. This art is entirely the opposite of what one would understand from reading Sir Robert Hadfield's paper submitted to-day to this Institute.

The first bar of rolled manganese steel which I ever saw was made by the Edgar Allen Co., of Sheffield, England, and was given to me for analysis when I was chemist of the William Wharton, Jr., Co., of Philadelphia. I found manganese 12.6 per cent. It was a very fine-grained, smooth bar, about 3 in. wide and perhaps $\frac{3}{8}$ in. thick. This was in the early part of 1901.

During the period 1901 to 1909 (more or less) many good men on both sides of the water struggled with the difficult matter of devising a method of heating an ingot of manganese steel in such a manner that it might then be rolled so as to afford a commercially available product. Outside of England and this country, no important progress in this art has been

recorded. In Sheffield, England, the Edgar Allen Co. worked out a method, and in accordance with it rolled more rails than have ever been made by any company outside of the United States.

A representative of the Edgar Allen Co. described the successful Sheffield practice as follows:

"You 'eat it just so 'ot. Then you rowl it to the center, rowl it to the center, you understand. If you 'eat it just a little bit too 'ot, you crush your — ingot. If you don't 'eat it quite 'ot enough, you break your — rawls."

In other words, as per the written word from the Schneider Co., the temperature range as per the English practice, as developed in Sheffield and transmitted to the works of the Schneider Co., covered "very narrow limits of temperature between which one is restricted in working it."

During the same years, that is to say, from 1901 to 1907, inclusive, there was developed in this country a method of heating manganese-steel ingots for rolling, which, instead of being limited to a very narrow range of temperature (which according to the best information as to the Sheffield practice was from 850° to 950° C.), enabled the operator to go to 1,150° to 1,250° C. in the heating of the ingots. The result was a soft, ductile ingot, corresponding with a 20 per cent. carbon-steel ingot ductility.

In 1912 the Edgar Allen Co. was still selling manganese-steel rails at approximately \$250 per ton. In 1913 they took license under my English patents and immediately reduced their price on rolled manganese-steel rails to approximately \$125 per ton. The American price for several years has been \$80 per ton.

J. W. RICHARDS, So. Bethlehem, Pa.:—We probably all know that Mr. Potter has done more work than any one else in the United States on the rolling of manganese steel, and we all know, also, that Mr. Hadfield was the pioneer in the production of the metal, and that he certainly did forge and roll it a great many years ago. But the reading of his paper seems to confirm Mr. Potter's statement that he minimizes the difficulties and as the proof of the pudding is in the eating thereof, the fact that by Mr. Potter's method manganese-steel rails are now being rolled and sold at less than one-half the price at which they were sold before his process came into use, would show that his methods are essentially a great improvement upon any previously known method of rolling this material.

SIR ROBERT HADFIELD (communication to the Secretary*).—With regard to Mr. Potter's remarks, I notice that he quotes a sentence of mine in which it is stated "There never has been any real difficulty in obtaining forged, rolled, or pressed manganese steel." As set forth in my paper, I again state that we in England have been rolling, forging, and pressing manganese steel as far back as the years 1887 to 1892, and have not found any special difficulties provided the ingots were made in a proper manner. Soon after the years referred to, among other articles, large quantities of

* Received Apr. 16, 1914.

dredger pins, all of which were forged, and of quite large diameter, $3\frac{1}{2}$ to 4 in., were produced. This was referred to in my paper to this Institute in 1893; that is, more than 20 years ago. In view of the fact that it was then possible for us to produce forgings and bars from ingots of small and large sizes, it will be seen that my assertion is thoroughly confirmed by the facts.

According to recent statements in the technical press, it has only just been recognized by those new to this art that the question of making a suitable and satisfactory ingot is one of the chief points in connection with the successful production of forged and rolled manganese steel, a point which we in Sheffield have had in view during the last 20 years.

As regards Messrs. Schneider's remark that manganese steel could be readily forged and rolled, but that it requires care in manipulation, whether as regards heating or other portions of the process, this applies as much to-day as when the statement was made in 1907-1909. It will be admitted by all who deal with manganese steel that narrower limits of temperatures and methods of handling must be employed in its working. There is less latitude as compared with ordinary carbon steel, and it is this, which is the gist of the statement very correctly put forward by Messrs. Schneider, especially as they had in view the production of rails up to 12 m. in length, the accomplishment of which is even to-day not a matter of the greatest ease. This is so not merely on account of its being a rolling-mill question, but in connection with other portions of the work which is necessary to be carried out in the manufacture of these rails. There is therefore nothing in the statement made by Messrs. Schneider to which exception can be taken.

In any case, however, as Messrs. Schneider definitely pointed out, they rolled manganese-steel rails in March, 1904, and in August, 1906, they rolled about 20 tons of these rails, of heavy section, 7 m. in length. Then, a few months afterward, 60 tons of manganese-steel rails were rolled, of heavy Vignole section, weighing 52 kg. per meter, and from 6 to no less than 12 m. in length. These rails are still in service, and have given exceedingly satisfactory results, as referred to in my paper. As these were probably the first manganese-steel rails of large section ever put into service with satisfactory results, full credit should be given to Messrs. Schneider. Therefore there seems to be nothing special in the claims they make in the remarks quoted which might not at that time have been made by any one dealing with manganese steel in its then new application for rails.

As is well known by those accustomed to dealing with manganese steel, one of the chief secrets of success is the careful handling during the first stages of the forging or rolling treatment. In other words, as Mr. Potter probably knows, if manganese-steel ingots are heated to temperatures such as those to which he refers, $1,150^{\circ}$ to $1,250^{\circ}$ C., unless there is

careful handling in the rolls, the results will be disastrous. For example, if the rolls are too severe in their draft, then, apart from the question of temperature or the manner in which manganese steel is heated, it will not be easy to obtain satisfactory product.

It is of the highest importance that manganese steel when put to work should be perfect as regards its toughness, otherwise most disastrous results may ensue. This is referred to because some of the manganese-steel rails produced in this country are stated not to have been satisfactory. This is mentioned in the American Railway Engineering Association's report for July last year, where it is stated that a number of rails put into use were while in service discovered to be broken. This is another proof that unless the manufacture is carried out very carefully, a point I have always particularly insisted upon, the use of this peculiar material, manganese steel, may prove dangerous. As is well known, manganese steel in its brittle condition breaks much more readily than carbon steel. It is also easy to have intermediate conditions in which under ordinary tests the material appears to be tough, yet, owing to imperfect heating and toughening treatments, it has a dangerous propensity to break with little warning. Another point may be emphasized: namely, that if the toughening of manganese steel is carried on at too wide a range in temperature, the risk is run of serious breakages occurring sooner or later. In such case the material would be blamed, when it is not really at fault, but the incorrect methods of treating it. These facts have in the past been fully and specially emphasized in my various papers and other communications.

With regard to the further remarks of Mr. Potter that there have been difficulties to overcome in dealing with this peculiar steel, this statement of course is quite correct. Others among us interested in the forging and rolling of manganese steel have at times not felt very happy in the past when visiting the scrap heap! In view of the facts previously referred to, it must be admitted that the problem was solved in England more than 20 years ago; and as regards suitable methods for producing rails, these were fully set forth to the metallurgical world in 1903, as a result of experimental work which had been going on for some time previous to that year. For example, attention may be called to the fact that in a recent technical publication¹ it is stated that the method by which manganese-steel rails can be satisfactorily produced is "by using inverted molds (that is, ingots cast with the large end up and the small end at the bottom) with refractory tops, putting carbon on the top of the ingots after pouring, and keeping this in a state of incandescence with compressed air." If this system was not employed, the maker, it is stated, met with innumerable difficulties in handling the fluid steel, or, to use the language of

¹ *Troubles with Manganese Steel: Iron Age*, vol. xciii, No. 7, p. 447 (Feb. 12, 1914).

the article referred to, before using this device "We were making some of the most beautifully piped ingots you ever saw." Methods of this nature and similar to those above described have been urged by me ever since 1903, and full details have been set forth in my papers and other communications. Under the system my firm has made, in a most satisfactory manner, very large tonnages of perfectly sound manganese-steel ingots, and showing little if any more waste than ordinary carbon steel. Under the same system are produced carbon-steel ingots absolutely sound and free from piping.

It is therefore no little satisfaction to find that if the article referred to in *Iron Age* correctly represents the facts, as no doubt it does, the successful production of manganese-steel rails in this country is being based upon the system described so many years ago. If this system is a success for producing ingots from which manganese-steel rails are made, may I not urge that it also possesses the same advantages for ordinary steel ingots from which carbon-steel rails are made.

With reference to the statement that "Nine important steel companies in the United States, one in France, and two in England have wrestled with the heating of cast manganese steel," I can only repeat that long before the date mentioned, 1901, we had wrestled, and this most successfully, with the art of producing rolled and forged products of this steel. It will be seen, therefore, that it is not possible to agree with the remarks made under this head, and it seems to me that others, in the light of the above-mentioned facts, will take the same point of view.

Reference is made to the efforts of those who were not the inventors of manganese steel, and therefore could not have been the first to produce it. It was probably not until some 10 or 12 years after my company produced this material in rolled, forged, pressed, and other worked forms, that the firms referred to produced the material which I discovered and first worked at between the years 1883 and 1887, before announcing to the world in my papers this discovery of mine. Therefore, at least 15 years before 1901, the date when Mr. Potter says he first saw a bar of rolled manganese steel, my firm had been doing this, which was naturally so, seeing that I discovered the material. Moreover, in view of my papers to the Institution of Civil Engineers in 1888, and later to this Institute in 1893, and the references therein to the many tons of rolled bars, sheets, axles, tires, forgings and other forms of manganese steel, I am at a loss to understand why in the remarks referred to these evidences are overlooked.

With regard to the temperatures used for forging and rolling, there is full and definite evidence on record that the early practice of dealing with manganese-steel ingots from the year 1887 and onward was to heat the ingots very much hotter than the temperatures to which Mr.

Potter refers. He seems to have got an erroneous impression both with regard to the large quantities and as to the manner we in England have been rolling, forging and pressing manganese steel during the last 20 years. I should be very glad to show him our experimental records, and am sure he would then see that many of the suggestions he makes were anticipated by our early work. He would also then see the very large tonnages of forged manganese-steel work we have turned out from the year 1887 and onward.

I should like to point out, with reference to the remarks of Professor Richards, that there seems to be some misunderstanding about the question of the work previously done with regard to reducing manganese steel by forging and rolling from ingots. May I once more emphasize that for more than 20 years rolling, forging, pressing, and otherwise manipulating manganese steel in forms other than those cast has been carried out on a large scale. Long before the year 1900 many thousands of tons of such forged and rolled products were manufactured and used. The methods of carrying out such work—that is, the temperatures employed for rolling and forging—are practically the same to-day, 1914, as at the end of the last century.

Moreover, in the paper presented to this Institute by the writer in 1893, entitled *Iron Alloys, with Special Reference to Manganese Steel*, full particulars and photographs are shown of manganese steel (a) forged into railroad axles, (b) rolled into railroad tires; also many other worked forms of this material.

As regards the selling price, that is quite another matter. There were in former years many reasons why the selling price should not have been reduced, but the cost—and this is the important point—of working manganese-steel ingots into, say, billets, blooms, bars, or forgings was, as just mentioned, pretty much the same many years ago as now, excepting of course improved machinery, such as more powerful rolling mills, presses, etc., effecting diminution in cost; this, however, would not make a large reduction in the cost. It would be quite possible to show the original costs of working manganese steel and compare them with to-day's figures, in order to show that the above statements are correct; in fact, owing to higher wages and more costly fuel, probably there would be a slight increase in the cost.

Notes on Blast-Furnace Operation with a Turbo Blower

Discussion of the paper of S. G. VALENTINE, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 86, February, 1914, pp. 291 to 297

J. E. JOHNSON, JR., New York, N. Y.—I did not have a chance to read Mr. Valentine's paper before the meeting and I would like to know how he reaches the figure of 41 cu. ft. of air per pound of coke. It is

only two-thirds of the amount theoretically required and would indicate that one-third the total coke was lost by solution and never reached the hearth at all.

S. G. VALENTINE.—I would like to say, in reply to Mr. Johnson, that the figure of 41.7 is based on the calibration of the turbo blower. That calibration, I am advised, the General Electric Co. guarantees to be correct within 5 per cent., and they claim that it is really correct within a closer figure than that. Taking our gas analyses, as I have done in the paper, and where the figures are given, and figuring back from that I came within 8 per cent. of that, my figures in the calculation being a little higher, running 45 cu. ft. of air per pound of coke charged. Of course, I am not able to certify to the correctness of that engine calibration, but understand it is based on very accurate and carefully made measurements.

J. W. RICHARDS, So. Bethlehem, Pa.—I am not speaking as expert on the mechanical side of the turbo blower, but as I understand it, it is adjusted so as to take in a constant amount of outside air and deliver it to the furnace, so that its delivery is not a variable amount of air under variable pressure, but a constant amount of air under whatever pressure the furnace gives against it. The calibration, being done upon the inlet, is probably done very accurately.

Mr. Valentine's figures show that much less air is received by a furnace than it is ordinarily credited with. The old way of getting the amount of air going into the blast furnace, from the piston displacement, is a very rough and approximate way of getting at it, and we all know that the more accurate way is to take the analysis of the gases and work back from the balance sheet of the furnace, to find out how much air the furnace is receiving. This is more accurate than to take the piston displacement and make an allowance for efficiency. In fact, you can in that way compare the net delivery of a blower plant to the piston displacement of the engine. I have done that in many cases and it figures out, according to the modernness or the antiquity of the blowing engine, to an efficiency of delivery of anywhere from 85 down to 50 per cent.

The blast-furnace man who takes his piston displacement and tells you from that how many cubic feet of air he is blowing in per pound of coke, is giving you, probably, a misleading idea of how much air is entering his furnace.

The feature of blowing in a constant amount of air against the variable back pressure of the furnace is the most valuable thing about the turbo blower. The ordinary engine, without such regulation, gives the furnace the least amount of air when it needs the most; that is, when it is becoming stopped up and the back pressure is rising. The turbo blower is supposed to automatically deliver a constant amount of air under the variable pressures. We have there in an extremely valuable

improvement in blast-furnace practice. I think it might be still further improved if the amount of air could be increased as the amount of moisture in the air increases (if the furnace has not a dry-blast plant) so as to compensate to some extent for the variable amount of moisture which is in the air.

I have calculated the volume of air received by Mr. Valentine's furnace, from the oxygen in the gases, and find it to correspond within 0.1 cu. ft., per pound of coke charged, with the value calculated by him on the basis of the nitrogen in the gases. This agreement checks up the accuracy of his data, and makes it in my opinion quite certain that his furnace was receiving just about 48 cu. ft. of air (at 60° F.) per pound of coke charged. If the regulator on the turbo blower registered less than this, I believe the difference due to inaccuracy in the calibration of the registering apparatus; if our reciprocating engine friends say that this is an impossibly small amount, the answer is that they have been too long accustomed to working from piston displacement allowing too high coefficients of delivery, and so have become accustomed to charging their furnaces with more air than they were actually receiving. I consider this important point fully proved by Mr. Valentine's data.

J. E. JOHNSON, JR.—It is perfectly true that the reciprocating blowing engines have efficiencies as low as 50 per cent. in some cases. I have made some tests along this line and I know that this is so. Nevertheless, I think that good engines such as are in common use now will show an efficiency of about 90 per cent. of the theoretical, including everything. It seems to me that the volumetric efficiency of Mr. Valentine's turbo blower must have been about 120 per cent., as I do not believe the results which Mr. Valentine reports can be obtained on a highly refractory burden when only two-thirds of the coke is being burned by the blast and the other third by oxygen from the ore. I think we must have more definite and positive information as to the nature and accuracy of the calibration of this apparatus before we should be asked to accept these figures.

KARL NIBECKER,* Youngstown, Ohio.—Mr. Valentine's paper as presented is of the most vital interest concerning the operation of a blast furnace when blown with a turbo blower. I regret exceedingly that up to the present time we do not have sufficient data available to enable me to make a direct comparison with the results obtained by Mr. Valentine. The machines which the company with which I am employed is at present operating have not up to the present time been tested sufficiently to enable me to give definite figures, but, from the data which we now have available, it appears as though the results which we are obtaining will not check in several respects with those as given in the paper.

Has Mr. Valentine ever calibrated the constant-volume governor as furnished by the builders of the blower? Has a Pitot tube, or some

* Non-member.

similar standard device for measuring air, ever been used in order to check the graduation of the regulating beam with the amount of air actually being delivered by the blower? It has seemed to me that it is quite important to carefully calibrate the measuring device as furnished with the blower with the machine actually operating under service conditions. It seems to me highly probable that the calibration furnished by the builders of the machine under test conditions may not be accurate under working loads. The calibration may be quite different under actual operation when blowing a blast furnace.

It has been our experience that in many instances a sufficient correction factor for the slip and losses in a reciprocating engine is not assumed of a proper value, and this may account for the figure obtained by Mr. Valentine as to the amount of air delivered by the engine. It is also probable that the amount of air as figured for the turbo blower may be in error, even though a calculation of the air delivered to the furnace is made from the gas analysis and is found to check within a small per cent.

It is seen that the calculated amount of air required by the furnace is more than that delivered by the blower, if we accept the calibration of the regulator as correct.

We have noted conditions where the amount of air calculated from the gas analysis does not check with the air delivered by the engine after proper corrections have been applied to the engine, and I am therefore inclined to question the absolute value of checking the air delivered by an engine in this way, except as a most general indication.

I would like to ask Mr. Valentine whether he has made any comparison of the amounts of steam required per ton of metal produced by the two methods of blowing. It seems to me that this steam consumption is of the utmost importance in any furnace plant where an excess of steam can be made available for use in power generation. From our experience, it seems as though the turbine will deliver very nearly the same amount of air per pound of steam consumed as the reciprocating engine, providing that they are both working at the best vacuums for their respective design. It is probable that the turbine will consume slightly more steam than the engine per 1,000 cu. ft. of air delivered, but we do not have the data to state the exact ratio.

The type of governor with which our machine is equipped is the multiple Venturi tube of the Rateau design and seems to give a very constant flow. While the Venturi tube may be an accurate means of measuring air, we are inclined to believe that the setting of the regulator on this governor, of whatever type used, should be calibrated in place by some reliable means of measurement, such as the Pitot tube or throttling disk.

LEONARD WALDO, New York, N. Y.—I have lately had occasion to get air into open hearths under varying pressures, and I have been very much troubled by the fact that, with a guaranteed pressure of 120 lb.

on the mains, the pressure gauge of the main at the open hearth showed remarkable facility in dropping down to 50 and 52, considerably less than the pressure required for the service.

In the case of the turbo blower, I had hoped that we had gotten something in which the volume of air bore some relation to the pressure, for, after all, it is not the pressure which does the work, but it is the flame which takes place through the combustion of the fuel, and I think that in the minds of most people who are looking for air the turbo blower is the most promising thing, if the expense of the production of the air is not too great.

The open-hearth people have a strange faculty of translating these things into cost per pound of compressed air, and the people who put in these turbo-generator plants have a habit of placing the apparatus clear off at the end of the building, and driving them by belts, with the result that there is quite an increase in the charge. The air is very nice, but it costs like hot air generally does. Therefore, this paper is a move in the right direction, and I sincerely hope that the turbo blower is going to be a solution of the very great difficulties confronting us in the modern problems of combustion, in having adequate air in the right place at the right time and under the right pressure.

J. E. JOHNSON, JR.—I do not wish to take up an undue amount of time in discussing this subject, but there is one point raised by Mr. Nibecker which is very important; that is, the probability that the turbo blower takes more steam than the reciprocating blowing engine. It is a more or less open secret in steam-engineering circles that from boiler pressure down to atmospheric pressure at least, the reciprocating engine is under no disadvantage as compared with the turbine in regard to economy, and a good many cases are on record where people have gone from turbines back to reciprocating engines in power plants because they could obtain just as good economy and liked the operation better. The efficiency of the blowing engine, mechanically, is very high. Nordberg built some compressors which gave 95 per cent. mechanical efficiency, and I have in my possession some indicator cards from large blowing engines which show 92 per cent. A properly designed high-speed blowing engine has a high mechanical efficiency, and the thermodynamic or diagram efficiency of the air end is also high, whereas the turbo blower has an efficiency of only 70 per cent., which means, considering the two efficiencies together—that of the compressor and that of the engine—that the turbine must use less steam than the steam engine in the ratio of 70 to 90, and it is extremely doubtful if it can do this even with high vacuum. It seems to me that the opinion of the Youngstown Sheet & Tube Co., that the turbo blower requires more steam per unit of air delivered than good reciprocating blowing engines such as they have, is probably correct.

KARL NIBECKER.—I would like to ask Mr. Valentine if he has noticed any difference in air pressure required at the furnace to produce a given amount of iron with the turbo blower as compared to the engine. I have heard the statement made that the blower will produce the same amount of iron on less pressure than the engine, the theory being that the constant flow from the rotating machine will cause the same amount of air to be delivered through the furnace with less pressure. If this condition is true, it should be possible to blow a furnace with less horse power when a turbo blower is used than with a reciprocating engine. This would apparently indicate less steam per ton of iron. Whether this difference in air pressure and work done is enough to reduce the steam consumption on the turbine, I am not prepared to say.

If Mr. Valentine has any data concerning the steam, it will be of considerable interest.

J. W. RICHARDS.—I would ask Mr. Johnson, assuming that the figures in Mr. Valentine's paper are accurate within 10 per cent., whether this does not show that this furnace was working with an excess of heat in the hearth and a deficit in the shaft, and thus explain the fact that carbon was burnt economically in the upper part of the furnace?

J. E. JOHNSON, JR.—I should say it proves exactly the opposite.

GEORGE C. STONE, New York, N. Y. (communication to the Secretary*).—I fully agree with Mr. Valentine's conclusion that: "The turbo blower measures the volume of air used with more accuracy than the calculations based on piston displacement in other types of blowing engines; that as a corollary to this most furnaces run on reciprocating engines are using less air than they generally figure."

One reason for this he appears to have overlooked. In all of his calculations the temperature of the air is assumed to be 60° F. and its weight 0.0763 lb. per cubic foot. This is a fair assumption for the air in the engine room, but the cylinder, its heads, and the piston are at a much higher temperature, due to the heat of compression; consequently the air is heated on entering the cylinder, and the latter is filled with air at a temperature considerably above 60° and, therefore, contains a less weight of air than the calculation calls for.

The heat of compression is very considerable. Kent gives it as follows:

Gauge Pressure, Pounds	Temperature of the Air, Degrees F.
0	60
5	106
10	145
15	178
20	207
25	234

* Received Mar. 2, 1914.

The weight of air also varies considerably with its temperature, as follows:

Temperature, Degrees F	Weight of Air, Pounds per cu. ft.	Relative Weight
20	0.08274	108.3
40	0.07943	104.0
60	0.07638	100.0
80	0.07352	96.3
100	0.07090	92.8
120	0.06845	89.6
140	0.06617	86.6

The worst case Mr. Valentine gives shows a difference of only a little over 6 per cent. between the engine ratings and the theoretical. This would correspond to a temperature of the air in the cylinder of about 90° instead of 60°. This could readily be accounted for by the heating of the air by the piston, cylinder walls, and ports when working at very moderate pressures. This, I think, accounts for the fact stated in his corollary. As a further corollary it would follow that the higher the blast pressure the less the weight of air delivered.

A six-stage turbo blower should not be affected by the heat of compression to nearly as great an extent. As the compression in the first stage is very slight the heat imparted to the incoming air is also very slight. Aside from this the rating of the turbo blower is based on calibration, which would correct for errors due to this cause. This probably accounts for the more satisfactory work of the furnace with the turbo blower, as the volume and temperature of the air are constant regardless of the pressure, while with a reciprocating engine delivering a constant volume the weight decreases with increased pressure.

F. L. GRAMMER, Leesburg, Va. (communication to the Secretary*).—In Mr. Valentine's most interesting paper there is one item of carbon loss which he omits, as every one has done heretofore: viz., the amount blown out at the downcomer into the dust catcher. If he makes 25,000 lb. of dust a day and 10 per cent. is coke, there are 2,500 lb. of coke sent out of the furnace as dust, not as CO or CO₂ or in pig.

This is a small amount per ton, about 11 lb. per ton of metal, but would affect the final result a little.

It would affect it more in using unscreened coke from bells and using double bells, and where great volumes of wind are employed with high velocity of exit of gas.

S. G. VALENTINE (communication to the Secretary†).—In reference to the points raised in this discussion, I would say that a correction of

* Received Mar. 28, 1914.

† Received Apr. 15, 1914.

the calculation of air from the gas analyses to a temperature of 60° F., which was overlooked, gives as a result 48 cu. ft. of air instead of 45.5 as compared with the figures shown by the blower calibration of 41.7. This discrepancy of 6.3 cu. ft. may be due in part to the gas analyses thereby giving high results, or in part to any error of the calibration which might give low results. But even at 48 cu. ft., taking the highest figure, it appears that instead of the 131,821 cu. ft. of air required to burn all the carbon at the tuyères, only 103,056 cu. ft. of air were needed, indicating that 469 lb., or 21.8 per cent. of the 2,147 lb. of coke per ton pig, were consumed in the upper part of the furnace either by action of the ore or by the CO₂ of the gas. To my mind the economical coke consumption per ton of pig still finds its most likely explanation in the economical effect of direct reduction by carbon. The calculations of Dr. Richards referred to in the paper seemed to indicate that this is a fact.

The theoretical amount of air needed to burn a pound of carbon or of coke is easily figured. But in actual practice we know very well that we do not have to furnish this quantity for every pound of coke charged, for the simple reason that the coke does not all reach the hearth to be burned by the oxygen of the air. It is, therefore, self evident that any rating of air delivery which shows enough air blown to burn every pound of coke charged is an excessive rating. It seems, therefore, that in many cases less air is being actually used than the reciprocating-engine ratings would indicate. Mr. Nibecker states truly that sufficient corrections are not generally used in figuring on such air deliveries. I would add that some of the factors of these corrections are difficult, if not impossible, to accurately determine.

Relative to steam consumption by turbo blower, I cannot furnish any actual data. In our practice, operating with the condenser, the indications are that our steam consumption is not higher than it would be on a reciprocating engine of equal efficiency. Naturally, any blowing engine must have a sufficient steam supply to meet high furnace pressures. The turbo blower automatically speeds up at such times, thereby maintaining the supply of furnace gas under the boilers and therefore helping to maintain its steam requirements automatically. This is an advantage to the power equipment as well as to the furnace itself.

In regard to the pressure required for a given amount of iron with a turbo blower as compared to an engine, I do not think there is much difference to be noted. Pressure, being chiefly due to the resistance of the stock in the furnace, is probably less variable under a steady stream of air than it is under an intermittent or pulsating supply and might possibly average lower under the same furnace conditions. The fluctuations of pressure shown by a delicate gauge are less in the case of a turbo blower than they are with a reciprocating-engine air supply.

Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron

Discussion of the paper of ALBERT SAUVEUR, presented at the New York meeting February, 1914, and printed in *Bulletin* No. 86, February, 1914, pp. 254 to 257.

H. C. H. CARPENTER,* London, England.—I cannot pretend to be entirely uninterested in this paper but as beta iron has had plenty of discussion lately, I will be as brief as possible. The position really is this—that Professor Benedicks's theory that beta iron could be explained as a solution of gamma iron in alpha iron has given rise to a number of researches, and although it appears to me now that the theory, at any rate in the form in which it was put forth, is untenable, yet even so, I think that Professor Benedicks was quite justified in his original paper, because undoubtedly the researches have advanced to this stage.

It is very curious that there never has been any serious disagreement regarding the meaning of the A1 and A3 points. And although in this case widely different apparatus was used, and apparatus of different sensitivity, there has been on the whole very good agreement with regard to the results; but we cannot even to-day say that that is the case with regard to A2.

I desire to refer to a statement in Professor Sauveur's paper, which reads as follows:

"On the contrary, if, more rationally in my opinion, we believe that an abrupt heat evolution on cooling and a corresponding heat absorption on heating, when occurring in a pure element without change of state, must be taken as an evidence of allotropy, then A2 must be an allotropic point."

If we test the break on the curve at A2 by that rule, then we are in this difficulty—that Professor Burgess's curves which were presented at the last meeting show, as I pointed out in the discussion, a sharp break at 768° on heating—that is, at exactly the same temperature as on cooling—but the direction of the curve indicated that there was either an evolution of heat at that temperature, or else the cessation of the absorption of heat, both of which would have the same result on the form of curve. Undoubtedly, on cooling at Ar2 we also have an evolution of heat at 768°. You cannot show, in my opinion, that those things are, strictly speaking, reversible; for the whole criteria of allotropic change is that you have an evolution of heat on cooling, and if that is so, it must be an absorption of heat on heating, but, as I have just pointed out, it is not an absorption of heat on heating, at any one point, but an evolution or a loss or cessation of the absorption of heat at that point. But, as was agreed upon by various protagonists in the discussion at that meet-

* Non-member.

ing, there is undoubtedly an absorption of heat going on over a lower range of temperature, and in my opinion a study of the matter shows that fact.

If we look at Fig. 2, we see that the curve breaks away at about 680° , slopes to the right, and there is that movement at the left, which, I take it, corresponds to the absorption of heat, which extends from 680° to 765° , or, roughly speaking, from 80° to 90° . My own results show that absorption in the same way. Indeed, the contention in London at the May meeting was that my own curve showed what they said it did not—that is, an absorption of heat.

My position is really this, I am convinced, and I owe it to this meeting to say so, and I owe it to the subject as well—I am convinced that Professor Burgess has made good the discontinuity in the curve and has borne out Professor Sperry's results at about 768° . The interpretation of the curve is not yet clear. It does not appear to be exactly the reverse of the curve on cooling, because it is not an absorption of heat. We have in that connection to remember the recent work of the Japanese writer Honda, in which his conclusion, quite independent of these papers, is that the absorption of heat in this range of temperature is connected with magnetic change. However that may be, I am convinced they place the point of discontinuity of the curve at about 768° , and future research must show us what is the interpretation to be given to that result.

HENRY M. HOWE, New York, N. Y.—This beta iron Osmond used to call *Decipium*, just when you thought you had found it, it was not there. The case is somewhat comparable to that of the planets Neptune or Saturn, if either of them were not visible. Suppose Neptune were invisible and could not be seen, to prove its presence would be extremely difficult, but its presence would be susceptible of proof from the perturbation of the other planets. In the same way, this beta iron has not been isolated—it may not have been isolated, and it may not be possible to isolate it, but we accept the evidence of its presence from the perturbations which it causes in all sorts of phenomena. The two strong perturbations difficult to understand on any theory other than the existence of beta iron, are the excessive hardness, and the transition form between gamma iron and alpha iron. No matter how you arrive at that transition form, you have arrived at that transition form by cooling rapidly, so as to catch the metal in transit between gamma and alpha, and you get extreme hardness. That is the way we get our hard cutting tools, and we get the extreme hardness of hardened steel. If, on the other hand, you bring about that intermediate state of an intermediate degree of carburization—if you take soft nickel steel and carburize it part way toward the gamma state, you get it intensely hard, and if you carburize it further and get it over into the gamma state, it becomes soft again;

that is to say, whatever way you bring it about, if you catch the material between gamma and alpha, it is much harder than either gamma or alpha, and it is so much harder that it is difficult to explain it. It is not impossible, but it is difficult to explain it, and the explanation seems to be that that extra hardness is simply due to the solution of gamma in alpha.

It is difficult to understand why the solution of gamma in alpha should be so much harder than either gamma or alpha—gamma iron itself is a solution of cementite in gamma iron as we know it—when we dissolve further alpha iron in that solution, according to Professor Benedicks—and the reason why these intermediate states are harder than gamma iron is that alpha iron is dissolved in what was a solution; that is to say, starting in with a solution of cementite in austenite, and dissolving into that solution alpha iron, we must, to explain that theory, dissolve alpha iron in that solution and we must increase the hardness of the solution—I think it is easier to believe and more natural to believe, and after all we take theories which are easier and more natural—more harmonious with the general aspect of nature—it is easier to believe that that extra hardness is due to the presence of this perturbed condition in beta iron, than to suppose it is simply due to the solution of alpha iron into a softer material, into a solution which already existed.

GEORGE K. BURGESS, Washington, D. C. (communication to the Secretary*).—I am glad to see that Professor Sauveur has thought it worth while to examine with a recording apparatus the heating and cooling curves of pure iron and to note that, using a furnace supplied with *direct* current, he substantially confirms the results obtained by Mr. Crowe and myself using *alternating* current heating and also *gas* heating.¹

We have since had the pleasure of demonstrating to Professor Carpenter, on the occasion of his recent visit to the Bureau of Standards, with a 0.4-g. sample of his electrolytic iron, the independent existence and location of A2 and A3 both on heating and cooling.

It appears, therefore, to be abundantly proved to the satisfaction of all who have recently partaken in the experimental work on the subject, that, as Professor Sauveur states, "The existence of A2 in pure iron as an independent point is no longer a debatable question."

PROFESSOR HENRY LE CHATELIER, Paris, France (communication to the Secretary†).—I beg to offer the following observations on the paper of Professor Sauveur:

1. The suggestion of Professor Benedicks, that the electric heating

* Received Mar. 23, 1914.

¹ *Bulletin* No. 82, Oct., 1913, p. 2537, and No. 84, Dec., 1913, p. 2870.

† Received Mar. 26, 1914.

of furnaces may play a certain part in the development of the point A2, deserves experimental test. It certainly does not permit an absolute denial of the existence of the thermal phenomenon, for this is equally proved in cooling, during which there is no electric current at all. It is, however, perfectly logical to admit that the disappearance of energy due to the loss of magnetism in an electric field may liberate a certain quantity of heat. It would be necessary to compare the cooling of the same sample inclosed in a tube cooled on the outside by circulating water, and surrounded by a solenoid permitting or preventing, at the will of the operator, the passage of the current, which would then produce no thermal effect upon the sample, by reason of the intervening envelope of water.

2. I feel it a duty to repeat here, what I have often said already, that the point A2 cannot be regarded as marking anything like a change of allotropic state. The change at A2, which exists in all ferro-magnetic bodies, differs from allotropic changes in being continuous, exactly like the progressive change of the properties of molten sulphur. It is demonstrated in a certain way for nickel, by the measure of dilatation. Moreover, the change is almost rigorously reversible; there is no notable retardation in the transformation, ascending or descending, while the transformation A3 and all the allotropic transformations suffer retardations which are sometimes enormous.

SIR ROBERT HADFIELD, London, England (communication to the Secretary*).—I have read with much interest Professor Sauveur's paper, Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron, and while regretting not being able to support the conclusions presented from the allotropist's point of view, yet I trust the following statements may be of service in helping to elucidate the problems referred to. The reasonable and moderate expressions of opinion so ably presented by Professor Sauveur show that he is willing to try to look fairly at both sides of this important subject.

Although the paper is only a brief one, yet the curves shown in it are of considerable interest, and raise what may be termed momentous questions.

Although mention is not made of the exact composition of the electrolytic iron experimented upon—for example, was it free from hydrogen, as no doubt it was, and assuming that it was—we have then to consider that the material in question, representing pure iron, shows a change point at that part of the heating or cooling curve known as the A2 point, which Professor Sauveur considers, as I understand, represents an allotropic change in the iron.

On the other hand, Dr. G. K. Burgess and J. J. Crowe, in the last

* Received Mar. 27, 1914.

paragraph of their reply to the discussion on their paper, The Critical Ranges A2 and A3 of Pure Iron, read before this Institute in October, 1913, state that if the A2 point could be proved to be due entirely to the magnetic change point, then it would appear that beta iron could not exist.

If the recent work by Prof. K. Honda on Transformations of Iron and Steel at High Temperatures, which appeared in the *Science Reports of the Tokyo Imperial University*, is correctly understood, this very demonstration seems to have been achieved. Professor Honda clearly states in the research just mentioned that the A2 point is due to disengagement of heat associated with the magnetic change. His words are as follows:

"With a refined method of measurement it was found that in iron, nickel and cobalt at the magnetic transformation, a distinct heat evolution took place. The heat evolution extended over a wide interval of temperature. With iron, steel, and nickel, the transformation is accomplished with a low intensity of field in a small interval of temperature, the interval increasing with rising field intensity. The author [Professor Honda] therefore arrives at the conclusion that in pure iron only two states, alpha and gamma, occur. The magnetic transformation hitherto termed alpha to beta is not a true phase-transformation, but a change of property as the result of temperature. Nickel and cobalt have only a single state in the solid."

Does not this, therefore, rather answer Professor Sauveur's very fair-minded comments, when he says:

"If it be insisted that an allotropic point necessarily implies a crystallographic change, or, in other words, that polymorphy and allotropy are synonymous, then it may be claimed, with reason, that A2 is not an allotropic point, since crystallographic differences between beta and alpha iron have never been discovered?"

It is true he adds that:

"On the contrary, if, more rationally in my [Professor Sauveur's] opinion, we believe that an abrupt heat evolution on cooling and a corresponding heat absorption on heating, when occurring in a pure element without change of state, must be taken as an evidence of allotropy, then A2 must be an allotropic point. Are not such thermal disturbances due to spontaneous changes of internal energy necessarily caused by molecular changes; i.e., in the absence of change of state, by allotropic changes?"

It seems to me that Professor Honda has answered Professor Sauveur's question. Therefore, it appears to me most positively proved that the A2 change is not a definite evidence of an allotropic point.

It must not be forgotten that those who have been unable to accept the allotropic theory have had to question and contest the following statement:

"I conclude that hardened steel owes its properties principally to the presence of beta iron, which is hard and brittle by itself at the ordinary temperature."

The above is not a statement by one of no importance. It is a clear and definitely presented statement by one of our greatest masters of metallurgy, the late M. Osmond, whom we all respected. While allotropists have made certain changes in the theoretical presentation of their case, yet, as it is generally understood, those representing this line of thought still claim the above.

Probably some of this trouble has arisen for the reasons pointed out by Professors Edwards and Carpenter in their interesting paper on The Hardening of Metals, being read before the Iron and Steel Institute at the May meeting this year, namely, that "At a later period it was considered by Osmond, Roberts-Austen, and others who took up the views of the so-called allotropists, that the Ar2 point was quite distinct from the Ar3 point, and they came to regard the Ar2 point as corresponding to an allotropic inversion of the iron." Then, as these two gentlemen point out, "From that time onward, the beta theory as it is applied to the hardening of steels really became an entirely different theory, but unfortunately the same nomenclature was retained."

If allotropists would only define more clearly what is really now meant, it would be very helpful. There are, however, many different points of view among the allotropists. I do not mean to say for one moment that the carbonists can offer a clear solution of the problem, but the latter seem to me to come more nearly to the facts and phenomena noticed in the hardening of steel.

Incidentally, the following may be of interest: It may not now be remembered that it was probably owing to certain views expressed in my first paper on Manganese Steel, read in 1888,² that an allotropic theory exists to-day. In my original paper referred to, I made the following statement, under the section headed "Influence of Carbon upon Manganese and Iron":

"If manganese were the sole cause of the hardness noticed, it would naturally be supposed that the latter would be much harder; this however is not so, and the apparent anomaly occurs that when the percentage of iron (Fe), known to be a soft metal, diminishes, and is replaced by the supposed harder metal manganese (Mn), the hardness of the alloy decreases. This seems to prove that manganese must not be considered as the sole influence or cause of hardness. Even in those specimens containing as high as 20 per cent. Mn, and in which further modifications may occur to complicate the question owing to the carbon reaching higher amounts, 1.5 to 2 per cent., the hardness is less than that of the 4 per cent. Mn steel in its cast state.

"That iron (Fe) itself is a soft metal is well known, but it is worthy of consideration, at any rate, whether it possesses the power of becoming intensely hard, either by some peculiar form of crystallization, as discussed further on, or owing to its forming certain combinations or alloys. In other words, whereas the hardness of steel has hitherto been entirely attributed to the bodies present in much smaller proportions as compared with the iron, may it not be possible that the part played by iron (Fe) is

² *Proceedings of the Institution of Civil Engineers*, vol. xciii, pp. 72, 73 (1887-88).

equally, if not more, important, perhaps owing to some change of form of the iron itself not hitherto suspected, as in the case of the intensely hard 4 per cent. manganese-steel, where carbon cannot possibly be the cause, and where manganese would not appear to be the sole cause? . . . It consequently becomes more important to ascertain whether the manganese itself causes the hardness in this steel, or is only a means whereby iron is caused to take this nature."

In the paper entitled *Iron Alloys with Special Reference to Manganese Steel*,¹ it will be noticed that I had already taken up a new modified line of thought as compared with that suggested in 1887-88.

Now why was it I abandoned the theories advanced, it should be remembered, so far back as 1888? My reply is that in the first place I found that some of the beliefs then expressed were founded on data which at the time appeared to be correct. I afterward found, however, that as regards the determination of the carbon percentages actually present in one or two of the specimens upon which I based certain conclusions, the analyst had underestimated the percentage present. That is to say, the carbon percentages were, by more correct modern analytical methods used for determining carbon in alloy steels, found to be considerably higher, that is relatively to the smaller amounts present, 0.4 to 0.5 per cent., instead of about 0.3 to 0.4 per cent., as then estimated. The increasing and rapid influence of carbon over 0.35 per cent. is well known. Consequently, with corrected figures before me, some of the original remarks did not hold good, and the disregard of the influence of certain somewhat higher percentages of carbon had been too much.

Singular to say, Sir W. C. Roberts-Austen, in contributing to the discussion on my paper in 1888, said "he thought that the author had a little under-rated the influence of the small amount of carbon which the manganese iron contained." In other words, this scientist, although afterward a strong allotropist, mentioned the very point I and others have so strongly emphasized: namely, that we have arrived at the conclusion that it is rather to the influence of the carbon present than to any allotropic change in the form of the iron present, that we must look for the correct explanation of the many peculiarities met with in steel and its alloys. As time passes, I am still more convinced of the correctness of our attitude.

It should not be overlooked that your distinguished metallurgists, Mr. Metcalf and Professor Langley, in 1893 read a paper before the American Society of Civil Engineers, vol. xxvii, p. 382, in which they gave a brief statement of the carbon theory of hardening, to which paper the reader is referred for fuller information. Briefly, their statement was that carbon alters the molecular aggregation, and the resulting strained grouping is brought about within certain temperature limits;

¹ *Trans.*, xxiii, 148 (1893).

also molecular stresses are set up, which produce hardness in a similar manner to that of cold working.

This view has been further elaborated in a most important paper by A. McCance, B. Sc., A. R. S. M., entitled *A Contribution to the Theory of Hardening*, to be read before the Iron and Steel Institute at the May, 1914, meeting. The writer considers this one of the most important contributions to the metallurgical world on this subject. It is therefore doubly of interest to find, to use the words of the late respected Mr. Metcalf and Professor Langley, that hardening is the result of molecular stresses, which produce hardness in a similar manner to that of cold working. In this paper by Mr. McCance it is stated:

"Based on the experimental work of Curie, the electronic theory of magnetism was first put on a sound basis by Langevin, and has since owed most of its development to Weiss. Every atom is supposed to have a definite field of its own, caused by its internal system of revolving electrons, and under the action of an external field the molecules (or atomic aggregates) tend to set themselves all in one direction, giving rise to a definite intensity of magnetism which is the total effect of all the molecular magnets in the substance added together. In gases and liquids they are prevented from all turning in the one direction by forces and collisions depending on the translational velocity, and in solids other forces which may be called 'crystalline' forces come into play. Instead of preventing the molecules from turning round under the action of an external field, these crystalline forces may actually assist them to turn, and it has been conjectured that this is the special action which goes on in the so-called 'ferro-magnetic' bodies, which are bodies like iron that become strongly magnetic under weak external fields. As on heating a liquid, when a temperature is reached at which the forces between the molecules become insufficient, the liquid becomes a vapor, the forces causing ferro-magnetism gradually become inoperative on heating until a temperature is reached at which they disappear rapidly and the body becomes non-magnetic. . . .

"In iron, then, the property of loss of magnetism cannot be used as a proof of allotropy—it is no sudden transition, but an effect depending only on temperature. .

"Accompanying every alteration in the magnetic intensity, there is a variation in the specific heat, . . . proportional to the rate of change of the square of magnetic intensity at any temperature, and it has a maximum value always at a temperature slightly below the magnetic transition temperature. . . . It begins gradually and rises to a maximum at about 760° C. for pure iron, becoming zero as the iron becomes non-magnetic; and the important point is that this change in the specific heat cannot be distinguished in a cooling curve from a true absorption of heat due to an allotropic change. In the author's opinion this confusion has actually taken place at the Ac₂ point in iron. The variation is quite large enough to account for all the thermal change, and the gradual character of the Ac₂ point is entirely in accordance with this nature.

" . . . The temperature at which the non-magnetic condition commences has no connection with the true allotropic ranges, and takes place quite independently of them—thus pure iron becomes paramagnetic at 820° C. (Curie), which is 100° below the Ac₃.

"There is this great difference between a pseudo-change point caused by a sudden change in the specific heat and a true liberation of heat caused by allotropy. When the temperature is rising very slowly, the effect due to specific heat is masked in a differential heating curve, since the difference measured by the differential galva-

nometer is proportional to the product of the rate of heating and the specific heat. With a true thermal change point, however, the amount of heat liberated is independent of the rate of heating.

"The sharpness of the A2 point is much less on heating than on cooling, . . . as the change is more gradual when approached from the low temperature side than from the high temperature side.

"The failure of Carpenter to notice the Ac2 point in pure iron, must be ascribed to the nature of the change being as above described. . . .

"The problem of beta iron is thus reduced to a very simple one, and without making any assumptions regarding the nature of alpha or gamma iron, while taken with the evidence produced by other experimenters, the conclusion is undeniable that beta iron as a separate allotropic modification has no existence, since it is merely alpha iron which has lost the properties of ferro-magnetism."

Professor Weiss believes that by the electrical disposition of what he theoretically terms "magnetons" he is able to offer more satisfactory explanations of magnetic phenomena than by any other theories. He also appears to consider that the increase of magnetons, with their important bearing upon magnetic properties, is independent of allotropic and perhaps even chemical modifications, of the metal under examination. I have dealt with this more fully in the joint paper by Professor Hopkinson, F. R. S., and myself, to this Institute.⁴

CARL BENEDICKS,* Stockholm, Sweden (communication to the Secretary†).—I quite agree with the author in laying stress on "a substantial foundation of experimental facts." However, I have the impression that this foundation is not very much consolidated by repeating, apparently with less accuracy, or at least with no greater accuracy, than by previous workers, exactly the same style of experiments—however interesting in themselves. From Professor Carpenter's and from Dr. G. K. Burgess and J. J. Crowe's important work, we know that occluded gases have the effect of rendering the thermal changes in the A2 region much more considerable than they are when the gases are removed. Professor Sauveur's curves show an evolution of heat, or absorption, at A2, which in comparison with that at A3 is much more considerable than in Professor Carpenter's or in Messrs. Burgess and Crowe's curves, and we find no indications how, or even if, the gas content, so important for the thermal irregularity at A2, was lowered.

As for increasing the experimental foundation of our allotropic views, the writer considers it more appropriate to repeat earlier experiments with an accuracy much higher than that formerly used. This is the reason why the writer, likewise on iron kindly supplied by Professor Carpenter, lately has repeated the dilatation work of Svedelius, Le Chatelier, Charpy and Grenet (and Broniewski), using a compensation method which, owing to special arrangements, gave a sensibility about

⁴ *Bulletin* No. 87, March, 1914, pp. 513 to 530.

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† Received Apr. 2, 1914.

1,000 times greater than that used at the work of the last-named authors. Simultaneously, on the same material, magnetometric and magnetostrictional measurements were carried out. The result, which might interest Professor Sauveur, is that there is *not the slightest trace* of discontinuity at A2. If there is a discontinuity, it is quite surely less than three-thousandths of the discontinuity at A3. Even the continuous length decrease which was found to occur together with the disparition of the ferro-magnetism, and which is a necessary effect of magnetostriction, is a rather weak one.

Until some still more fundamental work is published than J. Willard Gibbs's papers on the Equilibrium of Heterogeneous Substances, we must consider this phase rule as a reliable standard. In order to prove that an *allotropic point* occurs at a definite temperature, according to Gibbs's theory, it is a necessary condition to prove that *two phases occur* at that temperature. This is done by proving that a real discontinuity occurs in the properties of the substance at a definite temperature, *at least in a single case*. It is not in harmony with Gibbs, and too narrow, to select as a criterion the property of crystallization, as was proposed by Messrs. Burgess and Crowe in their important paper. Any property is to be accepted as proof of allotropy point, provided it gives a clear discontinuity; that is to say, two different values of the property at one and the same temperature. Now, no such discontinuity, in the strict sense of the word, has ever been proved to occur at A2, and since the dilatation experiments, denoted by Messrs. Burgess and Crowe as so important as to have a crucial bearing, have been found not to give the slightest discontinuity, we may be very sure in concluding that *A2 is not an allotropic point*.

I have some slight hope that Professor Sauveur will judge of the writer's theory somewhat more favorably than as yet, after reading the paper which the writer has prepared for the Iron and Steel Institute's spring meeting. Therein are also to be found some reliable measurements on ferro-magnetic mixtures (magnetite, iron, nickel) giving an experimental support to the writer's attempt⁵ to explain the fact that the disparition of ferro-magnetism occurs at a much lower temperature than the allotropic transformation point, A3.

N. BELAIEV,* St. Petersburg, Russia (communication to the Secretary†).—The few remarks I have to add to the most valuable and instructive paper of Professor A. Sauveur are in relation to the crystallization phenomena in iron-carbon alloys. Let us consider the so-called Widmanstätten structure with its characteristic sharp, straight lines; the first look at such a specimen leads us to the idea that a *sudden pre-*

⁵ *Journal of the Iron and Steel Institute*, vol. lxxxvii (1913, I), p. 334.

* Non-member.

† Received Apr. 11, 1914.

cipitation of a proeutectoid element (ferrite or cementite) took place; but if the crystallization occurred suddenly the allotropic change must also be a sudden one and not a gradual one, covering a wide range of temperature, as Professor Benedicks argued.

On the other hand, the same crystallization phenomena show me only two distinct crystallographic bodies—the gamma iron and the alpha iron. The first (Fey) is related to the structure of austenite—the “granula” or austenite grains; the second, with the pearlite grains. The network of the proeutectoid element shows us the first; the pearlite lamellæ, the second.

To express myself quite clearly, I say that only between gamma and alpha iron is there a sharp crystallographic difference to be detected; in other words, gamma and alpha iron are polymorphous bodies. But in my opinion different allotropic modifications do not necessarily imply a crystallographic change. From this point of view, not the crystallography of the studied modifications, but the change of internal energy, must be considered as the strongest proof of allotropy. As to this change, it is made clear principally by the heat evolution or absorption and we have to study these on our cooling curves.

The author’s researches, as well as those of Dr. G. K. Burgess and J. J. Crowe, show us quite unmistakably that in pure iron A2 exists, always as an *independent point*.

This conclusion is very important and the reported curves are highly instructive, and I hope that everybody “before abandoning Osmond’s theory, so satisfying because so simple and rational,” will seriously and attentively study the paper of Professor Sauveur.

ALBERT SAUVEUR (communication to the Secretary*).—I am appreciative of the interesting and important discussion of my paper contributed by so many distinguished authorities. It is a great pleasure to find myself in substantial agreement with Professors Carpenter and Howe with Captain Belaiew and Dr. Burgess. Professor Carpenter, however, is not yet convinced that the A2 point marks a reversible transformation, and, on that ground, he would question its allotropic character. I find it difficult to follow his contention that the thermal disturbances at A2 indicate an evolution of heat, both on heating and cooling. This seems quite inconceivable to me, for, thermodynamically speaking, such a phenomenon would be in the nature of perpetual motion.

Certainly Professor Howe’s statement that it is easier and more natural to believe that the extreme hardness produced by preventing the complete transformation of iron-carbon alloys is due to the retention of a transition form of iron between the gamma and the alpha state rather than to ascribe it to a solution of alpha iron in gamma iron, or rather in austenite

* Received Apr. 13, 1914.

(itself a solution of the carbide Fe_3C in iron), cannot be disputed. We should resist unnecessary complications in scientific theories. As Professor Howe well puts it, the theories which should most appeal to us are those which are more "harmonious with the general aspect of nature."

Professor Benedicks contends that the cooling curves of Professor Carpenter's electrolytic iron, reproduced in my paper, add no information of value to the present controversy, because they were determined with less accuracy, or, at least, with no greater accuracy, than similar curves taken by Dr. Burgess. While I am quite willing to grant his claim, I had hoped that, in view of the fact that a direct current was used for heating the iron, my curves would have been considered as having some significance, owing to Professor Benedicks's claim that in Burgess's experiments the occurrence of the A2 point might have been caused by his having used an alternating current. I might also add that in my experiment the iron was repeatedly heated previous to the taking of the thermal curves for the purpose of expelling the occluded gases.

Professor Benedicks's statement that he was unable to detect the slightest trace of dilatation at Ar₂ in Professor Carpenter's iron, even when using a method of extreme accuracy, is very important and will undoubtedly receive the attention it deserves. But does it prove the non-allotropic character of A2? While dilatation taking place abruptly during the cooling of an element may prove the existence of allotropy, does the absence of such dilatation prove its non-existence? An allotropic transformation does not, I believe, imply discontinuity in *all* of the properties of an element. If discontinuity is detected in some of its properties, is it not sufficient to prove its allotropy? Of course Professor Benedicks claims that "no such discontinuity, in the strict sense of the word, has ever been found to occur at A2," and, on that basis, he rejects the allotropic theory of the A2 point. Many investigators believe, however, that discontinuity in several of the properties of iron has been shown to occur at A2.

The absence of dilatation at Ar₂, which Professor Benedicks uses as a weapon with which to deal a blow, which he considers final, to beta iron, might easily be turned against his own theory with apparently telling effect. The reasoning is as follows: (1) the point A3 marks an allotropic transformation; (2) the Ar₃ transformation is accompanied by a marked dilatation; (3) Professor Benedicks believes that the A2 point is part of the A3 transformation; (4) the Ar₂ point should therefore be accompanied by dilatation; (5) Professor Benedicks finds complete absence of dilatation at the Ar₂ point; (6) *ergo*, the A2 point cannot be part of the A3 transformation.

Professor Le Chatelier doubts the allotropic character of the A2 point for three reasons: (1) because a similar point is detected in all ferromagnetic bodies; (2) because the A2 transformation is continuous; and

(3) because the A2 transformation is "almost rigorously reversible." Taking these objections in order:

(1) The fact that the thermal point exists in all ferro-magnetic bodies, corresponding to the final disappearance of magnetism on heating, does not in my opinion argue against its allotropic character. It indicates a body capable of existing under two sharply distinct conditions, *i.e.*, a magnetic and a non-magnetic one, and implies the existence of a dividing line between the two. Is that not allotropic? Or will it be contended that the magnetic and non-magnetic conditions of an element represent one and the same allotropic condition? It is evident, as I have so often said, that the point in dispute must be settled one way or the other according to our definition of allotropy, and as to this, unfortunately, scientists entertain widely different views.

(2) Does the fact that the A2 transformation is continuous bar it from the allotropic class? To begin with, is it not somewhat misleading to describe it as a continuous transformation and *nothing more*, seeing that the end-point of that transformation is so intense and of such magnitude as to produce a distinct and spontaneous evolution of heat, indicative of a notable change of internal energy?

(3) As Professor Le Chatelier states, and as so conclusively shown by Burgess and Crowe, the A2 transformation appears to be rigorously reversible; that is, the Ar2 and Ac2 changes occur at exactly the same temperature. There is here a complete absence of hysteresis, whereas the A3 transformation and allotropic transformations in general, so far as they have been studied, always exhibit a temperature gap between the two opposite phases. In my opinion, however, this absence of hysteresis can hardly be considered as a proof of the non-allotropic nature of the A2 point. In elements, and in many other substances, the passage from the liquid to the solid state and the reverse change (solid to liquid) generally occur at exactly the same temperature, and may not these changes of state be considered as major cases of allotropy? Until we have a more intimate knowledge of the nature of the forces at work producing hysteresis, we can hardly claim that when those forces are absent or only feebly active there can be no allotropic transformation.

Replying to Sir Robert Hadfield's valuable criticism of my paper, I should say that the electrolytic iron used in my experiments was kindly supplied to me by Professor Carpenter and that it was similar to the iron used by himself in his investigation described in his paper on The Critical Ranges of Pure Iron with Special References to the A2 Inversions.*

The iron was repeatedly heated and cooled before taking the critical curves and, in my opinion, must have been practically free from hydrogen.

Referring to Sir Robert's quotation of a statement of Burgess and

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Crowe to the effect that "if the A2 point could be proved to be due entirely to the magnetic change, then it would appear that beta iron could not exist," it should be borne in mind that this conclusion of the authors is warranted only by their definition of allotropy, which they advanced in a tentative way as "an allotropic transformation is one accompanied by crystallographic change," a view which is by no means generally held by scientists, as will be apparent from Professor Howe's and Captain Belaiew's contributions to the present discussion.

Granting the accuracy of Professor Honda's work and conclusions, the allotropic or non-allotropic character of the A2 point still remains a question of definition. Professor Honda recognizes the existence of iron under three conditions, *i.e.* (1) gamma iron, non-magnetic and stable above the A3 point; (2) alpha iron, magnetic and stable below the A2 point; (3) non-magnetic alpha iron, stable between the A3 and the A2 points; and why should we cease to call this last variety beta iron? In which way has Professor Honda demonstrated the non-existence of beta iron?

Osmond's statement quoted by Sir Robert and reading as follows, "I conclude that hardened steel owes its properties principally to the presence of beta iron, which is hard and brittle at the ordinary temperature," still stands.

Sir Robert alludes to two extremely important papers recently presented before the Iron and Steel Institute in London, one of them by Professors Edwards and Carpenter, and the other by Mr. McCance. These contributions are the most important ones that have been offered for many years and I hope to discuss them at some length. It will suffice for the present to call attention to the fact that these two splendid arguments do not upset the allotropic theory of the hardening of steel. On the contrary, as I have said, they reinforce that theory. The basic claim of the allotropic theory is the existence of a condition between the gamma state and the alpha state and which when retained in the cold imparts great hardness to the steel. In the two papers referred to it is made evident that such a condition must exist and must be retained in order to explain the hardening of steel. It matters not, so far as the allotropic theory is concerned, what that condition really is; for convenience, it is called beta iron. The fact that it may be, according to Mr. McCance, alpha iron which has lost its ferro-magnetism, or interstrain alpha iron, or, according to Edwards and Carpenter, amorphous alpha iron, does not by any means eliminate the allotropic theory. It merely gives us more insight in regard to the possible nature of beta iron. It adds to our knowledge of beta iron and, therefore, rather strengthens its existence.

Captain Belaiew's very clear statement in regard to his conception of allotropy is very satisfactory indeed. It appears to me a more rational one than the view which considers a crystallographic change as a criterion by which to judge allotropy.

The Need of Uniform Methods of Sampling Lake Superior Iron Ore

Discussion of the paper of C. B. MURRAY, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 85, January, 1914, pp. 135 to 141.

WILLIAM KELLY, Vulcan, Mich.—I would like to ask the author a question. A good deal of the Lake Superior ore is made up of mixtures from various mines, of which the product differs considerably. I know of one case where ore running about 52 to 53 per cent. in iron, with considerable lime and magnesia, a self-fluxing ore, is mixed with ore running 61 per cent. in iron, in which there is very little lime and magnesia.

Now, in this method of sampling in the vessels I would like to know whether a correct average result can be obtained if all the low ore is in the bottom and the high ore on top, or the reverse. Do you not get in this method of sampling more of the top ore than of the bottom ore? I want to disclaim, of course, any intimation that any one in the Lake Superior region would attempt to put the high ore on top in order to get a sample that would run above the average. That would not only be dishonest, but in the long run of course it would not be to the advantage of the shipper. Still, it has been suggested that in this method of sampling there is more of the top ore obtained than of the middle and bottom of the cargo, and I would like to ask the author whether such is the case.

C. B. MURRAY—Mr. Kelly brings up a question which is often asked by those not familiar with ore sampling at lower Lake ports. Ores are mixed, more or less, in cargoes, and unless the various grades are represented in the sample the complete sample is not representative of the cargo.

The standard method of sampling is particularly adapted to such cases, and when properly carried out takes the right proportion from the top, middle, and bottom parts of the cargo. The ore, as it lies in the boat, approaches a cone shape under each hatch. The sampler starts sampling in each hatch when 5 or 6 ft. of the ore face is exposed. As the ore is worked out, and fresh surfaces and larger areas are exposed, the sampler continues work, taking equal volumes of ore at regular, measured intervals.

WILLIAM KELLY.—The thought that was in my mind was, that as you take out the ore from a pile, the top ore falls down all the time.

J. W. RICHARDS, So. Bethlehem, Pa.—Then that would be counteracted, as I conceive it, Mr. Murray, by the fact that the cone being made larger, you take more of the lower layers as the cone increases in size.

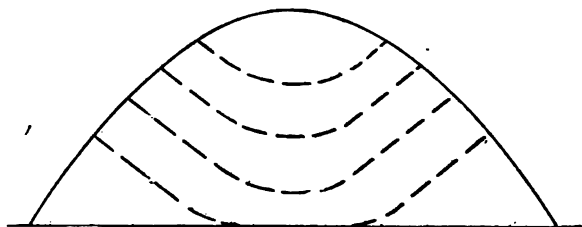
C. B. MURRAY.—Yes, that is the idea. As the areas exposed become larger, more places are touched and more ore is taken, but the area represented by each sample remains the same. Some of the top ore may

slide down the exposed surface when the grabs begin working, but as the area to be sampled increases in size, there is less tendency for this ore to slide down.

This can, perhaps, be best shown by a cross-sectional sketch of a hatch (Fig. 1).

RALPH H. SWEETSER, Easton, Pa.—This paper by Mr. Murray has called attention to a detail of the iron business that is very important, not only from the commercial side, but also from the metallurgical side. The blast-furnace man who makes iron for the market, or who makes it for a department of a steel plant, that demands the top limit, for instance, on manganese, has been bothered a great deal lately by trying to make the manganese in his iron check up with the reported manganese in the iron ore.

We had trouble like that a couple of years ago, and we made investigation of the ores that we were receiving. I think one of the biggest changes in blast-furnace practice in the last several years has been in the character of Lake ores shipped as Mesaba ores. Blast-furnace men have the greatest trouble in dealing with soft, fine Mesaba ores.



Broken lines designate surfaces at various stages of unloading.

FIG. 1.—CROSS-SECTION OF HATCH.

Now, the ores put on the market at the present time do not have these characteristics as much as they used to have, and the trouble which the furnace men have now is to really work one ore—that is, an ore which is under one name, but is really a mixture of two ores. The Mesabas will have, most of them, a very fine ore mixed with a great many lumps. Now, when these lumps and fines are loaded over the side of the vessel, there will be a segregation of the fines and lumps, and the samplers can tell on what side the ore is loaded, as the lumps will go to the far side and the fines stay on the other side.

In one particular case we had a shipment of ore, in which the heavy ore was away over on the right-hand corner of the boat, showing, apparently, that the ore was loaded from the right-hand corner of the vessel. This ore was mixed, fines and lumps, in the proportion, I think, of about 15 to 20 per cent. of lumps, and the rest were fines. In this particular ore, the lumps carried a great deal more manganese than the

fines did. We could not make the results in the furnace check up with the analysis of the ores. That was the case in one hatch of a 30-hatch boat. I think the hatch was No. 21, and the total depth of that ore was 16.5 ft. The boat was about 42 ft. wide inside.

As Mr. Murray has explained, as they begin to unload the boat, the ore sampler will take a sample near the top part. The sampler will make his rounds, and it used to be the practice to make a round when they were one-half way down in the cargo, and then another sampling when they had gone three-quarters way down into the cargo.

This slide (referring to slide on screen, not here shown) would not give a fair sample. There is the ore, as it appeared in hatch No. 7. That is 5 ft. 3 in. deep in the center, which is just about one-third of the total height of the cone of ore which was in that hatch. The lumps are about one-third the way across. If you take a cone of ore like that, nearly one-half of the total weight of the ore would be in one-third of the height, and in order to get proper samples you would have to take as many samples in the lower one-third as in the upper two-thirds. If the character of the ore in the lower one-third is different from that in the upper two-thirds, you do not get a fair sample of the ore. In sampling the ore, as found in a boat, it is the rule to take a scoopful from so many scoopfuls. If they came to a lump, it has been a rule in the past to take a cubic inch of lump—the scoop will hold 9.9 cu. ft. of content—and if you take only a cubic inch of lump you are not giving the lump the same attention as the fines, and in the case of most Lake ores the fines are richer, and therefore you will get a higher result in the sample for that cargo.

I have been surprised to find on the Atlantic seaboard in some of the ores coming in that the fines are fewer than the lumps, and that if you do not get the right proportion in the sampling you get a lower result than the ore actually contains.

I think that the lumps should be sampled closely to the sides of the vessel. There are two or three things that prevent this from being done. Those who have seen the sampling of the boats know that just as soon as they clamp tight a man will start cleaning the side decks, and will sweep the dust over on to the ore. It is the rule of the samplers that they will not take samples after the dust has been swept on to the ore—it may be dust or it may be coal. Therefore, as a rule, the lower edge of the piles will not be sampled, and you do not get a true sample. This is because you do not get the lumps. We found in certain lumps of one ore that there were different minerals from the general run of the ore. In one case we found lumps which would contain 21 per cent. of iron, 21 per cent. of silicate, and 21 per cent. of manganese, and we would question why we had such varying results in pig iron made from that ore. If we ran to lumps, the manganese went up. If we ran to fines,

the manganese would run down. Whatever is decided upon as a practical method of sampling cargoes at the Lake ports would apply also to the sampling of cargoes at the Atlantic ports, because there is the same condition of loading obtaining in both places.

C. B. MURRAY.—The question Mr. Sweetser has brought up, about getting the proper proportions of lump and fine, is a very vital one. The lump and fine in many ores vary widely; in some the lump is richer, and in others is poorer than the fine ore. Here again the condition is ideal for using the standard method of sampling, and if the method is carried out according to instructions the sample will be representative even under the conditions which Mr. Sweetser illustrates.

It is generally the case that when samples are not representative the method has not been correctly carried out. In the case of the sampling referred to by Mr. Sweetser, the area exposed during the last stage of sampling was so much larger than that exposed during the previous stages that the sampler would, undoubtedly, get fully as much ore as he had previously accumulated in his sample from that hatch, and if he properly measured his intervals he would take a proper proportion of the lump in the wings of the boat.

Mr. Sweetser is mistaken regarding the sampler's instructions as to taking lump ore. In sampling a cargo of all hard ore, the instructions are, "At each point sampled, take lump or fine ore equal to 1 cu. in."; but in sampling soft ore, which often contains lump, such a cargo as I understand Mr. Sweetser refers to, the instructions are, "When a lump is encountered, a portion shall be broken off equal in bulk to a scoopful of soft ore."

If, in this particular cargo, the sampler did not do this, then a correct sample was not obtained, but it was not because of the fact that the standard method was not correct, but because it was not carried out.

A. J. ENNIS,* Philadelphia, Pa. (communication to the Secretary†).—From my experience in the Eastern section of this country, where there are official iron-ore samplers entirely competent to perform judgment work, perhaps my appreciation of the Western dilemma, where the West doesn't pay sufficient wages to attract the services of judgment samplers and hence must reduce the operation to mechanical labor without judgment, is not sympathetic. Neither do I fall into line with Mr. Murray's sampling suggestions for the manikin sampler. For example, sampling off the long sides of the triangle cannot do the material justice. Samples should be taken from all sides. To limit sampling to the long side is to become "a one-piece-of-rock man," as it were, filling the scoop with rock where there is but one piece in sight. I agree with Mr. Murray's statement to the effect that no mathematical formula can be

* Non-member.

† Received Jan. 17, 1914.

applied to the sampling of iron ore, whether from stock piles, cars, or ship's holds. The principle of ore sampling is, in its entirety, proportion, and for this judgment is primarily necessary. The conspicuous lack of judgment is everywhere responsible for incompetent sampling at blast-furnaces. The small-scoop practice might produce fair average results in soft fine ores, but its size— $3\frac{1}{2}$ in. wide, $2\frac{1}{4}$ in. long, and $1\frac{1}{4}$ in. deep—would debar its usefulness on hard-ore mixtures ranging from richest lump to gangue or rock, the size of scoop being insufficient to take care of lumps 6 to 15 in. in size, including surrounding ore. A scoop at least of shovel size would be necessary to the taking of *all* sizes in given spots and places. This would, of course, importantly increase the bulk of the total sample and necessitate more men to handle it. In sampling hard ores Mr. Murray reverses himself and really leaves the matter to the sampler's judgment when he allows him to take an average of the lumps from which the cubes of ore are broken. This would occur at hundreds of points throughout the cargo and constitute a judgment result on the part of the men whom he says lack the quality of judgment.

In my opinion, sampling from boats is to be roundly condemned, forasmuch as there will always be considerable unsampled ore rolling to the vessel's sides, and the fact seems to be ignored that approximately one-half of the cargo is in the lower one-third of the base of the pile. As I understand the method of Lake ore sampling, this lower third is left untouched, hence it would appear that about one-half of the cargo remains unsampled unless, peradventure, samplers have gone into said lower third of the pyramid.

The present method of sampling iron ore from cars at the docks of New York, Philadelphia, and Baltimore has satisfactorily stood the test of more than a third of a century, and pioneer samplers of 30 years ago are still on the job fortified by this long period of experience. In my opinion, it is not within the ability of man to produce for the sampler a more thorough exposure of ore than comes to pass when bucketful after bucketful of it is dumped on to cars under the eye and observation of the competent Eastern sampler inspecting its flow and accurately sampling therefrom.

L. S. AUSTIN, Salt Lake City, Utah (communication to the Secretary*).—In July and August, 1908, I spent six weeks observing the sampling and handling of iron ore from the Mesaba ore deposits to the furnaces at South Chicago, Gary, Cleveland, and Pittsburg. In reading Mr. Murray's paper on the subject of uniform methods of sampling, I do not find that conditions have changed at the receiving end since this investigation. I am, in consequence, of the opinion that the shipping companies are quite right in insisting upon settlements on their samples.

* Received Feb. 11, 1914.

True, there are discrepancies between the mine and the furnace samples, but the latter are not and cannot be authoritative, since they are too imperfectly and cheaply taken, and to take proper furnace samples would cost more money than the smelting companies have heretofore been willing to spend.

I will describe the methods of sampling iron ore on the Mesaba range, and compare them with those pursued at the docks at lower-lake points and at the furnaces.

The sampling of the ore on the range has been made by taking portions uniformly over the top surface of the car load. This is described in a pamphlet issued by the Chemists' Committee of the U. S. Steel Corporation on *The Method for the Commercial Sampling and Analysis of Iron Ores*. This is done immediately after loading, so that the ore may be assigned to the proper pockets at the ore docks. The united sample of a "drag" of 10 cars is taken at a time, and it is evident that an average sample of this unit can be counted on, since numerous points over all the cars are taken and at uniform distances apart. Wherever in the sampling a lump of ore is encountered this is broken and a proper proportion of it is taken. These "drags," of 8 to 12 cars but averaging 10 car loads, are made up in train loads of 50 cars and dispatched to the terminal yards, where the cars are assembled into "blocks."

The weighing is done over automatic scales, which register the weight while the train is moving slowly over the scale platform. The car number and the name of the mine are put on the registering tape, and a duplicate record is also made. A great deal of pains is taken to examine and keep the scales in repair, to check the railroad weights, and to compare them with those of a railroad scale of the ordinary type.

Loss of Weight before Weighing.—While in theory the sample of a car is to be taken at the time of weighing, the time between the taking of this sample and the weighing is brief, and in the solid body in which the ore exists in the car the drying-out would be trifling. This idea is corroborated by the statement made me that trial car loads set out and weighed from time to time have shown an inappreciable loss of weight.

Loss of Weight.—Ore may be spilled from the cars before weighing, but this can at any time be gathered up and loaded. Ore shakes down in transit, and therefore no spilling occurs after weighing until the docks are reached. Here, when discharging into the vessel, some ore escapes over the edges of the chutes into the water, but as compared with the whole bulk this does not amount to much. From the railroad weights a deduction of 1 per cent. is made for contingent losses, such as spilling and moisture loss.

Make-up of Train Loads.—The content of any train load is computed on the basis that each drag of 10 cars is uniform. Figs. 2, 3, 4, and 5 represent by graphic curves the make-up and composition of four train

loads, from which the variations can be seen. At the terminal yards these trains are broken up under orders from the shipper, who is at the mine; the ores for a given cargo are selected from them, and these cars are sent to the docks.

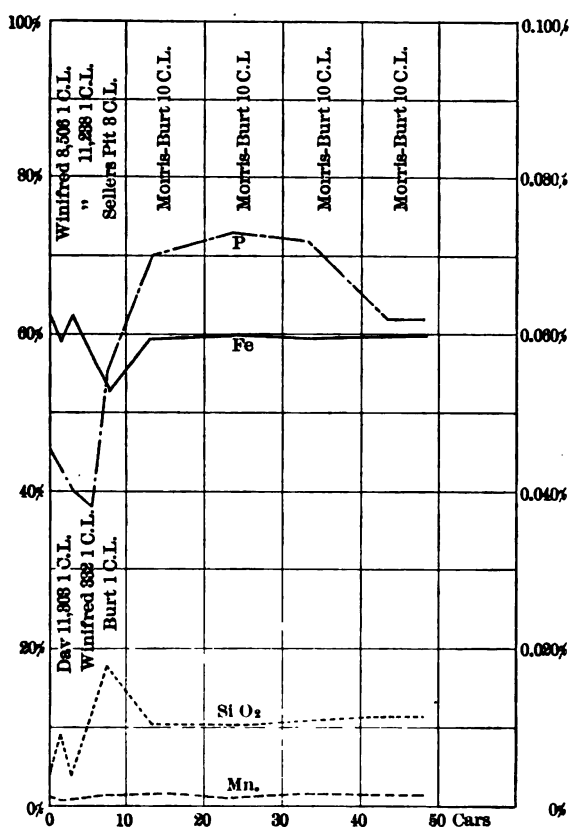


FIG. 2.—TRAIN No. 334.

Selection of Cars for Cargoes.—Upon examining the figures it can be seen which of the drags are Bessemer iron, for example, to go out as a Bessemer cargo. We find that for this grade the line for phosphorus must fall a certain distance below the line for iron. In Fig. 2 the first five carloads, and in Fig. 4 the six carloads of Meyers ore, belong in this category, with such a margin of safety that three carloads of Hartley Block 71 can be added. This ore is of a grade called "Group 2," whose requirements are that it is to contain 60.5 per cent. iron, 0.048 to 0.050 per cent. phosphorus, 5.25 per cent. silica, and 0.69 per cent. or less manganese, all based on dry weight. This corresponds to 55 per cent. iron and 0.45 per cent. phosphorus "natural";

that is, the ore as mined and containing its natural moisture, which may vary from 10 to 15 per cent.

Making up Blocks.—Upon notification that a certain vessel is to arrive at the docks, the shipper prepares a sheet called a "block"; that is, a selection of cars at the yard which will make a certain grade or "group." He wires orders for the terminal yard to send down to the docks certain cars, picked out from the train loads as above specified, and conforming to the requirements of the group which is to be shipped.

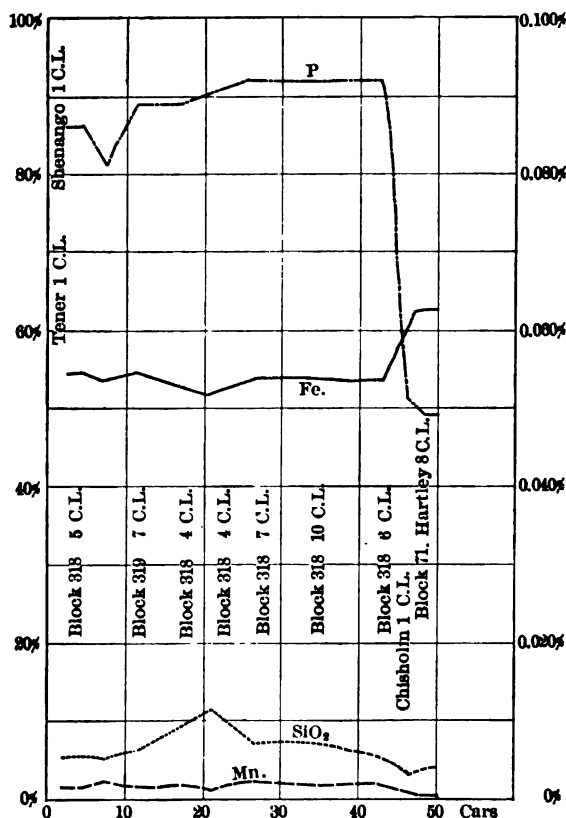


FIG. 3.—TRAIN No. 340.

Figs. 6 and 7 are two blocks, graphically represented, one for the boat "Clemson," and one for the boat "Henry Phipps." Block 156 for the "Clemson" consists of a portion of trains 330, 340, 342, trains 343 and 344, a special train load of 390 tons of drags 140 to 151, inclusive, and "holds" 18 to 21, inclusive, being cars that have been held over and not sent down with former train loads, also 110 tons belonging to block 166, which had been put into a pocket at the docks, and which had been held until now for shipment. Proportions of older blocks (called "holds")

found not to be needed for a given cargo are also held in cars at the yard, and are put into later blocks, as shown in Fig. 7. In Fig. 6 it will be seen that the elements vary as between different train loads but the average will conform to the requirements of the group which is to be made up. Thus the "holds" 18 to 21, and train 340, would be too high in phosphorus were it not for the fact that the rest of the cargo is lower than the requirements in that element, and can stand the addition of

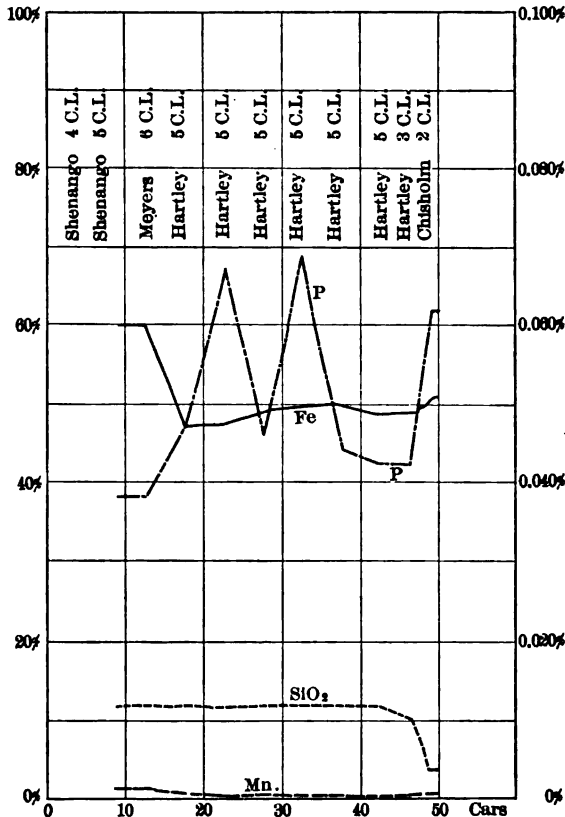


FIG. 4.—TRAIN No. 349.

these holds. In Fig. 7 the specifications of Group 3 are fulfilled, the average conforming to them, though the variations in silica are such that, were it to go to the furnace no better mixed than here shown, trouble would result. To lessen this difficulty, orders are given by the shipper to the dock superintendent to scatter or distribute "off lots" through the pockets selected for holding the block in readiness for the intended vessel; and in addition, even to put this in the middle of the pocket, so that it will be evenly mixed with the rest.

The Ore Pockets and Dock.—Fig. 8 is a cross-section of a dock with the ore pockets and the hinged chutes for delivering the ore into the hold of the vessel. The position of the successive car loads is shown, with the order in which they fall into the pocket. These chutes are let down when the vessel is in position, the chute gate is opened, and the ore slides into the vessel. The boat itself has numerous hatches, large boats having 36 at 12 ft. center to center, so that ore may be filled into as many as 20 hatches at a time. Commonly the force of loaders is not sufficient

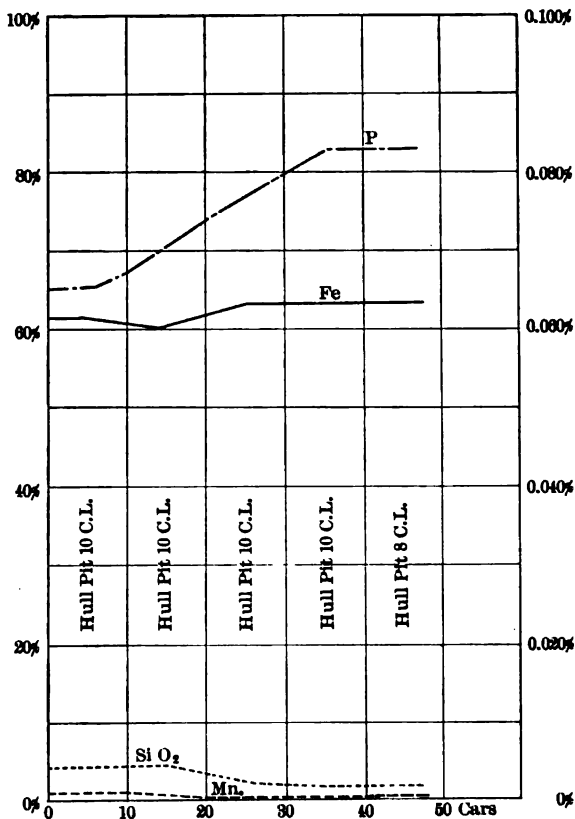


FIG. 5.—TRAIN NO. 350.

to keep so many going at once. The ore is poked by the loaders both at the gates and from above. It comes away first at the front, then at the back, as shown by the broken lines *a* and *b* in the right-hand pocket, cars Nos. 1 and 3 (Fig. 8) being the last to leave the pocket. Still, if two or three grades of ore go into a pocket, some portions on top accompany that below, so that a good deal of mixing results. We may therefore see ore of different colors, belonging to one or other of the car loads of a pocket, sliding into the hold of the vessel. Viewed from above these

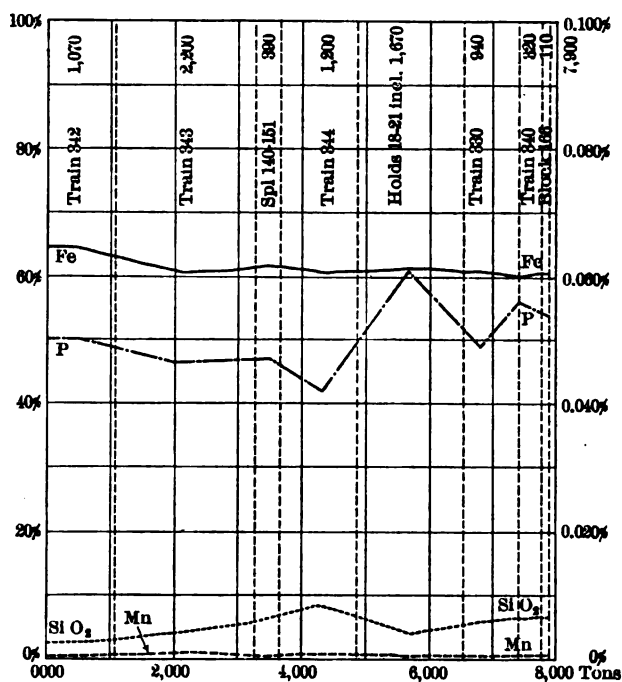


FIG. 6.—BOAT "CLEMSON." TONNAGE 7,900. BLOCK 156.

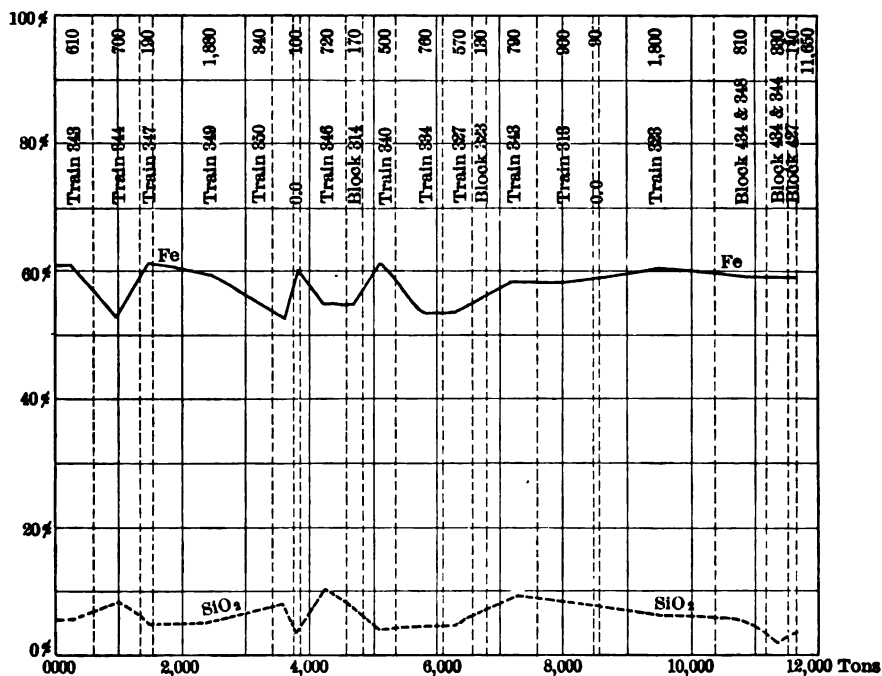


FIG. 7.—BOAT "HENRY PHIPPS." BLOCK 323.

layers appear to separate as they shoot downward, but they are not deep. There is a segregation of lumps and fine, the coarser ore rolling to the

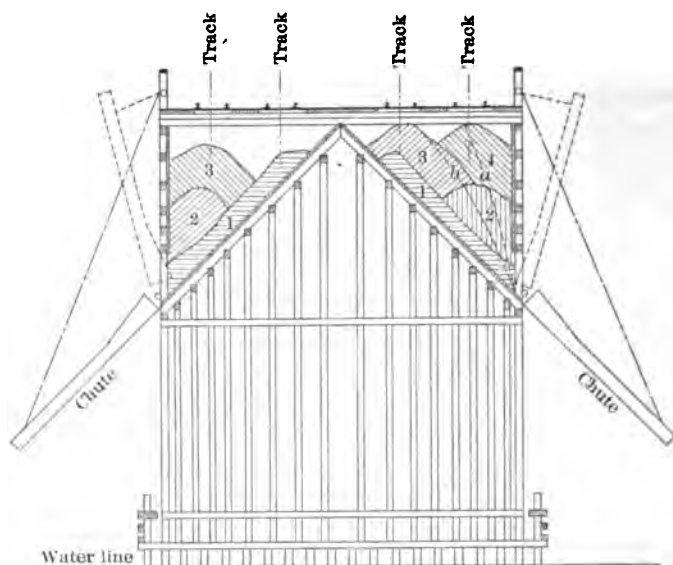


FIG. 8.—CROSS-SECTION OF DOCK AND ORE CHUTES.

sides or borders of the heap forming in the hold, the fines keeping to the center or apex. This is shown in a cross-section of a 600-ft. boat, Fig. 9.

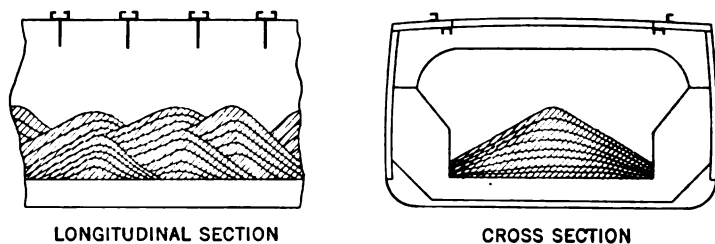


FIG. 9.—SECTIONS AND ELEVATION OF ORE BOAT, SHOWING POSITION OF ORE IN THE HOLD.

The same figure shows the ore layers in longitudinal section, and below is a longitudinal elevation of a 600-ft. ore boat.

Lumps and Fine.—Lump ore is often of the same grade as the fine; at other times, as indicated by Fig. 10, it is of lower grade. Thus in the Pewabic mine sample of that figure the oversize of $\frac{1}{4}$ -in. mesh (45 per cent. of the whole) contains Fe, 37 per cent.; the size between 16 and 20 mesh (8 per cent. of the total) contains Fe, 41 per cent.; and this lump ore is low in grade as compared with the regular shipping ore. Fig. 11 shows still greater variations of lumps as picked from a cargo of the boat "Corona" in 1907. Here three kinds, blue fine No. 1, blue lump ore,

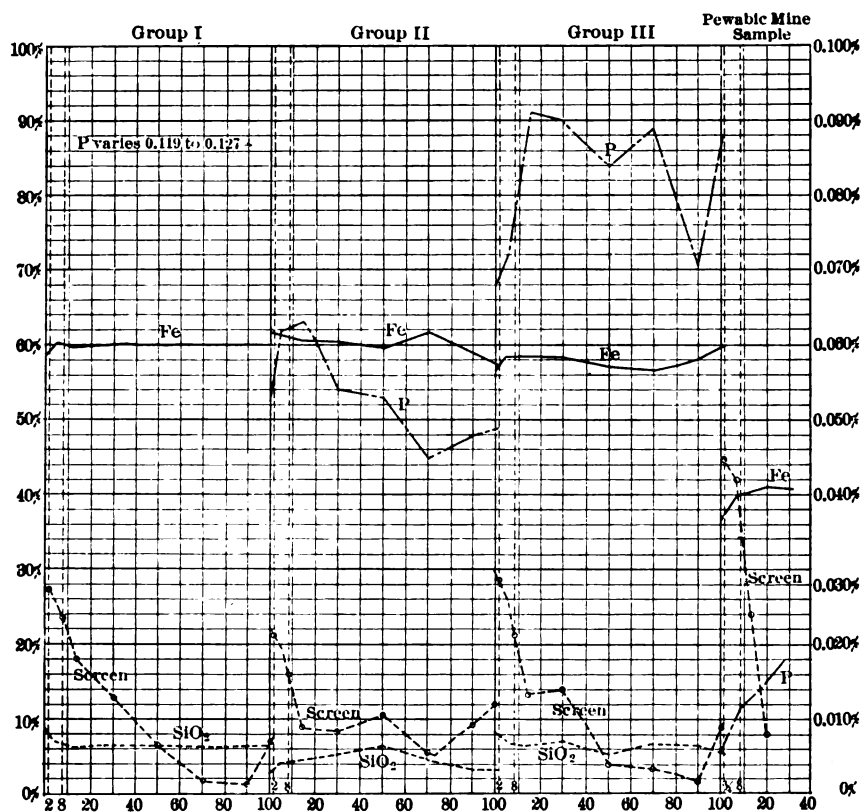


FIG. 10.—SCREEN ANALYSIS AND CONTENTS OF ORE.

and blue lump ore with stripes, carry over 60 per cent. Fe, while the three samples of slate vary from 45 to 8 per cent. Fe. Evidently such low-grade lumps, if they found their way into the small samples so often taken, might easily give a low result. It has been stated that from certain mines lump ore may be found which is of higher grade than the fine; I have no data indicating this. We may accordingly expect to find a difference between the finer center and the coarser borders of the pile, formed when a boat is being loaded.

Losses in Handling.—The loss of weight in transit due to drying is slight, but is more in loading and discharging. In the stock pile it may either dry out or take on moisture from rains, so that furnace weights afford no adequate check on the shipper's weights. One may note weigh scales forming part of the unloading rigs. These are not always in use. Often a part of the cargo is unloaded to cars to go to another works, part

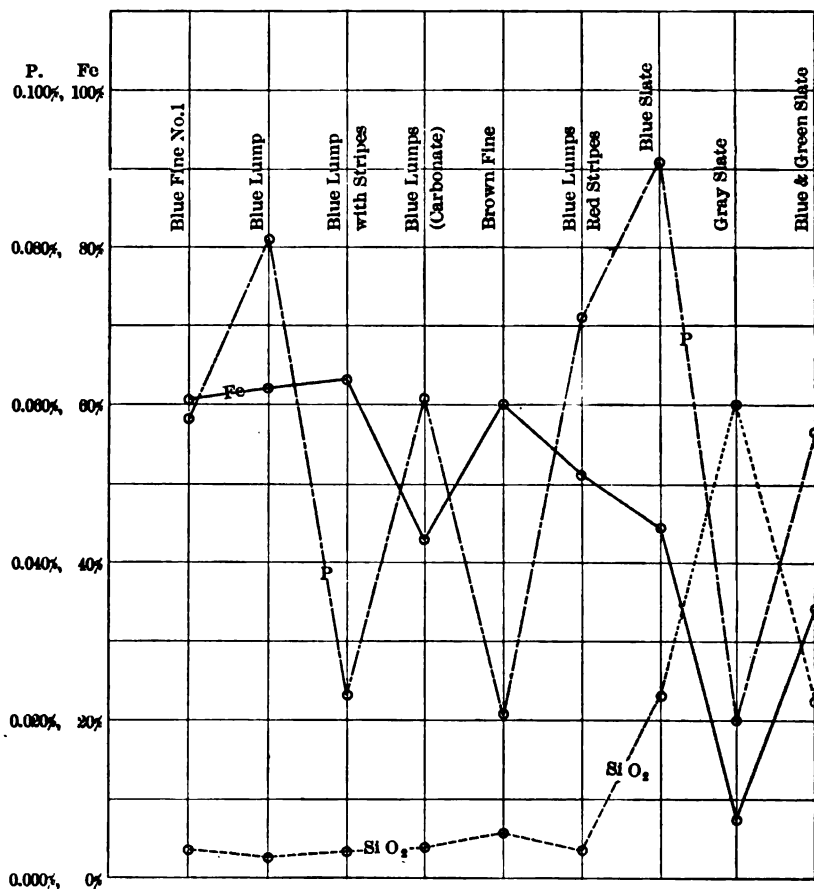


FIG. 11.—ANALYSIS OF IRON-ORE LUMPS.

to the stock pile. The cars are weighed, their aggregate subtracted from the reported weight of the cargo, and the remainder charged to the stock pile. When one considers losses due to moisture and spilling this method seems inadequate for keeping a close record of quantities.

Time for Loading.—The cargo boats vary in capacity from 3,000 to 12,500 tons at a maximum draft of 20 ft. The typical lake ore carrier may be taken at 10,000 tons. When not overcrowded, and with the handling of ore and boat well attended to, a cargo is shot into a vessel

in 5 hr., to which should be added 2 hr. for shifting to the desired pockets, and 2 hr. for waiting time. Another delay may result from a vessel arriving and finding another in advance of it, covering the pockets from which it has to draw.

Distribution of Cargo.—To oppose the buoyancy of the unloaded bow and stern, the boats are more heavily loaded at the end hatches, as graphically shown in Fig. 12, where the number of car loads per hatch is given. Now, in sampling, if equal weights of ore are taken at each hatch proportionate representation will not result. As far as my observation goes no such variation is allowed for in taking the sample at the lower-lake or receiving ports.

Cargo-Sampling Method.—The method recommended by the Chemists'

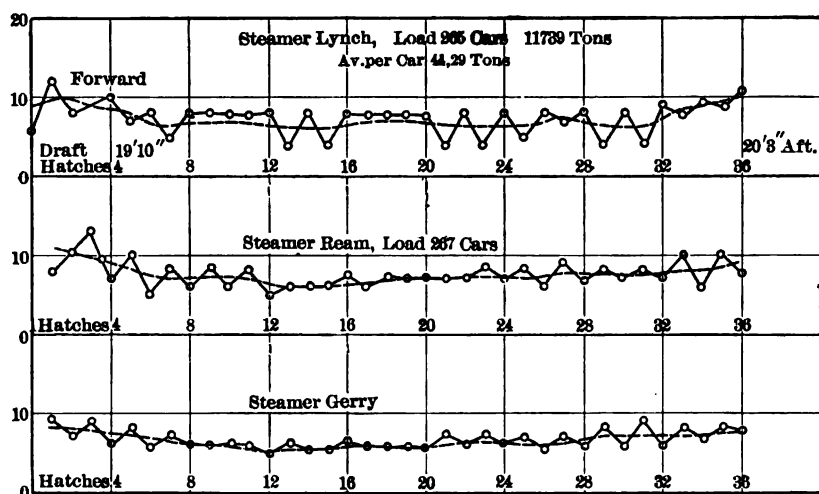
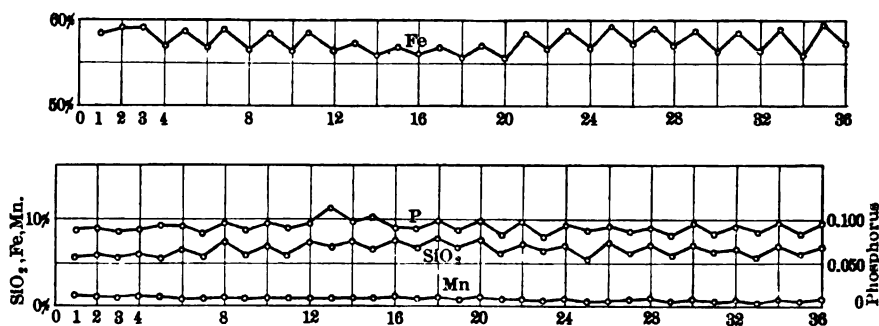


FIG. 12.—DISTRIBUTION OF CARGO.

Committee of the U. S. Steel Corporation for the sampling of a cargo, in a boat having hatches at 12-ft. centers, is, in any given hatch, to take a surface sample from the top of the heaped or coned ore, then, when the ore has been largely removed from a given hatch, a sample from the freshly exposed faces. The surface sample may amount to one-eighth, the face samples seven-eighths of the total. Now, were the last car loads shot into the vessel ore of high grade, the cargo sampler would get a high result; were they low, then samples would be depressed, even though the rest of the shipment were high grade.

Actual Results on a Cargo.—To arrive at the actual composition of a specific cargo I obtained from the train dispatcher at Hibbing, who assigns cars and makes up the blocks, the numbers of the cars and their composition, and from the dock superintendent, the pockets to which cars

are assigned. These cars constituted block 377, and were loaded into the boat "Ream." I also got from the dock superintendent the numbers of the pockets, and of the hatches of the vessel into which the pockets were discharged. In this way it has been possible to state with precision what has been the composition of the ore in each hatch, and that at the top surface of the coned-up ore. This is shown graphically in Fig. 13, in which the average analysis of the ore of each hatch is given.



Railroad weight of cargo 11,793 tons. B/L weight, 11,678 tons. Average analysis: Fe, 58.94; P, 0.090; SiO_2 , 6.48; Mn, 0.91; Moisture, 13.27 per cent.

FIG. 13.—CONTENTS OF CARGO, STEAMER "REAM."

In the following table are given the weights and contents of the cargo of the "Ream," 11,796 long tons, the gross weight being billed at 1 per cent. less than the railroad weight, or 11,678 tons.

TABLE I.—Cargo of the "Ream"

	Tons	Iron	Phosphorus	Silica	Manganese	Moisture
Hull Pit	5,529	60.74	0.080	4.56	0.71	14.34
Sellers Pit	1,524	58.26	0.091	8.90	0.42	10.19
Tener	272	54.07	0.085	6.86	2.26	11.13
Morris-Burt	714	57.42	0.083	6.28	1.32	12.20
Hartley	80	53.60	0.099	8.74	1.44	17.90
Burt	2,993	53.89	0.115	8.59	1.28	13.29
Glen	341	54.95	0.070	7.99	1.07	13.43
Clark	176	59.16	0.056	5.28	0.94	11.33
Winn-day	49	56.82	0.089	9.30	0.94	14.00
	11,678	58.94	0.090	6.48	0.91	13.27

Where, as in the shipper's sample, the actual weights and analyses that correspond are taken, the average of the cargo is 58.94 per cent. iron. On the other hand, assuming, as in cargo sampling, that an equal weight of ore is in each hatch, the resultant average will be 57.52 per cent. iron only. This will illustrate that in cargo sampling serious errors may result.

Complaint has been made by furnace superintendents that unexpected variations in cargoes, not shown by averages, have caused trouble in the

working of blast-furnaces. In this particular cargo the variations are not serious, as the figures given below indicate:

Table II.—*Variation in the Boat "Ream"*

	(1) In ore delivered at the dock					
	Iron	Phosphorus	Silica	Manganese	Moisture	
Maximum	60.74	0.115	8.90	2.26	17.90	
Minimum	53.60	0.056	4.56	0.42	10.19	
Average	58.94	0.090	6.48	0.91	13.27	
	(2) In ore delivered into the boat					
Maximum	59.50	0.114	7.80	—	—	
Minimum	55.60	0.080	5.50	—	—	
Average	57.40	0.092	6.60	—	—	
	(3) Surface sample of ore in boat					
Maximum	58.8	0.156	11.00	—	—	
Minimum	51.9	0.067	3.90	—	—	
Average	55.4	0.103	8.30	—	—	

In this table we note, first, that as the result of spreading out of exceptional lots, and because of the consequent mixing, the ore has averaged better as it is shot into the hold. In (3) we see how a surface sample may vary from the average of the cargo, and also that if the ore from that point were introduced into the blast-furnace it would cause more serious trouble than if the full depth in any hatch were used:

Stocking Ore at the Furnaces.—Ores of the same group are stocked together. If such ore were more carefully bedded, and for the furnace

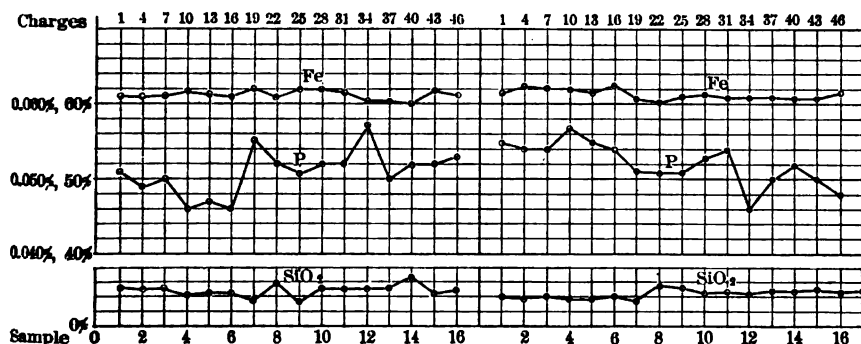


FIG. 14.—ANALYSES OF GROUP 2 ORE FROM CHARGE POCKETS.

charges reclaimed from top to bottom of the pile, we would count on greater uniformity. If stocked haphazard from two or more cargoes, even when of the same group, the variations, when delivering from the receiving bins to the charge skips at the furnace, are shown in Fig. 14. The sample was taken as the ore was delivered into the skip at every skip load. It will be seen from the curves that at charge 34 in the first trial, and at charge 10 in the second, the phosphorus passed the limit

which would permit the production of Bessemer pig. We may infer that the silica did not vary enough to cause a serious disturbance in blast-furnace working.

Stocking Iron Ore from the Boat.—Fig. 15 is an elevation of a Hoover & Mason unloading rig, of the stock piles made by it, of the concrete trough into which the ore from the vessel is delivered and from which it is reclaimed to the stock pile; also a cross-section of the vessel being unloaded. Ore is taken from the boat in a 6-ton grab, which may deliver it either to a receiving hopper on the rig itself, or to the concrete trough behind it. Ore from any hatch is, at one setting of the unloading rig, delivered at one point at the trough. Here some segregation occurs, the lumps rolling to the foot of the pile. Thus we get a series of conical heaps of ore, their centers fine, lumpy where they overlap. The ore in the trough is reclaimed and sent to the stock pile adjoining. Since, however, the two ends of the bridge may be independently moved to the extent of

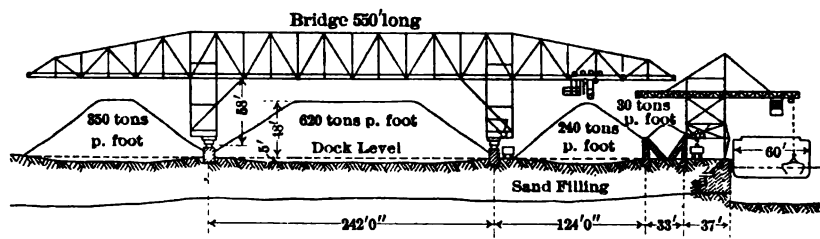


FIG. 15.—DISTRIBUTING AND HANDLING BRIDGE.

30 or 40 ft. more at one end than at the other, it should be possible to so set it that coarse ore may be placed upon the fine, and thus some mixing be done. The grabs on the bridge are of 14 tons capacity. By carrying up the track walls of the bridge, it would appear that storage capacity might be increased, and the long slope be diminished. If, with this, attention should be given to carrying up the pile evenly and progressively from the first, the ore bed would again be more uniform. Not sufficient attention has been given to this point and as long as the cargo was unloaded promptly it did not seem to matter how it was done. Ore naturally tends to segregate, and pains and study should be given to averaging it.

Cargoes, at the time of loading, are made up to a guaranteed composition. In Fig. 16 this composition is shown graphically, as well as the actual analyses of several cargoes as taken at the mine and at the receiving port during 1908 on Mesaba iron ores. As regards the mine and cargo samples there are plus and minus differences, indicating that the results are intended to be fair, though the cause of difference or the comparative accuracy is not indicated. My own opinion is that, as taken at present, the mine sample is the more reliable, though it could be improved. It

has the advantage that numerous results are given, and the actual weight of the ore is used in computing the average.

A sample of the ore taken automatically, as it drops from a grab or from a Hulett machine, would give an aliquot portion of both coarse and fine. This sample could be kept in a pocket until taken away in a drop-bottom tram car. Such a car would deliver to the sampling mill, where the material would be automatically sampled by well-known methods. The system should be properly worked out so as not to interfere with the un-

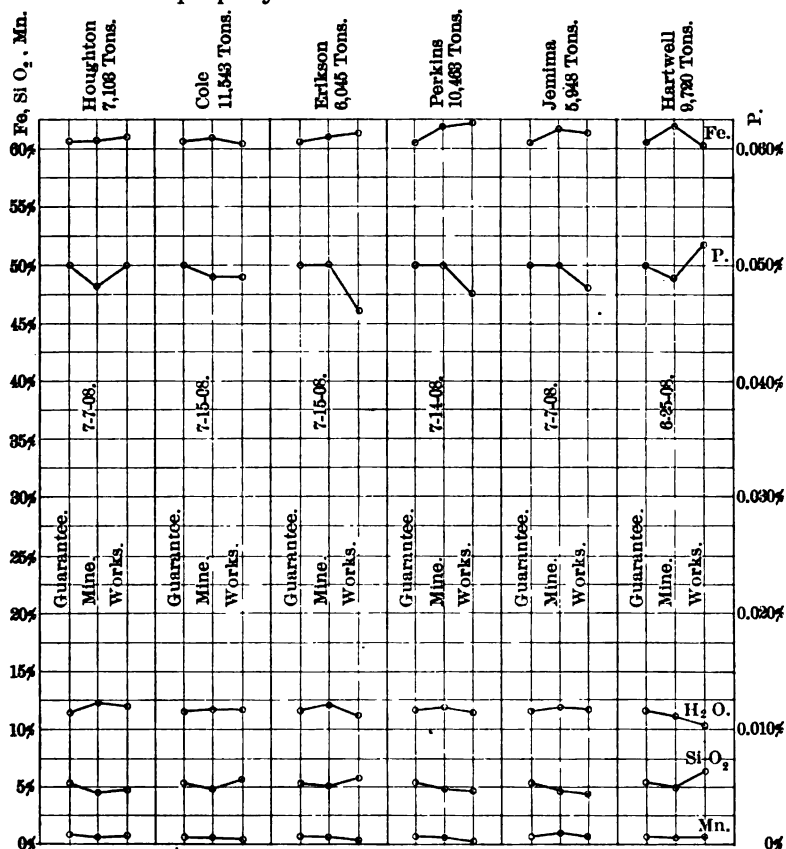


FIG. 16.—GUARANTEE, MINE, AND WORKS RESULTS ON CARGO SHIPMENTS.

loading. I compute that for a 1 per cent. sample, taken from the falling ore, it would cost 0.75c. per ton.

Of the present methods of cargo sampling, that carried out by some of the independent chemists at Cleveland seems to me the best. They use a short-bladed scoop which takes a portion of uniform width. Wherever a lump of ore is encountered the sampler judges how much of it he is to take. The method of taking is by rounds, as described by Mr. Murray. The moisture sample is taken from the ore samples as collected

in a can at each hatch. The contents of a can are emptied on the floor, the lumps broken, the whole mixed and returned to the can. Later a weighed portion of this is taken for drying at 212° F., and the moisture loss determined. The united ore samples of all the hatches are used for the average sample.

When it is considered that \$50 to \$75 is paid for the work of sampling and analyzing a cargo, the remuneration seems small for the care and responsibility involved; it amounts to 0.10c. to 0.15c. per ton. Compare this with the cost of sampling a precious-metal ore. Here the shipper cheerfully pays 50c. per ton if it is sampled at a customs sampler, 5c. to 10c. per ton if he engages a watcher to oversee the works sample. Nothing justifies spending only \$50 to \$100 to obtain results on material worth \$40,000.

At the furnaces, sampling seems to be done only with the care needed for controlling blast-furnace operations, and to get results that will check the mine. The blast-furnace superintendent seems to know quite well that the results of sampling are approximate only. I cannot see how he would feel like taking the responsibility of incurring expense, additional thought and pains to change the present system.

The U. S. Steel Chemists' Committee recommends a method of sampling called by them "grab" sampling; that is, taking a scoopful from the top surface of the ore in the grab as it is rising from the hatch. The method has much to commend it, as there is an equal representation of ore throughout the cargo; the chief objection to it in my mind is that the lumps would not be properly represented in the hasty taking.

Indeed, too much in regard to sampling is a matter of opinion and theory, and such conjectures could be set at rest only by sampling car loads, etc., in an absolute way, and by comparing these results with the present ones.

One may say then that sufficient attention has not been given to the sampling and stocking of ore, and that in consequence variations that occur here and there in a cargo, may appear at the surface in an aggravated way. This is the converse of the proposition that errors are disposed to balance one another, but it also logically leads to the admission that errors may be cumulative.

In view of the conditions of ore handling, I see no way for obtaining a uniform ore charge, unless each plant takes care of and beds its own ore. It has no right to complain of the shipper, who has only to guarantee the whole cargo.

A number of other points, not alluded to in this discussion, were developed as the result of my investigation, all of which, if put in practice, would conduce to reliability in sampling and to regularity of the ore charge, and consequently to regular running and to uniformity of the pig produced.

Pig Steel from Ore in the Electric Furnace

Discussion of the paper of ROBERT M. KEENEY, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 86, February, 1914, pp. 349 to 367.

J. W. RICHARDS, So. Bethlehem, Pa.—I am extremely interested in this paper of Mr. Keeney's, which is one of the most elaborate published, so far, on the attempt to produce a low-carbon product directly in the electric pig-iron furnace. I think that the commercial side of it is very important. If we can make a low-carbon product, equivalent in composition, as far as carbon is concerned, to steel, and containing only moderate quantities of the other impurities, silicon, manganese, sulphur, and phosphorus, the refining of that pig steel to finished steel in the open-hearth furnace is a much easier operation than the refining of ordinary pig iron. That has been proved already. Several years ago I discussed that matter in connection with the Swedish electric pig-iron furnace, and I have always thought that it will eventually develop into commercial practice.

JOHN CRAWFORD, JR., Heroult, Cal. (communication to the Secretary*).—The numerous and careful experiments carried out by Mr. Keeney on the possibilities of successfully making "pig steel" in the electric furnace have been studied by the writer with a great deal of interest. Judging from Mr. Keeney's conclusions, the five main advantages claimed for pig steel over normal low-silicon electric-furnace pig iron are:

1. That it can be converted into steel with greater facility and at a lower cost.
2. That it admits of greater economy in power consumption.
3. That it admits of greater economy in reducing material.
4. That the carbon content can be controlled within a limit of 2.20 per cent. maximum.
5. That the losses of iron in the slag will not exceed reasonable limits, say 6 per cent. FeO.

As to the first claim: From published reports, white electric-furnace pig iron, sufficiently low in silicon and carbon to approach or actually come within the limits of pig steel, has not only been used successfully but has met with a very favorable reception by Swedish steel makers. Hence a recital of our efforts to dispose of a similar product to steel makers on the Pacific Coast may be of interest. In May, 1912, there was in our stock yard about 150 tons of white iron which had been made some time previous in our shaft furnace. This was semi-porous and quite similar both in appearance and analysis to the pig iron which was

* Received Apr. 9, 1914.

at that time reported as being used successfully for steel making in Sweden. The analyses and weights of the several casts were as follows:

Silicon Per cent.	Carbon Per cent.	Weight Tons
0.56	2.47	6
0.19	2.60	12
0.26	2.82	5
0.43	2.92	5
0.86	2.66	5
0.46	2.72	6
1.09	2.86	10
1.11	2.58	10
0.94	2.89	12
1.53	2.82	10
0.97	2.71	8
0.75	2.60	12
0.73	2.60	14
1.00	2.73	9
0.60	2.43	10
0.37	2.39	10
Average 0.74	Average 2.67	Total 144

The sulphur was uniformly under 0.03 per cent.; the phosphorus under 0.04 per cent.; and the manganese varied from 0.22 to 0.28 per cent.

Stimulated by the results which were being attained in Sweden we made considerable effort to induce steel makers on the Pacific Coast to try it. Finally by making a very attractive price we induced one firm to use it. This concern operates both basic and acid open-hearths for making concrete reinforcing steel. The report of the works superintendent on it was rather indefinite but not enthusiastic. He said "It worked pretty well but normal pig iron was more satisfactory." Since that time we have sold this concern about 1,000 tons of normal low-silicon pig iron and also a few odd carloads of white iron which approached in analysis pig steel. They advise that no difference is noted in the working of normal electric-furnace pig iron as compared to normal blast-furnace iron of similar analysis. The pig iron brought the market price, but the pig steel always had to be offered on more favorable terms, and even at a reduced price it did not call forth any repeat orders.

These remarks are not intended to be construed as a contradiction of the favorable reception which pig steel has been accorded by the Swedish steel men, but it does seem to indicate that to gain the advantages claimed pig steel must be used "according to prescription" and a good deal of "missionary work" might be necessary before American steel makers, who have no patriotic interest in the matter, would do the necessary experimenting to prove the advantages claimed.

In regard to claims 2 and 3, while there is certainly an appreciable saving both in reducing material and in power in making low-silicon as against high-silicon iron in the electric furnace, how much this saving can be further increased by making pig steel instead of normal low-silicon pig iron is an open question. So far as the writer knows, no data have been published covering a continuous, systematic effort to make pig steel on a commercial scale. Messrs. Nyström and Leffler, in closing their report of a 215-day run in electric smelting at Trollhättan, make the statement: "The possibility of a very low carbon product—pig steel—has not yet been tried." As this report was made almost two years ago and no further data on making pig steel have been given out, it is fair to assume that either the possibilities of making it did not seem feasible or that if tried the results were not altogether successful. This opinion seems to be further verified by the statement of Ivar Barthen, Assistant Chief Engineer, Jernkontoret, Stockholm,¹ who says, referring to electric-furnace operations in Sweden: "The first expectations of the inventors may have been that only as much coal should be put in the [electric] blast-furnace as was needed by the chemical constitution of pig iron, as regards carbon; the heat indispensable to the process being secured by electric current. Even if those expectations could not be realized, still about 65 per cent. of the coal ordinarily consumed is saved."

We find in Messrs. Nyström and Leffler's report that for a period of 215 days' continuous operation, during which over 3,000 tons of iron was made, the silicon averaged only 0.72 per cent., which seems to indicate that although they were not endeavoring to make pig steel the metal produced was approaching it more nearly than it was normal pig iron. Since there are no data available to indicate that the fuel consumption has been lowered, we assume that the figures as given in their report are still substantially correct. It may be interesting, therefore, to make some comparisons between carbon economy attained in making low-silicon pig iron in Sweden, the results obtained by Mr. Keeney in his experimental furnace, and those obtained in producing the few casts of low-silicon iron previously noted as having been made at Heroult in a shaft-type furnace.

From the Trollhättan report we gain the following information:

Pig iron produced (95.36 per cent. Fe), metric tons.....	3,215
Coal charged (72.89 per cent. C), metric tons.....	1,301
Per cent. carbon in pig iron (average).....	3.406
Per cent. silicon in pig iron (average).....	0.72
Per cent. FeO in slag (mean).....	3.40

¹ *Iron Age* vol. xciii, No. 4, p. 253 (Jan. 22, 1914).

Furnace	Per Cent. Carbon Used to Iron Produced	Per Cent. Carbon Used in Reduction and Losses to Iron Produced	Carbon in Iron, Per Cent.	Silicon in Iron, Per Cent.	FeO in Slag, Per Cent.
Trollhättan.....	29.5	26.1	3.40	0.72	3.40
Keeney.....	28.7	27.6	1.07	0.11	6.56
Heroult.....	29.9	27.2	2.67	0.74	4.30

A comparison of these figures shows that though Mr. Keeney has made pig steel very low in carbon and silicon, the per cent. of total carbon consumed is little better, considering the greater loss of iron in the slag, than is attained in shaft-furnace practice when making low-silicon iron. The reason for this lies in the fact that in the shaft furnace the carbon used in reduction is oxidized to CO and CO₂ in the ratio of about 2.5 to 3 CO to 1 CO₂ by volume (the average given by Nyström and Leffler is 2.68 to 1), whereas in the small rectangular furnace used by Mr. Keeney we find the average ratio CO to CO₂ to be 4.7 to 1, with a minimum of 1 to 1 in experiments Nos. 1 and 2, where the FeO loss in the slag was 14 and 13 per cent. respectively, and a maximum of 11.2 to 1 in experiment No. 12, where the FeO loss in the slag was only 0.50 per cent. (Experiment No. 13 was excluded as it appears erratic.)

If we recast Mr. Keeney's figures on a basis of CO:CO₂::2.75:1, which makes them comparable² with results attained in the shaft-type furnace, we have:

Furnace	Per Cent. Carbon Used to Iron Produced	Per Cent. Carbon Used in Reduction and Losses to Iron Produced	Carbon in Iron, Per Cent.	Silicon in Iron, Per Cent.	Fe in Slag, Per Cent.
Trollhättan, shaft type.	29.5	26.1	3.40	0.72	3.40
Keeney, shaft type	25.5	24.4	1.07	0.11	6.56
Heroult, shaft type.	29.9	27.2	2.67	0.74	4.30

These revised Keeney figures show a saving in carbon of 4 to 4.5 per cent., or an economy in reducing material of about 5 to 6 per cent. The

² Since at Heroult the circulating gases were not scrubbed and since at Trollhättan the moisture was reduced to 0.5 g. per cubic meter, and because part of the carbon consumed in reducing CO₂ in the circulating gases to CO is later recovered by the reducing action of the CO on the ore, no account was taken of these in revising Mr. Keeney's figures. Though not strictly accurate, the figures given are close enough for comparison.

metal is low in carbon and silicon and the slag loss is not unreasonable. They are certainly very attractive, but can they be attained or even closely approximated in actual practice? The writer does not think so. He believes we have already attained very close to the practical limit of carbon economy in the types of furnaces existent. Any further appreciable decrease in reducing material will be at the expense of lower recovery.

Mr. Keeney cites (p. 363, Loss of Iron in the Slag), a result obtained at Demnarfvet, where pig steel running 1.70 per cent. C was accompanied by a slag containing only 0.23 per cent. FeO. In the absence of knowledge of the fuel ratio used and the analyses of the slag and metal taps immediately preceding and following it is hardly safe to make any observations on this, but in experiment No. 12, which Mr. Keeney cites as analogous, we note the carbon used is 29.8 per cent. of pig iron produced, which cannot be considered any economy over present shaft-furnace practice.

The writer's justification for the opinions given in the preceding paragraphs is based on observations of both shaft-type and rectangular-type furnace at Heroult and on such data as have been available concerning electric iron furnaces, operating elsewhere. The results of his deductions may be summarized as follows:

The theoretical amount of carbon required to make pig metal from iron ore depends—

1. On the ratio in which the carbon combines with the oxygen in the ore to form CO and CO₂.
2. On the carbon absorbed by the metal.
3. On the carbon used in reducing silica and the other metalloids.
4. On the carbon consumed by oxygen dissociated from aqueous vapor and atmospheric oxygen occluded in the reducing material and sucked into the furnace through cracks.
5. On the portion of the carbon consumed in reducing CO₂ to CO which is not later recovered by the reducing action of the latter on the ore.

With sufficient carbon present to satisfy these requirements we should make pig iron, with an accompanying slag free from iron. But whether we do or not depends on another factor, which, though understood by all metallurgists, does not seem to have been given the importance it merits in electric smelting; this is, the ratio of size of the reducing material to the ore and the homogeneity of mixture of the two when they have descended to the smelting zone. Even in the shaft-type furnace, where a considerable reduction by CO can be counted on, much ore reaches the smelting zone of the furnace wholly or partly unreduced. Whether this is smelted at this point to an oxide which combines with the silica of the slag, or is reduced to metal, depends on the intimacy of its contact with carbon. Our experience with the shaft-type furnace indicated that even with a brittle reducing agent, like charcoal, if an excess over theoretical

calculations was not used the slag losses were erratic and generally high. The absorption of carbon by the metal was also erratic, but it did not always vary in inverse ratio to the slag losses, which might be expected and is indicated by Mr. Keeney's experiments. Obviously, if lump coke is substituted for charcoal conditions will be much worse. On one occasion it was necessary for the writer to be absent when one of our rectangular furnaces was being started upon coke, and the furnace man in charge failed to follow instructions *re* crushing the coke. On his return, two days later, the writer found melted oxides were being tapped from the furnace, and practically no metal. Without altering the amount, coke crushed to the proper size was used in the charge, and within a short time normal foundry iron was being produced.

Hence it is apparent that to keep slag losses within reasonable limits we must either use a considerable excess of lump coke or use crushed coke. The former method will result in an accumulation of coke in the crucible, low voltage, excessive current densities, high power consumption, poor power factor, melting roof, etc. The latter method, while presenting no metallurgical or electrical trouble, would in a shaft-type furnace make the burden so dense as to offer too much resistance to the ascending gases and would further aggravate the natural tendency of coke to "hang up," with subsequent annoying if not dangerous "slips." Crushed coke has been used satisfactorily by us in our rectangular furnaces for making all grades of foundry iron and doubtless could be utilized for making pig steel without excessive slag losses. But in furnaces of the rectangular type there is little reduction by CO in the stacks, hence the ratio of CO to CO₂ is high and the desired fuel economy is not attained.

To summarize, our experience at Heroult has indicated:

1. That though pig steel may have signal advantages over normal low-silicon pig iron for steel making, American steel makers will probably be found to be "from Missouri" and "will have to be shown."

2. That though there is considerable economy in both carbon and power in making low-silicon iron as against high-silicon iron, carbon requirements cannot be made practically to approximate the theoretically calculated amount.

3. That pig steel of 2.20 per cent. maximum carbon can be made in existing types of rectangular electric furnaces with reasonable slag losses, but with high carbon consumption as compared to shaft-type furnaces.

4. That pig steel of 2.20 per cent. maximum carbon cannot be made practically in existing types of electric shaft furnaces with carbon economy approaching theoretical figures except by accepting a high loss of iron in the slag.

5. That pig steel can be made with lower power consumption than normal low-silicon pig iron, but the saving per unit of production will be mitigated by the lower percentage of recovery.

6. That the best commercial possibilities for making pig steel with low fuel consumption and high percentage of recovery seem to be offered by the use of briquettes of concentrates and reducing material, or flue dust and reducing material, in a furnace of the shaft type.

Though not properly a part of this discussion, since Mr. Keeney mentions "Trouble was experienced at the Hardanger, Norway, electric pig-iron plant with coke as a reducing material," it may be of interest to briefly summarize our deductions from 90 days of operating our rectangular furnaces on coke.

1. Any ordinary grade of coke, by crushing it, can be used satisfactorily from both a metallurgical and an electrical standpoint to produce any normal grade of pig iron. But, depending on its porosity and crushing strength and its analysis, there is for each variety a size to be determined by experiment which will give from all standpoints best operating conditions.

2. Soft, porous cokes are to be preferred over hard, dense cokes, because the former can be fed in larger pieces and still insure homogeneity of mixture when the burden descends to the smelting zone. Further, they cost less to crush, and there is less loss in fines and flue dust because of the larger size which can be used. For ore crushed to pass a $2\frac{1}{2}$ -in. grizzly a soft, porous coke gives best results when crushed to pass a $1\frac{1}{2}$ -in. grizzly. Certain hard, dense cokes had to be crushed to pass a $\frac{3}{4}$ -in. grizzly.

3. If the coke is of the proper size when it reaches the smelting zone, operations may be carried on with as high voltage, low current densities, good power factor, and as cool a roof when using coke as when using charcoal.

4. The power consumption per unit of production is higher with coke as a reducing agent than with charcoal, but the increase is due to the extra amount of slag made. This depends on the composition and amount of ash in the coke and the amount of sulphur in the coke.

5. The carbon consumption per unit of production is higher with coke than with charcoal. This is due partly to the greater flue-dust losses from crushed coke than from lump charcoal and partly to the formation of carbides, in the high-lime, desulphurizing slag, all of which are not subsequently decomposed.

6. If electrodes are regulated to form a free-burning arc the use of coke has no bearing on the electrode consumption. If, however, electrodes are inserted in the charge to form a submerged arc, as is our practice, the consumption of electrode material is considerably greater, especially with graphite electrodes. This is due principally to the greater abrasive action of coke than charcoal, though partly also to the attack of the lime slags on the electrode material. We found the electrode loss from breakage (using 12-in. graphite electrodes) very excessive, doubtless

due in part to the greater density of the burden when using crushed coke as against lump charcoal. On this account, after carrying our experiments far enough to demonstrate the commercial feasibility of operating on coke, we have shut our furnaces down pending alterations to permit the use of 24-in. carbon electrodes, which, besides being stronger, will better resist the abrasive action of the burden.

7. With a coke running 1.25 per cent. sulphur we had little difficulty in keeping the sulphur in the pig below 0.05 per cent., and much of the iron made was under 0.03 per cent. sulphur.

8. No trouble was found in making foundry irons up to 3.50 per cent. silicon or higher if desired. In fact, on account of its high, easily reduced, siliceous ash, coke will produce a high-silicon iron somewhat more readily than charcoal.

9. Except for a somewhat more open grain, no difference has been noted thus far in the physical characteristics of iron made with coke and iron made with charcoal.

10. Crushed coke, especially when high in ash and with a large percentage of fines present, has a tendency to cause the burden to hang. This we believe would prohibit its use successfully in a furnace of the shaft type. However, in the rectangular furnaces there is an opportunity to dislodge scaffolds by barring down, if the hang is of any duration. There are several ways in which the presence of a hang up can be detected before it has become serious. A mixture of 15 to 25 per cent. of lump charcoal with the coke is quite effective in keeping the burden from hanging up.

In conclusion, we may say that coke has nothing to recommend it over charcoal for an electric-furnace reducing material excepting the difference in price. However, "black-butts," screenings, and even breeze if the dust is removed, can be mixed to make a satisfactory reducing material. As such grades of coke can usually be secured at a very attractive price this may more than offset the disadvantages of its use.

The writer apologizes for offering the above statements as "glittering generalities" unsubstantiated by definite figures, but, since the data obtained varied considerably with the chemical and physical properties of each coke tried and the size to which it was crushed, even a synopsis of this rather voluminous data is considered beyond the scope of this discussion.

Data Pertaining to Gas Cleaning at the Duquesne Blast Furnaces

Discussion of the paper of A. N. DIEHL, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 89, May, 1914, pp. 759 to 802.

A. E. MACCOUN,* Braddock, Pa.—To the practical blast-furnace man this paper is of the greatest interest. Every furnace manager in this

* Non-member.

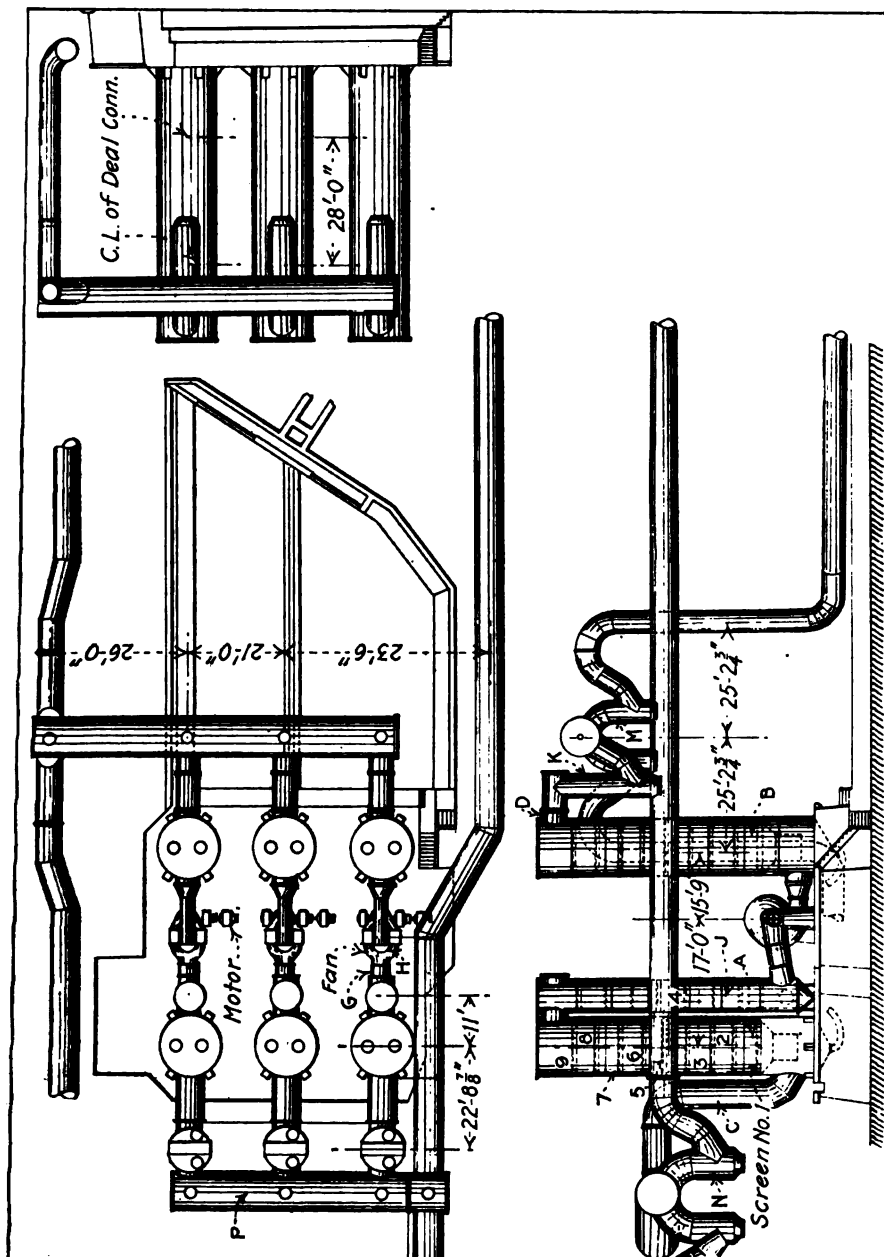


Fig. 1.—PRIMARY GAS WASHER AT EDGAR THOMPSON WORKS OF CARNEGIE STEEL CO.

country is striving to procure a better product and decrease the cost of production. One important factor in obtaining these results is the maximum utilization of the energy contained in blast-furnace gases.

At the Edgar Thomson Works of the Carnegie Steel Co. we installed, in 1906, primary washers to wash the gas supplying the stoves of four blast furnaces, and in addition to this to furnish enough primary washed gas to supply Theisen washers for three Westinghouse 3,000-h.p. gas engines. We are now installing additional primary washers to wash the gas for the stoves of all the 11 blast furnaces located at this plant. Fig. 1 gives the general outline of one of these newer gas-washing installations.

Being intimately connected with this work since its inception, as well as being familiar with the system installed at the Duquesne plant, I wish to call attention to the valuable data Mr. Diehl has submitted to this Institute, before proceeding with the discussion. These data represent extensive tests covering five years of operation, and were obtained only after a large amount of costly research work, and are therefore very valuable to any one interested in the subject. In fact, I do not know that such complete data on this subject have ever before been published.

Mr. Diehl's tests show the absolute necessity of obtaining clean gas for use in blast-furnace stoves if we expect to secure the most efficient results.

Different Systems of Gas Cleaning

At the October, 1913, meeting of the Institute Mr. Forbes read a paper describing the different systems of gas cleaning in use. There are two distinct cleaning processes, wet washing and dry cleaning.

The usual practice for a plant with several blast furnaces is to use an efficient dust catcher on each furnace and at times additional auxiliary dust catchers, or dry cleaners, so as to catch as much as possible of the heavy material sent over with the gases, as this material can be better taken care of in the dry state. From here the gas is collected in one central dirty-gas main and conducted to the additional cleaners; or it can be used directly as dirty gas in emergencies.

This arrangement tends to equalize the variations in the amounts of gas given off by the different furnaces, and better and more regular results are obtained from the cleaning plants.

Many attempts have been made to discover a practical dry-cleaning or filtering process which would clean gas sufficiently to allow it to be burned in the blast-furnace stoves with the best possible results. The sensible heat of the gas which is lost by washing amounts to about 4 per cent. of the total heat value of the gas, assuming that the unwashed gas, by the time it reaches the stove or boiler burners, is at very nearly 300° F., which is approximately its true temperature.

There are numerous objections to most of the proposed dry-cleaning

processes. Of course, with gas engines, it is absolutely necessary to cool the gases, so as to have the smallest possible engine cylinders and accurate ignition. Both of these results are best obtained with dry, cool gas.

Some have tried to combine the wet and dry systems by directing hot gases against water surfaces. Most of these systems, if they are very efficient in the removal of dirt, are objectionable, on account of the outgoing gas becoming laden with moisture.

We are at present experimenting on a cleaner of this type modeled after the smoke consumers at the New York Edison power plant; it is known as the Murray cleaner. The gas is taken through a restricted orifice and injected against a diagonal plate on which a film of water is always running; it is a very simple device and the water consumption is low. On test this device has shown up remarkably well in that the gas loses very little heat in passing through the cleaner, and the outgoing gas is remarkably clean and low in moisture. We have placed this cleaner on one of our hot-blast stoves, and will try it by actual use over a long period, and thereby demonstrate its real value as a device for this class of work.

The gas for this cleaner on the test referred to was taken off the main dirty-gas flue connecting all furnaces. Inlet gas on an average contained 15.342 grains of moisture per cubic foot and 0.9210 dust; outlet contained 4.414 grains of moisture and 0.2252 dust per cubic foot.

The Cottrell process of electrical precipitation has also been mentioned in this connection. I understand the Bethlehem Steel Co. are to try this out on a small scale. There will no doubt be other developments along this line. It is to be hoped that some safe, efficient dry-cleaning process will finally be developed.

We have had very little experience in this country with primary mechanical washers, except in utilizing this type of washer in the further refining of gas for gas-engine use, and therefore are not in a position to criticise these primary washers to any great extent. They have not been looked upon favorably, however, on account of their power requirements and the fear of large maintenance costs, etc.

Tower washers of the baffle, Zschocke, and spray types are used to a large extent for primary cleaning at our blast-furnace plants on account of their simplicity and the absence of moving parts; all these washers give excellent results. At Edgar Thomson our original washers were of the baffle type, but it was found that the baffle plates used in the towers rapidly became built up in spots with flue dust, thus causing the gas to "channel" through narrow passages, and therefore the efficiency of the baffle was not as high as it should be. Aside from this, the water did not form a very uniform film over the edges of the baffles, so that the contact of the gas and water was not intimate enough. We are therefore abandoning this type of tower washer in favor of the Diehl spray type.

Diehl Spray Type of Gas Washer

Additional advantages of this type of tower washer are:

The rotary valve is on the outside of the tower and is easily reached for repairs. Some of the other types of rotary spray washers require taking the gas off the towers to make these repairs.

The area through the washer is not as restricted as through the baffle type, as this type of washer has the advantage of the entire area of tower, whereas the inner tube in the baffle type of washer takes up about half the area, and there is no cleaning done in this tube.

The counter-current principle is always most effective for the transfer of heat. Most of the various types of tower gas washers work on this principle.

The gas is introduced into the tower from below and ascends slowly, the water descending from above, so that the coolest gas always comes in contact with the coldest water and the hottest gas with the hottest water. The Diehl type of spray carries this principle even further, as the water, by means of pressure, is sprayed up a certain distance and then descends. Mr. Diehl's paper and tests show how efficient this intermittent-spray type of washer is, in that the difference in temperature between the gas leaving the washer and the ingoing water is approximately only 5° or 6° F.

In some cases fans are used to boost the pressure; in other cases the pressure from the furnace is sufficient. At Edgar Thomson we use fans between the towers, but very little, if any, cleaning is attributed to the fans, as no water is run on them at the present time. When they were originally installed a series of Koerting sprays were used and water was run on them continuously. This caused the fan rotors to eat away and the power required to drive them was the heaviest item entering into the cost of cleaning. Spray pipes at high water pressure are connected to them twice a week for a short time only, and the sides of the casing and rotor are thoroughly flushed out.

Mr. Diehl gives an interesting example of a proposed installation of his type of tower washer for two furnaces, washing only for stoves, which depends on the simple scrubbing of the gas without fans. It can be readily seen how well this plan works out and how low the cost of construction (\$22,155) for such a plant would be for the saving effected.

Moisture

I have known of some cases where gas-washing plants have been installed and very bad results were obtained by using the gas on the hot-blast stoves. In nearly all such cases it developed that the gas was not being cooled sufficiently to precipitate the moisture and eliminate it from

the gas. The moisture must be eliminated to as great an extent as possible; the gas should be cooled below 70° F. to obtain good results.

Fig. 2 shows the moisture curves of blast-furnace gas, giving at the various temperatures the grains of water present per cubic foot of gas reduced to standard dry gas at t° and 30 in. mercury.

Curve No. 1.—So-called "saturated gas"; i.e., gas mixed with saturated aqueous vapor. At and above 212° F., its value is infinity at atmospheric pressure.

Curve No. 2.—Moisture accompanying each cubic foot of dry standard gas produced from normal unwatered furnace stock.

Curve No. 3.—Moisture accompanying each cubic foot of dry standard gas produced from normal furnace stock to which has been added 6 per

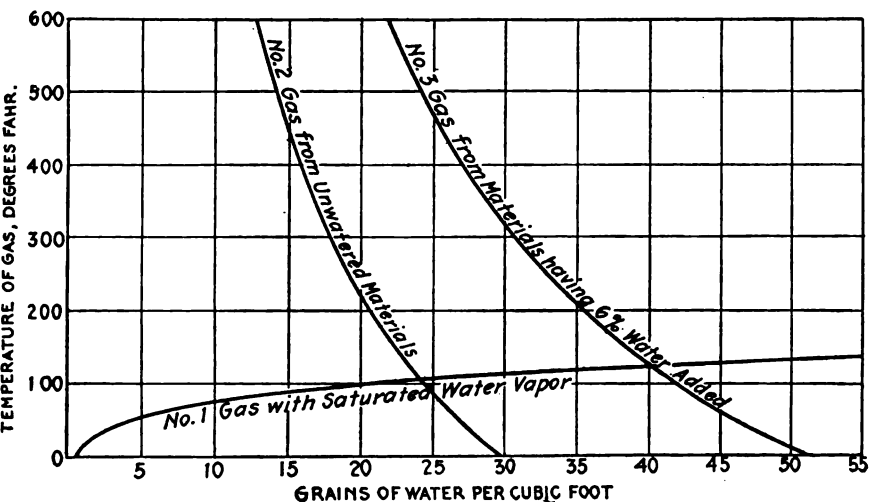


FIG. 2.—MOISTURE CONTENT OF BLAST-FURNACE GAS, CALCULATED TO DRY GAS AT t° AND 30 IN. MERCURY.

cent. of its weight of water to thoroughly moisten it and reduce the amount of flue dust.

At temperatures below the intersections of curves Nos. 2 and 3 with the "saturation" curve, No. 1, a considerable amount of the accompanying moisture of these gases cannot be held as true vapor, and is either held to some extent in a supersaturated state, or is precipitated out.

Study of these curves shows that the best results cannot be obtained from washed gas if it is not cooled sufficiently to obtain low moisture content.

One of the most important considerations in the use of clean gas is to be able to mix the proper amount of air and gas at the stove burner and thereby obtain perfect combustion. Mr. Diehl partly shows the effi-

ciency of dry cool gas in his tables, but he fails to state that it is impossible to burn dirty gas efficiently in any case, because if it is attempted the high temperature of combustion melts some of the flue dust carried by the gas down into the well of the stove, filling it up, and gradually forming a big solid lump of sinter, which it is nearly impossible to remove without the expenditure of a large amount of labor and a great loss of time. To avoid this when using dirty gas a low temperature of combustion is purposely sought in the combustion chamber of the stove at the expense of loss in efficiency.

Nitrogen Compounds

It is fair to ignore the formation of nitrogen compounds in the gas, as Mr. Diehl's experiments have failed to even find traces of these compounds upon special research analyses. The coloration observed at the washers could be readily produced by extremely minute traces of ferrocyanides.

The Efficiency of Hot-Blast Stoves

The importance of high heats is fully realized in modern blast-furnace practice and we wish to obtain these higher heats by utilizing blast-furnace gas as efficiently as possible, as the demand for the gas remaining after supplying the stoves and blowing engines is becoming greater all the time. Steel works are endeavoring to turn out better products, which necessarily require more refining and work, and an increasing amount of power. Blast-furnace gases have a much greater fuel value than an amount of coal containing an equal number of B.t.u.'s, as they require no expensive handling and make an ideal fuel for nearly any use around a steel plant.

The stove tests submitted by Mr. Diehl show conclusively the better efficiencies obtained by the use of washed gas, but in comparing these results it must be borne in mind that the stoves burning the clean gas on which tests were made were not originally built for this gas. They should have had much smaller checker openings; thereby much greater heating surface would have been obtained and their efficiency greatly increased over the figures presented. With dirty gas, where it is not possible to use small checkers successfully, the use of a 9-in. checker opening for stoves is common. On our newer stoves, adapted for clean gas, the checker openings have been decreased to 6 in., and in some cases to 5 in., the heating surface gained being approximately 30 per cent.

In this connection I wish to refer to another point that has not been brought out fully enough. With a *thoroughly cleaned stove* good results can be obtained for a short time with dirty gas.

Take a blast-furnace that has four clean stoves and uses dirty gas. At first excellent heats are obtained; gradually the checker openings begin to

clog with fine dust; it bridges over the top of the checkers, the heats begin to run down quickly, and the efficiency drops. I have known of such cases where the decrease in stove efficiency has been over 12 per cent. The coke consumption of the furnace then increases, the product decreases and the furnace practice becomes very poor. Then one stove at a time is taken off for cleaning, the results being still poor until the stoves are finally all cleaned, and we have the four stoves again in use. During this time the furnace has become accustomed to low heats, and it usually takes some time to bring it back to the former good practice. When this is finally accomplished it is not long before the operation has to be repeated and the stoves cleaned again. Therefore the stove tests do not represent at all the actual gain in blast-furnace practice obtained from the use of clean gas, as the greatest gain is having a regular, dependable hot-blast temperature and the elimination of as many variable furnace conditions as possible.

In addition to the above, the use of clean gas represents a large saving in repairs and renewals required on the stove linings, etc.; also on hot-blast valves and seats, pipes, tuyères, etc., as these are very badly cut and costly repairs are continually required when dirty gas is used.

We frequently find the bustle pipe half filled up on one side of a furnace that has been on for any length of time, which also tends to irregular distribution of blast.

All of these important factors must be considered in arriving at the true gain represented by the use of clean gas for stoves.

There are further economies to be made which will also increase the efficiencies obtained in hot-blast stoves, and which I wish to mention in this connection:

1. The tables Mr. Diehl has submitted and our own tests show the heat loss in flue gases from our hot-blast stoves to be approximately 20 per cent. This heat should be utilized to add to the temperature of the cold blast before it enters the stove to be further heated. I understand this is done at some of the German blast-furnace plants.

2. Radiation losses, figuring roughly on the exposed surface of the stove shell, vary from 7 to 12 per cent. This is hard to arrive at accurately, as no radiation formulas are available to suit this particular condition. By more efficient insulation between the brickwork and the shell a saving might be effected here. At present we have a 2-in. space filled with loam mixed with granulated blast-furnace cinder.

3. A more efficient burner will be devised to get as high a combustion temperature as possible in the lower part of the combustion chamber, and thus shorten the flame and confine all the combustion to the combustion chamber, so that the hot products of combustion alone will pass through the checker work. In most cases the combustion is not completed at the top of the checkers.

Savings Through Gas Cleaning on Stoves

At some blast-furnace plants hard ores or ores of good physical structure are used to a large extent in the burden. This, combined with slow driving, will greatly reduce the usual troubles in stoves caused by the flue dust carried in the gas; but such cases are rare, and do not apply to modern blast-furnaces using a large percentage of Mesaba ore in their burdens, which furnaces we are here considering.

Mr. Diehl gives the cost of primary cleaning gas for stoves per 100,000 cu. ft. as \$0.06329, or \$0.03252 per ton of iron, which would be very much reduced with a new installation arranged so that not so much electrical power would be required for operating the fans.

The saving on clean gas over dirty gas on stoves with regard to the cost of iron is given as \$0.16108 per ton of iron. This I consider very conservative; in fact, I consider that \$0.22 per ton of iron is a more representative figure for this saving for a modern blast-furnace using a large percentage of Mesaba ore.

Savings Through Gas Cleaning for Boilers

No attempt has been made at the Edgar Thomson Works to clean gas for boilers except on tests, as the boilers at our plant give very little trouble from the dust in the gas building up in passages and tubes, and what does gather there is easily blown away by steam from a hose fitted with a 1-in. nozzle. We have made boiler tests using clean gas, but could not see enough saving in its use for this purpose to warrant the investment. The saving depends on what type of boiler is used.

F. LOUIS GRAMMER, Leesburg, Va.—I ask Mr. Diehl whether he has ever analyzed blast-furnace gas when the furnace was hanging—I do not mean hanging around the bosh, but hanging due to throttle near the top—or whether he ever analyzed the gas immediately preceding such a slip. I remember that in blowing in a furnace at Cleveland, owing to congested traffic conditions, the locomotive was not able to get around to dump the dust catcher. The result was that our blast pressure rose. consequent to back gas pressure, to some abnormal figure, such as 26 lb., and yet everything seemed to be all right. There was a good deal of concern over the situation, and no one understood what the cause of the trouble was until it was discovered that the dust catcher had not been dumped. When the dust catcher was dumped the pressure dropped to about 14 lb.

I have an idea, which has not been fully tested—I only offer it as a suggestion—that possibly a throttling of the escaping gases would tend to make for excessive carbon deposition in the upper part of the stack, and as a preliminary to such a congestion this excess would be reflected in the

changed composition of the gas. This subject received a good deal of attention at my hands in 1904, when we had available nothing like such an elaborate presentation of the subject as Mr. Diehl has just given. I make that suggestion as of possible interest in case further investigations along this line are made. Personally, I believe that the throttling of the downcomer would tend to decompose the escaping gases so that the carbon decomposition would be increased. My impression is that "spilling" occurs more frequently on dirty flues.

A. N. DIEHL.—I do not recall any analyses that we have purposely taken before and after slips, with the object of determining the point which Mr. Grammer has in mind. As mentioned in my original paper, we did at one time take some analyses of gas directly after a slip to account for the back-firing of our gas engines. We suspected hydrogen in this gas due to moisture in the stock, and therefore prepared ourselves for this test. We have often noticed when a furnace becomes very tight and is hanging continuously, that the gas becomes very sharp, which can easily be noticed as it is burned in the stoves and boilers. This might account for the condition such as Mr. Grammer is referring to. I will be glad to make some experiments along these lines to obtain further information in the matter.

J. W. RICHARDS, So. Bethlehem, Pa.—I ask Mr. Diehl whether, in the dry dust or washing water which may have been used, there is any concentration of potassium salts; that is, whether any of these have been detected in the dust, or in the water used for cooling.

A. N. DIEHL.—I have shown in my paper analyses of the deposits sublimated on the ports and piston heads of gas engines, indicating a content of alkalies equal to 30 to 40 per cent. The water from the scrubbers is alkaline to a certain extent and we find soda and potassa present. We often find quite heavy deposits of cyanides and alkalies in the brick lining of the blast furnaces, some analyses of which show about 40 per cent. These deposits seem to permeate the brick.

J. W. RICHARDS.—I asked the question because Mr. Eckles of Washington had patented the use of potassium-iron silicates in the blast furnace, in order to collect the potassium which would be volatilized and caught in the dust coming from the furnace. I wanted to know to what extent that worked out in ordinary practice.

A. N. DIEHL.—On the fans of our Theisens we obtain deposits also of alkalies, but the quantity is comparatively small considering the large quantity of gas passing through. I do not have any figures to show the quantity obtained in this manner.

Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes

Discussion of the paper of HEINRICH J. FREYN, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 88, April, 1914, pp. 665 to 693.

RICHARD LAMB, New York, N. Y.—It is a fact that in Europe, especially in Germany, they have perfected the internal-combustion engines designed to use by-product coke-oven gas, and it is not uncommon to find as high as 5,000-h.p. units at work very successfully. In fact, I think the breakdowns and stops for repairs on these large engines in Germany are remarkably few. I have heard of a large engine running for a whole year without a shutdown for repairs.

In Germany they have had by-product coke-oven plants longer than we have had them here. In this country they have been installed mostly for use by steel plants, and possibly in three or four cases they have been built in connection with already established gas plants. These two kinds of by-product plants are radically different. One seeks to make coke and the other seeks to make gas as its principal product. If you try to establish a plant to furnish both gas and coke, it is a difficult matter to get suitable coal. A coal of about 28 to 30 per cent. volatile matter is about what is required; then you will get gas in quantity that will not make a soft coke, and the coke will be of a texture that can be sold to be used in place of anthracite coal. Where gas companies have established plants, the main object being to secure a large quantity of gas, they use a coal which makes soft coke. On the other hand, the steel companies are after a hard coke, and the gas production is but a small consideration and up to the present time as a rule is allowed to go to waste. The waste of gas at the steel manufacturing plants has been immense.

I was told that no one of the steel companies is using coke-oven gas for metallurgical purposes. However, I hear that the Krupp works in Germany is using the pure coke-oven gas in its open-hearth furnaces. I am told that, with a slight mixture of the blast-furnace gas with coke-oven gas, the Old Dominion Co. of Nova Scotia is using it very successfully in its open-hearth furnaces. The difficulty in using the coke-oven gas is due to the excessive heat it generates. Steel furnaces designed to use blast-furnace gas are not built to withstand the extra heat of the coke-oven gas, but there is no reason why furnaces cannot be designed to withstand this extra heat. By-product coke-oven gas has about 500 B.t.u. per cubic foot, while the blast-furnace gas has about 90 or 91 B.t.u. per cubic foot.

I have noted that in attempts in this country to use this gas for engines the failures have been largely due to not providing sufficient cooling capacity. As far as I have been able to learn, the only engines in

this country that have been built to use coke-oven gas are the two at Lebanon, Pa., and they did not meet the guarantee. The Nuremburg gas engine of Germany uses 8,700 B.t.u. per cubic foot of gas in producing 1 b.h.p. at full capacity load and about 9,300 B.t.u. for half load. In this country the open-hearth engines in Lebanon produced 1 b.h.p. with 11,000 B.t.u. per cubic foot and one-half horse-power with 14,000 B.t.u. per cubic foot. I do not recall now what their guarantee was, but the results were not acceptable, although the engine continued in operation. They have had trouble with the heating. Unless the United States Steel Corporation has put in a plant at Gary, which it intended to do, I do not think there are any other plants using strictly coke-oven gas in this country. I found, however, that the Westinghouse, the Snow, and the Allis-Chalmers companies are willing to make engines of large capacity and guarantee them at 10,000 B.t.u. per cubic foot per brake horse-power. You can buy engines on a guarantee of not to exceed 8,700 B.t.u. per cubic foot per brake horse power on full load, from the Nuremburg works of Germany. The Snow engine, I believe, could be made to show an efficiency of 8,700 B.t.u. per brake horse power on full load.

In establishing a by-product coke-oven plant to be operated independently of going steel or gas works, the question of what to do with the gas becomes a very serious one. If sold to gas companies you can get about 10c. per 1,000 cu. ft. for it, provided you will rectify it and bring it up to 18 to 21 candle power. The rectifying costs about 0.1c. per 1,000 cu. ft.; in other words, it brings the price for the gas down to about 9.9c. per 1,000 cu. ft., and a penalty is imposed if the quality of the gas goes slightly below the standard.

There are some very great advantages in selling the gas, independently of gas engines. You can transmit gas over long distances at very small cost, and you can sometimes get a market for it for fuel purposes, without rectifying it; and the requirement for the B.t.u. value will not be so great as if sold for illuminating purposes. Gas companies call for not less than 500 B.t.u., and demand a material penalty for going below that amount. It requires considerable operating care in a by-product coke-oven plant to keep the gas up to the standard, and great care in the purchase of the proper kind of coal is necessary in order to get satisfactory results. You should have coal with not less than 13,000 B.t.u. in order to get 500 B.t.u. in the gas. In negotiating with a very large concern, that could take a large amount of gas, several million feet per day, I studied with their engineers the relative value of gas and coal for fuel purposes. With coal of about 13,000 B.t.u. costing \$1.75 per ton, if I remember correctly, they figured that to make it advantageous to them, so as to justify their abandoning the coal and using the gas, the only contract that they could afford to make would be 7c. per 1,000 cu. ft. In computing the saving, the cost of handling ashes was not considered. I am convinced that

at no distant date there will be many plants installed in this country to generate electricity with engines operated with coke-oven gas. Coal now known as coking coal—viz., that suitable for use in beehive ovens—is not necessary in making coke in by-product plants. The fine or pulverized coal, which is the cheapest, is needed in the latter ovens, and mixtures of different coals produce satisfactory results where neither by itself would be suitable. Ten per cent. more coke can be made out of a given quantity of coal with a by-product coke-oven than with the beehive oven. In the production of electricity there is a very great advantage in having a gas-storage tank. As soon as the service is cut off, the gas engine can be stopped and the tank stores the gas not used. In this particular it is like a water-power plant with a water-storage dam. In computing the total cost of the plant to generate electricity, I found that it compared very favorably with water power. At the same time that I was designing the coke plant, I was also designing a hydro-electric plant to generate 39,000 h.p. The cost, including amortization charges and interest on the plant and all costs, was 3.1c. per kilowatt-hour for water power and 3.2c. per kilowatt-hour for the gas, and the plant is for New York City.

I figured the power factor pretty high in the case of the gas plant, as we had to make only about 700 tons of coke per day, which gives us about 5,000 h.p. I figured on the basis of 70 per cent. power factor, because we were in a place where we could get considerable night service and some 20-hr. service, but that shows there is quite a conformity in the cost between hydro-electric power and coke-oven generated electricity. It also shows that with coal at \$3.98 you can generate electricity for about the same cost as you can at a water-power plant, and that right in the very heart of large markets, where the transmission lines would be very short. For some time in Germany they have been making electricity at the mines and distributing it for 100 miles or more, and some of the companies engaged in this work have paid handsome dividends. In this country there are no plants making electricity from coke-oven gas, and there are billions of feet of gas going into the air, as dead waste, every day. In investigating the plant of the U. S. Steel Corporation, I mentioned that fact to my clients, and they tried to buy the wasted power. I understand that the company at once undertook to look into the matter, and they are going to generate electricity and distribute it, if they have not already begun to do so. In Alabama, at Birmingham, there are millions of cubic feet of gas per day going to waste from the by-product coke-oven plants, but as it happens there is a surplus of power at that point. There are two big hydro-electric companies being built, the works of one being finished and the works of another well under way, and there is still another which may bring its power from Tennessee into the Birmingham district, and right there is all the coal

they can use for centuries to come. Nevertheless it is a shame that this gas is being wasted. It is claimed that there are about 400,000 h.p. now generated with coal within a distance of 100 miles of Birmingham. The cheap power generated from gas and water now going to waste will ultimately supplant the present coal-generated power, and we can look for similar results from less-favored districts of the country.

A. E. MACCOUN, Braddock, Pa.—I wish to ask Mr. Freyn whether he has any figures on the cost of operating a blast furnace as a gas producer. I think this data would be interesting, considering the differential between the value of coal and coke, and also I would ask whether it is a paying proposition to run a blast furnace as a gas producer.

HEINRICH J. FREYN.—I am sorry I cannot give Mr. Maccoun any figures. Even if I had any, I could not give them, but I have none. I understand, however, that the cost of making blast-furnace gas compares very favorably with the cost of making producer gas; in other words, it is found at the gas plant that the use of producer gas made in one of the plants at the gas furnaces is distinctly a financially advantageous proposition, and this is very interesting from another angle. If you consider that the steel plant at Gary is dependent entirely on the use of gas power, and in view of the fact that a number of blast furnaces had to be blown out for well-known reasons, and that in addition the power plants had a little electric power to supply to industries in the vicinity, you can realize how important it is for the plant at Gary to have a solution of this problem. In spite of the fact that gas furnaces may be blown out in producing pig iron and steel, yet one of these furnaces using cheap coal can produce sufficient gas of such a quality that the power plant can be kept in operation just the same.

RICHARD LAMB.—An internal gas engine, as compared with the turbine steam engine, has a greater efficiency by 30 per cent.; in other words, an internal gas engine has 30 per cent. greater efficiency than the turbine.

J. E. JOHNSON, JR., New York, N. Y.—Do I understand you run the furnace as a slagging gas producer, and charge it with bad coke, bituminous coal, and other cats and dogs?

HEINRICH J. FREYN.—In South Chicago, where open-hearth slag was used, the flux necessarily resulted in a certain amount of iron, 60 or 70 tons per day, from a 500-ton blast-furnace. I understand that the pig iron so produced contains a great deal of manganese, and the credit for this particular iron has kept the cost of gas down.

J. E. JOHNSON, JR.—The statement that one of these plants can compete with water power is, unfortunately, I am afraid, not correct. There are a good many chemical industries that require a great deal of power, and it seems as though it was the easiest thing in the world to hitch on to the blast-furnace to get power and make a secondary profit

from the waste gas, but one of these processes, which I happen to know about, figures that it must have power at \$14 or better, in order to live. Very extensive figures have been published by Mr. Freyn, Mr. Bacon, Mr. Diehl and others, in connection with the development of gas power by blast furnaces, and these show that the actual cost of labor, maintenance, interest charges, and depreciation, etc., the absolute rock bottom charges, without any allowance for the value of the fuel at all, figure about 0.3c. per horse power-hour, and if you take 8,000 hr. to the year, that figures at about \$24, which is prohibitive.

HEINRICH J. FREYN.—It should be in kilowatt-hours.

J. E. JOHNSON, JR.—That would reduce it below \$20, but with a reasonable allowance for fuel it is still far beyond the price which many electro-chemical industries must have in order to live.

RICHARD MOLDENKE, Watchung, N. J.—In November last I saw in Germany the operation of the latest type of "slagging producer" made by Mr. Servais, and was much interested in noting the recovery of considerable quantities of ferro-silicon with the slag as it was tapped periodically. This material was the result of a reduction of the iron content in the ash of the coke breeze used, as well as from a considerable percentage of flue dust added to the fuel charges. The ferro-silicon ran about 12 per cent. in silicon, and should form a nice by-product to help bear the cost of gas production.

J. W. RICHARDS, So. Bethlehem, Pa.—There were some papers read at a recent joint meeting of the American Institute of Electrical Engineers, the American Society of Mechanical Engineers, and the American Electrochemical Society, showing that power was being generated in the vicinity of New York City at 0.4c. per kilowatt-hour, in the large copper refineries, this being the total inclusive cost of the power.

HEINRICH J. FREYN.—That is easy, without overhead charges, depreciation and interest.

J. W. RICHARDS.—I understand that all that was included.

EDWARD T. CHILD.—I ask whether Mr. Freyn's contention would not operate more against the gas engine than against the turbine plant or steam plant? Is not the overhead greater in the case of the blast-furnace plant?

HEINRICH J. FREYN.—Yes, a little.

EDWARD T. CHILD.—Mr. Lamb's proposition was based on the engine itself without reference to the overhead. In the last plant I went through they had four large gas engines and five turbines. All the turbines were running, one-half of one engine was running, and the other three and one-half sets were out of service on that particular day.

J. W. RICHARDS.—I would ask Mr. Freyn whether this statement I find in his paper is quite correct: "Coke-oven gas used for power

purposes, or as fuel in a steel plant, does not require any illuminating properties." I have had the impression that the illuminating property of gas was a very valuable property, increasing the radiation of heat from the gas, while it was being burned in the open-hearth furnace. For the same reason, if you extract the tar from producer gas, the gas is not nearly so efficient when used in the furnace, and I should suppose that the same would be true of coke-oven gas.

HEINRICH J. FREYN.—I must confess that I am not a metallurgical engineer. I know that the principal objection to the use of coke-oven gas in the open hearth was the lack of illuminating qualities, because the operator could not see the flame. If the tar that is recovered in the by-product plant were charged back into the coke-oven plant, it would be possible to recover the benzol, which is more valuable as a motor fuel to-day than gasoline, and that would be more advantageous than to burn it up in the open-hearth furnace.

Scientific Installations for the Economical Burning of Liquid Fuel of Any Specific Gravity

Discussion of the paper of WILLIAM NEWTON BEST, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 86, February, 1914, pp. 267 to 277.

E. H. HAMILTON, West Norfolk, Va.—I think I was about the first to use oil in large reverberators, for copper smelting, and our records show that we got better results generally per B.t.u. from the oil than from any other fuel. It did away with all fire-box losses from below the grates, and the boilers could be regulated to suit any purpose. Sometimes we ran the furnace for steam, and sometimes for oil. Incidentally all the time it was for smelting. We found the efficiency of the smelting depended upon the highest point of temperature, and we had to feed the charge at the hottest point of the furnace, which was in a different position from a furnace fired by any other kind of fuel. We moved the charge to suit the point of high temperature.

H. O. HOFMAN, Boston, Mass.—In regard to the use of fuel oil, I think it would be of interest if Mr. Best made some remarks upon the use of steam and air as atomizers. As I recall it, in perusing the paper, this subject has not been touched upon. In smelting sulphide copper ores in a reverberatory furnace, our metallurgists have tried steam, low-pressure air, and high-pressure air for atomizing; all have given up steam and low-pressure air and use air at a pressure of 12 to 14 lb.

In reference to the remarks made by the last speaker about the high efficiency of oil, I may recall that Mr. Mathewson, in his paper on the use of bituminous coal in firing reverberatory matting furnaces, found that

he did not derive any advantage in increasing the length of his furnaces beyond 100 ft. It has been found that you can do this in firing with oil; in fact, Mr. Sørensen, at the works of the Nevada Consolidated Co., has in operation a furnace 130 ft. in length; he believes that the greatest permissible length is 140 ft. This may be due to two causes: One is the high calorific power of the oil, and the second, that oil is fed continuously; we have continuous feeding and firing. With the modern method of supplying coal to the copper reverberatory matting furnace, in which coal is fed every 10 or 15 min. from the top of the furnace, there is an interrupted feed. With every addition of coal the fire must cool down. For this reason the temperature in the oil-fired furnace is higher than in the coal-fired, and the heat extends for a longer distance, which makes the longer furnace possible. The smelting power is also greater. Wherever it is economically possible, oil has replaced bituminous coal for firing. A good example of this rivalry is shown at the Garfield smelter, where bituminous coal and oil were used at the same time in two furnaces and where oil has replaced bituminous coal.

E. W. PARKER, Washington, D. C.—Professor Hofman might give us some information regarding the use of mechanical stokers in metallurgical furnaces.

PROFESSOR HOFMAN.—Some experiments have been made in using a mechanical stoker in firing a reverberatory furnace treating native copper ore at Lake Superior, but no report of the results has been made public as yet.

E. H. HAMILTON.—When I was at the Pueblo smelter we fired our mechanical roasters with oil, and the Standard Oil Co. kept putting up the price, so we put in mechanical stokers, and they worked very efficiently. The stoker was called the American stoker, and with certain coals it worked very well.

KARL NIBECKER, Youngstown, Ohio.—I notice that Mr. Best gives some values for the amount of oil used in various heating furnaces which would be equivalent to 1 ton of coal burned in the same type of furnace. I would like to know whether these values are compared with coal burned in the furnaces, or in the form of gas. As powdered coal appears to be one of the most economical methods of burning the fuel for heating furnaces, can Mr. Best give us any comparisons of the amount of coal required in a heating furnace, as compared to a ton of powdered coal? As there at present seems to be some doubt about the value of powdered coal in a boiler, I do not feel that that phase of the comparison would be of such vital interest."

If Mr. Best has any comparative figures along these lines I feel that they would be of considerable value to operators of heating furnaces.

LEONARD WALDO, New York, N. Y.—With fuel oil things can be done and quickly done that are harder to do in any other way. The heat

gradient can be pushed, instead of lagging along for hours. It can almost be put on a straight line. That comes from the extremely efficient way in which the oil can be burned.

In regard to the calorific value, the B.t.u. per pound, of 18,493, given for Mexican oil, I would say that many samples increase this to 19,200, from which follows this corollary: That gallon per gallon it is higher than any other oils, but pound for pound is somewhat less, as Mexican crude oils have a high specific gravity. In regard to the details through this paper of the burning apparatus for burning oil, no one who has not experienced the burning in the field can know the value of these suggestions. It is necessary that you should lead the horse to water, before he takes his drink, and in regard to these fuel oils, at 2,000 seconds on the viscometer, ordinary temperature of outside air, the oils may refuse to move, even though your pumps may be forced to the point where the gaskets break out and your pipes split. Therefore, we owe great thanks to the gentleman who has spoken, for the careful way in which he has set out the practical details. I do not know that they occur in any other literature.

In regard to flash tests and the fire risks of oil, the literature is based on the old flash and combustion points, which have crept in from the petroleum series, and when a Mexican crude oil comes with a flash point of 125° we feel that to use that on our work would insure our early departure from this world. But these flash points are determined, and the amount of actual vapor with 125° flash point, in perhaps 1 or 2 per cent. of the great bulk of the oil. Only 1 or 2 per cent. of it has that flash point. It will make its first flash, and the flash point is recorded as being 125°, whereas that is the flash point of but a very small fraction of the oil, and the total amount of vapor lying on the top of that oil is so slight that the oil will flash like lycopodium in a theater.

I was standing by some laborers who had a fear of the oil, and one said, "I am going to try the stuff. Come with me." He filled a galvanized bucket with the oil, got a light at the furnace, and plunged it in the bucket. Nothing happened, only the light was put out. The same conditions govern the experiments at sea. In the case of the "Volturno," the waves were covered, and if it had a dangerous point it would have been shown there. We must revise our whole idea of flash points when they relate to a very small fraction of a mass of the oil, the total sum of which is insufficient to raise the temperature of the whole body above a fraction of a degree. If you have a large tank of oil, and you are carrying a 10-ton mass to be cooled or quenched in oil, sometimes the heat of the large mass will raise the temperature of the mass of oil to a dangerous point. But if you have a trundle bed and a roof shoved out and covering the tank, the whole conflagration is at once extinguished, and you have a safe use of these oils. Where you have a quick extinguisher for the oil, it is safe.

Now, in regard to the use of air or steam for atomization: In the furnaces, it does not seem to make very much difference which is used, as far as the atomization of the oil is concerned, but there are certain indirect advantages in using air. I cannot refer to them here, as they involve the chemistry of combustion; but steam is used at some important furnaces for atomization, and it is very effective, but in those cases the air supply is carefully regulated from other sources.

Our literature is lacking most in viscosities at varying temperatures. Fortunately for us, but unfortunately for our predictions, the viscosity curve for these heavy oils must be considered. It increases rapidly after it reaches 40° or 45° F., and then it goes up very rapidly. I would like to suggest to the Bureau of Mines that we desire additional information in that field.

W. N. BEST.—In relation to the atomization of the oil, either steam or air may be used to atomize the oil. When using heavy oil in large furnaces it is necessary to use air or steam at 80 lb. pressure in order to thoroughly atomize the required quantity of oil and distribute the heat, but small furnaces require only about 20 lb. pressure. In either case about forty-nine-fiftieths of the air requisite for combustion passes in through the air nozzle at 3 oz. pressure, and therefore you have the strictest possible economy by using a small quantity of air or steam to atomize the oil and distribute the heat and admitting the additional air required through a nozzle at low pressure. Relative to the economy of using steam and air at 80 lb. as atomizer, air will show a saving over steam of over 12 per cent. in fuel, but it requires 8 per cent. of the oil saved to compress the air; therefore you have a net saving by the use of air over steam of 4 per cent. in fuel.

Now, answering the question relative to pulverized coal: The figures given relate to bituminous coal, and it is referred to in the paper as bituminous coal. I have had no experience in running tests against pulverized coal.

The data given in this paper are the result of 26 years' hard work, and thousands of tests, and there is not a week but men come into my office and say, "Mr. Best, we want our boilers equipped; we have gotten on so successfully with our furnaces that we want the boilers equipped with oil also." But I have refused to do it, as I know the additional quantity of oil required to represent a ton of coal in the boilers, whereas, in furnaces a 60 per cent. economy over coal is often effected.

E. GYBON SPILSBURY, New York, N. Y.—Dr. Waldo seems to place emphasis on the fact that there is little or no difference in the use of steam instead of air for atomizing the oil used either for melting or heating iron or steel. I would like to strike a note of warning against the use of steam instead of air as a means of introducing liquid fuel into iron-melting or heating furnaces. An experience I had some years ago demonstrated the

danger of the use of water vapor for this purpose. About 1887-88, the Bethlehem Iron Co. introduced the Archer water-gas producers for their billet-heating furnaces, and as it turned out later the results were very disastrous. At about the same time I adopted the same method for heating furnaces at the works of the Trenton Iron Co. The billets I had been purchasing from the Bethlehem Iron Co. had always been extremely soft and ductile; in fact, the company had made a great reputation for itself on this particular brand of open-hearth billets. All at once the character of the steel changed entirely and it became so brittle as to make it absolutely worthless. Investigations carried on both by our company and the Bethlehem Iron Co. failed to show any cause for this remarkable change of character; the records of the heat—taken from the open hearth where the billets were made—showed no change from former practice, and the analyses of the material were exactly the same as they had been for years previous. Nothing that we could do in the way of annealing or slow cooling seemed to have any effect on the material. Finally, Porter Shimer, after making a number of very exhaustive analyses, called my attention to the fact of a larger evolution of hydrogen gas in dissolving this material than he had noticed in other steels. In tracing the matter back it was found that the beginning of the trouble was coincident with the introduction of the Archer producers, and it was entirely remedied when ordinary producer gas was used in the heating furnaces instead of that produced by the Archer method. It was evident that the brittleness of the steel was due to the absorption of hydrogen by the hot steel the moment it came in contact with this hydrogen in its nascent state. While we found that annealing in the ordinary pot method failed to reduce or drive off the hydrogen, it was demonstrated later that a gradual occlusion of the hydrogen took place, so that after some few months' exposure to the air billets and rods would recover their ductility almost to the same extent as similar material which had not been exposed to the hydrogen effect.

DAVID T. DAY, Washington, D. C.—I would like to ask Mr. Best what he means by the vaporization point; and I would like to ask Mr. Waldo what oil was on the tank steamer that blew up in the river, where it came from, and what its flash point was.

W. N. BEST.—The point at which the oil when heated casts off visible vapors. For example, the Mexican oil vaporizes at 175° F. or thereabouts and you ordinarily carry a temperature of about 160° F. on the oil line to burners. If you carry it at 180° F. there will be noticeable puffs of flame from the burner, simply because the vapor and the oil pass intermittently out of the burner.

MR. DAY.—I only want to know how you distinguish that from the ordinary flash point.

W. N. BEST.—I do not deal with flash points in my paper. There are

two different ways of distinguishing the vaporizing point: one is by the color of the flame in the furnace (one moment there being perfect combustion and the next imperfect combustion), and the other is by the unsteady noise while the burner is in operation.

MR. DAY.—Then how would you determine the vaporization point, the point at which it vaporizes?

W. N. BEST.—That is determined by heating it with steam in a small receptacle, as shown in Fig. 1. The temperature of steam at 100 lb. is

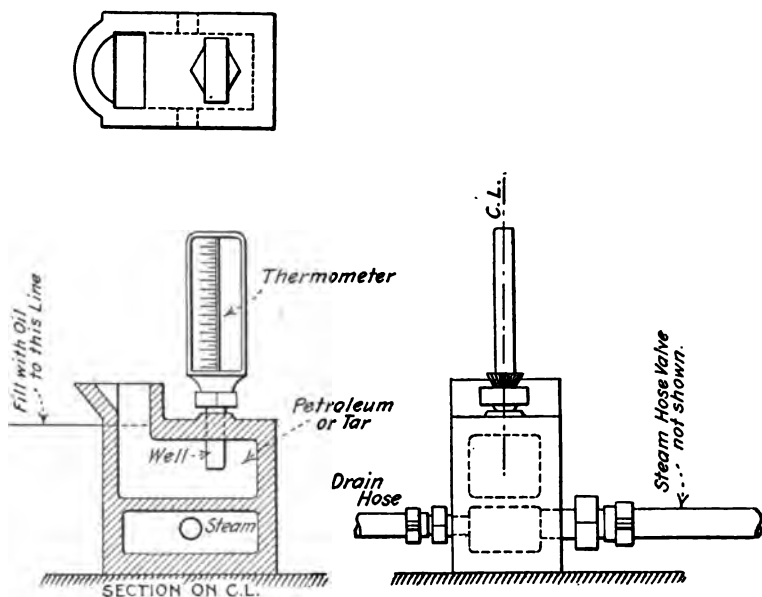


FIG. 1.—RETORT FOR DETERMINING VAPORIZATION POINT OF PETROLEUM.

338° F. Soon perceptible vapors will arise from the oil in the receptacle, and as soon as that occurs you have the temperature at which the oil vaporizes.

C. W. WASHBURNE, Washington, D. C.—Isn't it true that all hydrocarbons have a vapor pressure, and that there is some vapor escaping at all pressures; and there can be no scientific point determined? It must be some particular point at which a great deal of vapor is given off, and I should think that was variable.

W. N. BEST.—It is true that some imperceptible vapor is constantly passing from the oil, but that to which I refer is the vapor formed in large quantities when the oil is heated in order to reduce its viscosity, as described in my article. When the oil is heated beyond a certain point, vapor forms and causes an intermittent flow of oil through the burner. The oil should be heated to just below the temperature at which vapor in large quantities is generated and that temperature can be maintained

within 5° or 6° by the operator. I now refer to heavy oil, for light fuel oil does not require heating.

E. H. HAMILTON.—We used to burn 300 barrels a day, and when we sampled it with the same accuracy that we sampled gold and silver and copper ores, I found the B.t.u.'s were lower than the published statements. This was probably due to water in suspension, but it is a fact that when we tested those samples for B.t.u., we did not get as high results as we were supposed to get.

LEONARD WALDO.—Vapor is generated from all oils. I think we will see a complete change in our literature on the subject of flash tests of oil.

Now, Mr. Spilsbury has asked a question which I thought I had concealed under the remark that we would not touch on the chemistry of the burner. But he raises the question. Atomizing oil has a two-fold function; one is to atomize the oil, get it into a smaller molecule and deliver it in small quantities at distances of 40 or 60 or 100 ft., and that takes pressure. In many plants they prefer to substitute steam for the purpose, for while it is easy to get steam over 100 to 120 lb., it is hard to get air, and therefore while the contention of the members is quite true, there are steel makers to-day who adhere to the use of steam against this advice, and with as good result as in its use on the locomotive. I personally use air, and will go to any pains to get the air for that purpose.

In regard to that explosion: Explosions will take place in an oil tank when it is relatively empty. It is only necessary to have a small quantity of oil in the tank, and a careless workman who insists on lighting his pipe in the bottom of the tank, and all the data of your experiment can be measured afterward, and the original proportions for it are there.

Now, in these explosions at sea, I tried to get the data. In regard to the one cited, I have no data as to that. But in cases at sea where bad explosions have taken place from the flare-back of a flame under the boiler, they would have been avoided if the oil had been used more economically, and burned in the cylinder of the engine. Oil properly used and stored is statistically a safer fuel than coal.

Should the Apex Law be Now Repealed?

Discussion of the paper of CHARLES H. SHAMEL, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 88, April, 1914, pp. 635 to 644.

R. W. RAYMOND, New York, N.Y.—I wish to thank Mr. Shamel for his vigorous presentation of that side of the question before us, on which, if one should judge from the contributions here presented, he stands alone. For, in the first place, he does not in fact stand alone, but is backed by a host of citizens in the States affected by the "apex law," who have often exhibited heretofore their opposition to its repeal by the pressure they have

brought to bear upon their Congressmen, some of whom have been retired to private life, apparently as a penalty for supporting that reform. Moreover, many intelligent and public-spirited persons have ably supported his view in other publications than our *Transactions*; and I may say without discourtesy that the most complete refutation of Mr. Shamel's argument would leave other and stronger presentations of his case yet to be considered. But if he did stand alone, he would still be not only entitled, but cordially welcome, to a hearing here; and, after any amount of discussion, from which he emerged as victor or victim, he would be warranted in demanding that the Institute should not, by any official act or expression, commit itself to either side of this or any other question. For by its immemorial tradition, as well as its former and its present Constitution, the Institute is a forum for discussion, and not a tribunal for adjudication.

As is well known, I have been for many years an earnest opponent of the principle of the apex law. And I cannot but be pleased to find that my professional colleagues and fellow-members so generally approve the views on that subject which I have so frequently expressed since my first paper was presented at the Troy meeting, in 1883.¹ Yet I would not have the Institute transcend its proper sphere by formally advocating even this reform, so dear to me, and, in my judgment, so important to the mining industry. With this explanation I shall proceed to offer a reply to some of the statements contained in Mr. Shamel's paper.

Mr. Shamel's excellent elementary statement as to the foundation of law in custom, no one will care to dispute. But in his hands it is a boomerang; for the apex provision of the Act of 1872 was a monstrous violation of all known customs. The customary initiation of mining titles was discovery. Nobody had dreamed of granting to a prospector mining rights upon a vein which he had not discovered—especially upon a vein which somebody else had discovered. And the covering of an undiscovered "apex" with a claim located for the convenient working of another and altogether separate vein, had never been dreamed of as a legitimate basis of title against a real discoverer, who had developed the vein by shaft or tunnel. Nobody had ever heard of an "apex." The dictionaries said it was a point; the statute said it was a "top;" the courts finally declared it to be an edge, and then defined this edge as a surface. But the custom of miners, which Mr. Shamel regards as the sacred foundation of law, had nothing to say about it. Upon this absolute vacuum the apex was builded. Mr. Shamel's eloquent assertion of the merits and the due reward of the prospector and discoverer condemns this fanciful and unprecedented invention of a new structure of law, of which the base was

¹ The Law of the Apex, *Trans.*, xii, 387 (1883-84). See also my subsequent papers: The Law of the Apex, Appendix, *Trans.*, xii, 677 (1883-84); Lode-locations, *Trans.*, xv, 272 (1886-87); End-lines and Side-lines in the U. S. Mining Law, *Trans.*, xvii, 787 (1888-89); Imaginary Boundaries, *Trans.*, xviii, 182 (1889-90).

"the apex." Mr. Shamel cannot produce from ancient or modern history—even from his own treatise on *Mining, Mineral and Geological Law*—a custom, sacred or profane, which warrants this absurdity.

Even the present extra-lateral right, which is commonly regarded as simply continued from the Act of 1866, as that was simply a confirmation of pre-established custom, was in fact entirely novel and arbitrary, not only in its relations to the apex, but also in its relations to all surface ownership. For the earlier right was not properly "extra-lateral" at all, since the boundaries of the surface location did not affect it, and indeed the surface location conveyed no exclusive possession. What the Act of 1866 really granted was a certain length of the lode, and by implication the right to mine upon it to indefinite depth. The lines of the surface location did not condition this right. But the Act of 1872 grants no definite length of lode, and limits all underground rights by such conditions as the course of the apex and the position of the surface boundaries of the claim may at any time be shown to require. And as to those parts of the vein or veins apexing within the location upon which a valid extra-lateral right is granted, that right does not cover actual mining, unless it can be done wholly inside of the vein itself, without cross-cuts, stations, or even niches for supplies, or notches for timbers, excavated in the country rock, no portion of which is included in the extra-lateral possession granted. Mr. Winchell's paper, presented at this session, gives illustrations of this feature of the statute. I cannot say it is unprecedented; for there are amusing cases in the English law books, in which the mineral under a certain area has been sold by the surface owner without the right to break the surface by making roads, shafts, dumps, etc., and it has been held that the purchaser of the mineral owned it, but could not legally get at it or extract it, against the wish of his grantor, the surface owner. To guard against such a situation, our private mining leases usually contain express provisions for roads, dumps, mines, machinery, etc.; and it is universally held that the permission to mine and carry away minerals covers all necessary disturbance or excavation of country rock under the surface.

I do not suspect Mr. Shamel of consciously approving the foregoing violations of custom and equity by the law of the apex. His own argument condemns them.

Mr. Shamel disputes the assertion that the apex provision of this law has caused an enormous and disproportionate amount of litigation, on the strength of an ingenious statistical research which he has made, with the aid of an adding-machine, in twenty-two volumes of Morrison's Mining Reports.

Before I proceed to consider this remarkable investigation, let me note that, in this connection, Mr. Shamel quotes me as having "somewhere stated" that the apex provision has caused "99.9 per cent. of

mining litigation." I have not been able to trace this vague reference far enough to ascertain what was the statement of mine to which it alludes; but I have found the place which Mr. Shamel regards as his authority, namely, a letter in the *Engineering and Mining Journal*, the writer of which ascribes to me the statement that "the 'extra-lateral' underlies 99.9 per cent. of our mining litigation," and says that in the San Juan region of Colorado the proportion is the other way, "at least, so far as the number of cases is concerned"—the overwhelming majority being "adverse" suits, to say nothing of other suits over grub-stakes, division of interests, trusteeships, etc. This passage is given, without its context, in Shamel's *Mining, Mineral and Geological Law*, p. 273. I need only say, with regard to it, that I do not regard all litigation among miners as "mining litigation." Contests concerning advances of capital, tools or materials, on promise of participation in the profits of an adventure, are not peculiar to mining, though the advance may be called a "grub-stake." Nor are contests over the division of interests, trusteeships, etc., mining litigation, though they may concern property in mines or ores. But the "adverse" suits, said to constitute in San Juan the overwhelming majority, are fights over the priority or validity of possessory titles to surface locations, carrying extra-lateral mining rights which are limited by the surface lines. These prospective rights invariably underlie such conflicts. For the grant of the United States is not definite, but conditional and uncertain; and the surrender by a locator of a portion of his surface area, otherwise worthless, may lose for him a yet undiscovered "apex," with unknown possibilities in depth, or, by changing his surface lines, may abridge his rights upon the "discovery" vein over the apex of which (known, but not yet fully traced throughout the claim) he has made his location. This is the reason that "adverse" suits are so frequent and so bitter with regard to mining claims. I must conclude that the correspondent of the *Journal* confined his account of lawsuits "underlain by the extra-lateral" to those in which the nature and limits of that right were the questions at issue, excluding those in which the right itself was in whole or part the object or the cause of controversy. I think Mr. Shamel has fallen into the same error.

He declares himself to have been surprised at the result of his investigation. It is the method, not the result, which surprises me. He sorts 5,808 cases contained in Morrison's Reports into two classes, those in which the prefatory syllabus mentions apex rights, and those in which it does not. And he finds only 115 in the first class, and 5,693 in the second. Whereupon he declares that the apex cases are only about 1.9 per cent. of the whole, and that the chief argument against the apex law "is based on a gross mistake as to the facts in the case." Mr. Shamel suggests a psychological explanation of the error, avoiding the necessity of charging the authors with "willful falsehood." But, while grateful for his good will,

I prefer another explanation, equally protecting my honor, and not impugning my common sense. The amount of mining litigation due to the apex law is not measured by the number of leading cases in which questions of apex rights are directly decided, but by the time, money, and labor expended on lawsuits under this mining law, by reason of its apex rights. The argument which Mr. Shamel controverts is based on the injury to the mining industry wrought in this respect by the law; and that injury consists in the uncertainty of mining titles, which involves constant risk of litigation, and the nature of the litigation thus caused, which makes it ruinously expensive—including, as it often does, costly underground dead work, interfering with productive operation; the preparation of special maps, sections, and models; and the work of experts for long periods, as well as the costs of the long trials themselves.

Lawyers are well aware that a court will not decide a point of law not properly before it in the case under trial. This principle has been illustrated over and over again in the U. S. Supreme Court, by the decision of a case on some point of procedure, which rendered it unnecessary to pass upon the weightier questions at issue. It follows that numerous cases, supposed to involve the construction of the apex law, and carried up to the U. S. Supreme Court for the purpose of obtaining such a construction, were sent back to be tried again, because of defects which settled the controversy anyhow, without solving the more difficult problems presented. The report of such a case would say in its syllabus nothing about the apex law, though the case itself would never have existed at all, or would never have been carried so far, but for that peculiar law.

Again, many cases really caused by the apex provision, go up, even to the U. S. Supreme Court, on interlocutory proceedings. Several of the suits of the Montana Mining Co. described in Mr. Goodale's paper,² as well as others, which he has omitted to mention, belong in this category; and I could add to the number from my own experience.

Moreover, a large number of litigated controversies concern operations in what might be called the extra-lateral region, though they do not involve the construction of the extra-lateral right. These cases are chargeable to the apex law, because they could not possibly have arisen without it. In every such case, no matter what legal question is involved, the parties must show their respective titles, and the apex is immediately an issue, though it may not appear in the syllabus.

Still more important is the circumstance that the vast majority of mining cases decided in the State, or Federal district, courts do not get to the U. S. Supreme Court. A very few decisions of that tribunal have controlled the opinions of the courts below, in innumerable instances. The same is true of the decisions of the State Supreme Courts. Conse-

² The Apex Law in the Drumlummon Controversy, *Bulletin* No. 89, May, 1914, pp. 745 to 758.

quently, the leading cases selected by Morrison as authoritative in the construction of the apex law furnish no measure, even of the number (to say nothing of the importance and cost) of the contests caused by that law.

Mr. Shamel reminds us that all laws—even the Ten Commandments—can be violated. The obvious reply is, that the Decalogue can be easily understood, while the apex law cannot; and that the principal victims of the latter are those who have conscientiously tried to obey it. Mr. Goodale's paper, already cited, tells but one story out of many, which go to explain the present reluctance of capitalists to invest in mining property carrying the incubus of this law, under which no title is safe against attack based on "new developments," or new geological theories.

Mr. Shamel adduces the famous litigation over the New Jersey zinc mines as a sample of what might take place under a law not granting extra-lateral rights. Since he describes this case at length in his book, he must be aware that it originated in the granting of two mining leases on the same land—one for franklinite and iron ores, and the other for zinc and other ores—and that the only mining question involved in it was the distinction between these substances—the minerals themselves being mixed in the deposits mined. But the litigation was prolonged for 50 years by the appearance of new parties, disputes over intermediate conveyances, and other controversies which had nothing to do with mining law. The protracted war ended in a compromise between the grantees of the parties long dead; but there was no time during its existence when the construction of the law involved either the least difficulty or the least advantage to either party. Under any law, men may make ambiguous contracts, and their grantees may fight, if they see fit, over the interpretation thereof. It is the peculiarity of the apex law, that it grants without definition, and leaves its grantee open to attack forever. Besides, the New Jersey case is unique, whereas the attacks upon the *bona fide* purchasers of the U. S. mineral lands are innumerable.³

But Mr. Shamel, while opposing the repeal of the apex law, modestly proposes a few amendments of it, which would be as revolutionary as anything he deprecates. He would have the apex section "re-stated," so as to contain the results of careful comparison and codification of the decisions of the courts, and give full details for their application "to all

³ In his discussion of Mr. Winchell's paper, which I have seen while writing out these remarks for publication, Mr. Shamel presents, as another instance of prolonged litigation, the prosecution of Freeman, Hawthorne and others for criminal misuse of the U. S. post office in the promotion of vulgar mining swindles. I do not see the pertinency of this illustration. The ease with which fraudulent enterprises can be promoted by circulars, offering stock, is notorious. It is estimated that \$100,000,000 has been extracted from credulous investors by that method. Time and money are well spent in prosecuting such criminals, and we do not call the prosecution unnecessary "litigation."

the different situations of veins or other deposits, boundaries, etc., that have arisen in the past or that may be reasonably likely to arise in the future." This, he thinks, "would enable miners to determine their rights for themselves."

But this is only the beginning. The law should also contain geological definitions of the specific kinds of deposits to which it applies. These should be as complete and accurate as the resources of modern economic geology afford. Of course, Mr. Shamel, being a lawyer, knows that if the law authorizes the conveyance of extra-lateral right in certain classes of deposits in place, and not in certain other classes, then the conveyance, by patent, of such rights upon a given deposit, supposed to belong to the former class, would be absolutely void, as an unauthorized act of the Executive, if the deposit should be shown, at any subsequent time, to belong to the other class. No matter how many years of undisputed possession might have elapsed; the grant having been void at the beginning, the grantee would be liable to ejectment from his extra-lateral ground, and to damages for the trespass. This danger exists in the present law. But Mr. Shamel's amendment would make it more acute and more complicated. I notice that he includes "volcanic necks" among the classes to which he would have the law apply. But he does not say how the course of the apex of a volcanic neck is to be determined.

A further recommendation of Mr. Shamel, which he thinks still more important, is that "the enjoyment of extra-lateral rights to any subsequently located claim" shall "depend on the results of an examination by competent, impartial scientific authorities, instead of a battle of alleged experts in a law court before a jury."

I do not exactly understand the force of "subsequently located" in this statement. Under the present law, the rights of junior locators are already subordinated explicitly to those of senior locators; and the majority of disputes about mining titles originate in claims on the part of the senior locator to extra-lateral rights which he asserts and his opponents deny. In every such controversy, one party is *prima facie* the senior, and the other the junior, locator. Does Mr. Shamel propose that the senior locator shall have whatever he claims, while an impartial scientific board shall determine, not only whether the junior locator has a good defense, but whether he has any extra-lateral rights at all? Moreover, under the present law, in a controversy over apex rights, the junior locator may have, and always professes to have, the apex covering the ground in dispute. If he is right, then he is really the senior locator on that particular apex; that question of fact would have to be determined before the impartial scientific board could act at all; and when it was determined, there would be no need of the board.

After careful study, I reached the conclusion that Mr. Shamel meant simply the substitution, in the trial of "extra-lateral" controversies, of an

impartial board, instead of experts paid by the respective parties, with the further difference that the decision of this board should be binding upon judges and juries, and reviewable only by a higher board of the same character—"not a court." This proposition is at least conceivable and debatable, though it involves the overthrow of the present principles of our jurisprudence, the denial of the right of a citizen to appeal for justice to the courts of his country, the conferment of judicial authority upon officials neither elected by the people nor appointed with primary reference to their ability to construe or administer the law. In the administration of justice, such boards may advise, but the courts must decide.

But on reading further, we find that Mr. Shamel proposes his scientific board, not to settle controversies but to prevent them. His plan is, that a claim owner, after due notice to adjacent claim owners, shall apply to the U. S. Geological Survey (or some other scientific authority) to determine whether a given ore body shown in this ground "was one to which, by the proposed definitions, extra-lateral rights belonged or not;" that is to say, whether it did or did not belong among the classes—"veins, shear zones, contact deposits, volcanic necks, and possibly replacement deposits in limestone"—to copy Mr. Shamel's list. But the essential thing is not merely the character of the deposit, but the possession of the apex, to what extent and in what direction. Apparently, Mr. Shamel's board is to settle these points also, quietly disregarding the "parallel end-line requirement," and fixing end planes according to its own judgment. But the law grants extra-lateral rights on all other apexes within the location. If the owner finds another vein in his ground, must he call for the impartial board again, and get it to fix a new pair of end planes? Above all, are these "impartial" (that is to say, *ex parte*) decisions of the board to stand forever, in spite of all later developments of the facts?

Without further discussion of these propositions, I must say in conclusion that I think they constitute a very strong argument against Mr. Shamel's main position; for a law that needs, in the opinion of its advocates, such radical and revolutionary violations of sacred custom, constitutional principle, American liberty and administrative efficiency to make it perfect, had better be repealed than thus amended.

E. W. PARKER, Washington, D. C.—In two important instances in this country the law of the apex was done away with by consent of the parties. At Bisbee, and in one place out in Michigan, the two parties agreed to abolish the apex law, and by mutual consent took the vertical side line and other vertical lines as the delimitation of their properties. That worked out very satisfactorily.

SENATOR THOMAS J. WALSH, Helena, Mont.—Inasmuch^{as} the Drumlummon litigation has been mentioned here, I would like to make a statement, as it may not be known to all present, that that litigation out-

lived three generations of lawyers. The distinguished member of the Institute who has just addressed you was associated with the first generation, and I was in at the death.

It is to be borne in mind, and this I speak with entire deference, that the views of lawyers as well as of engineers, concerning the wisdom of the continuation of the present apex law, carrying with it the extra-lateral right, are likely to be formed or colored by the success or failure of their own efforts in causes in which it played a part.

I have glanced hastily over Mr. Goodale's paper, and I have no doubt it is characterized by the good temper and judicial spirit he usually displays. But it must be borne in mind that Mr. Goodale was an expert in that case, and he was on the losing side. I merely suggest that in reading the paper of Mr. Goodale, you will bear in mind that you are reading a statement of an advocate of one side of the litigation. I do not mean to say it is inaccurate. In looking it over it seemed to me that facts which seemed to us of very great consequence were not incorporated. However, I would not be misunderstood about this, and I do not understand that there is any contention that the final result of the Drumlummon case, the Nine Hour case, was colored with any injustice except so far as injustice inheres in the law which was applied to the case. I am myself convinced that the apex law should be abolished. That as a matter of fact it does give rise to interminable and very expensive litigation, is generally recognized.

Two ideas suggested by Dr. Raymond may, I think, be urged in very just criticism of the paper read in support of the extra-lateral right.

The Amalgamated-Heinze litigation was, as everybody knows, based almost exclusively on contentions concerning the apex right; and yet that litigation developed practically no law at all upon apex rights. The law had practically all been settled. It involved largely questions of fact under the law as it had been settled, questions of practice and procedure, and of substantive right other than such as had that foundation.

I have no doubt you could go through the reports of that litigation, which was continued for 10 years, and I would undertake to say you cannot find half a dozen syllabuses referring to any question under the apex provision. And yet the entire field of litigation almost was traversed. For instance, in that litigation we developed all the law applicable to the writ of supervisory control in our Supreme Court. Writs of prohibition, writs of mandamus, and to proceedings for contempt. And yet the whole litigation rested on the assertion of apex rights.

ROSSITER W. RAYMOND.—Wasn't there a very important interlocutory matter in the Drumlummon case not reported by Mr. Goodale, and which was most extraordinary, where they made a survey of the mine before a suit was brought?

SENATOR WALSH.—Yes, sir. That went to the Supreme Court of the

United States. It involved a statute in our State which permits examination of a property involved as well as of adjacent property, preparatory to the suit. Proceedings were instituted under that statute for the purpose of ascertaining the facts, and the constitutionality of that statute was attacked. The St. Louis Co. sought to go down into the workings of the Montana claim through the workings made by the Montana Co. The Montana Co. attacked that statute as being in violation of its constitutional rights. We finally went to the Supreme Court of the United States, which held that the statute was a proper legislative act. The order for inspection had been granted, and, the inspection having been made, a suit was instituted. Then after the suit was instituted there was a question of the equity powers for permitting an investigation or examination. The adverse was set up, and that resulted in a compromise whereby the strip was conveyed by the owners of the St. Louis to the Nine Hour, and the adverse was dismissed in pursuance of the stipulation.

Then the conveyance was refused, and an action of specific performance was instituted for the purpose of compelling the conveyance; and we had a lot of law on specific performance, and under what circumstances that must be enforced.

Now that was another line. Then, when the conveyance was eventually had, the question became important as to what apex rights were obtained by reason of the peculiar provision of the instrument of conveyance, and when the case was tried out upon the ordinary law of apex controversies, it went to the Supreme Court of the United States, which held that by virtue of the peculiar language and provisions of that deed of conveyance, the rules of law applied by the lower court were not correct; and the case was reversed and sent back for another trial.

Now, there was no question of apex right at all. That is to say, the law applicable to apex rights was considered settled. So that nobody upon the merest reflection should be bound to adopt the conclusion that the amount of litigation is to be determined by the number of propositions upon the question of apex rights.

Next, it is suggested in the conclusions arrived at by Mr. Shamel that the apex law should be more clearly defined, with the aid of all the adjudications made by the courts under varying and conflicting conditions during the past 40 years, and when so defined, and its boundaries fairly well fixed, litigation will to a large extent disappear. I think that is entirely impracticable. The fact about that matter is that we are confronted with exactly the same difficulty in legislation concerning amendments to the Sherman Anti-trust law. It is claimed by many people that now it is possible to fix definitely by bounds the things which would constitute a violation of that law. But the trouble lies in the new methods of arriving at the results which were intended to be obviated by that law; and as a result many practices which should be condemned will escape

and be considered legitimate. So it is a rather difficult kind of a task to attempt to elaborate the definition of the apex and eliminate litigation in the future. Indeed, I would be afraid that course would give rise to no end of litigation. Every provision, every clause, would then become the subject of renewed litigation, and the question would arise whether by reason of the peculiar language it was intended to abandon some of the principles which had been theretofore abandoned as being unjust. So, I think you would breed rather than obviate litigation by any such provision.

Then, Mr. President, I do not think the suggestion that these questions should be transferred from the courts to the Department of the Interior could ever have been suggested by any man who knows the temper of the Western people, who are most interested in this matter. Indeed, we have always maintained that the best feature of our mining law is that which throws into the local courts conflicts arising between conflicting interests, rather than have the crude methods pursued in the Land Office and other departments of the government applied to the decision of such matters.

And I undertake to say that unless the whole law is changed it will be impossible to compel the litigant to abandon his rights in court and trust them to the arbitrament of even the distinguished gentlemen at the head of the Geological Survey. I do not know why anybody should want to transfer a controversy between me and my neighbor, if a controversy should arise between us, in respect to the apex of a vein;—why the determination of that controversy should be transferred from the vicinity where it arises, into some government bureau in the city of Washington. Nor do I understand why it should be determined there, rather than before a judge out in the mining country who has made it a duty to be advised concerning mining law and mining business; and tried before a jury that has imbibed more or less mining law knowledge by reason of long residence in the vicinity where such questions are constantly arising.

I do not mean at all by anything said in the beginning that I do not agree with the conclusions at which Mr. Goodale arrives, that perhaps this is the time for the abolition of the apex law.

CHARLES H. SHAMEL (communication to the Secretary*).—I have been favored by the Secretary with a copy of the stenographic report of the discussion relating to my paper on the apex law. This report, however, is plainly so defective that I can only be sure of the general drift of the remarks of the speakers. Dr. Raymond is mistaken in thinking my paper especially directed at his former utterances. My argument was based on the origin of law in general from custom and

the undesirability of reversing legal rules founded on custom. Dr. Raymond says that the apex law, instead of being founded on previous custom, was absolutely revolutionary in character. As nearly as I can understand him from said report, he bases this assertion on the alleged fact that before 1872 no one had ever heard of the "apex"; nobody knew what it meant. This is a mere play on the mathematical definition of the word. The fundamental principle of the so-called apex law—that is, the right of the discoverer to follow a dipping vein wherever it might go without regard to surface boundary lines—had been known and practiced generally in the precious-metal mining districts of the United States ever since such mining began after the great California gold discoveries of '49. This is proved beyond any doubt by the quotations I gave in my article and by the collection of the local district rules and regulations previous to 1872 made by Clarence King for the 10th census.

Because, in the law of 1872, the word "apex" was used in a statute for possibly the first time in an effort to render the principle definite and precise does not render the principle new. Such argument is mere verbal dialectic and indicates there is nothing more substantial to offer against this argument for the apex law.

I submit that the position is still unshaken that the apex law, being founded on previous general customs of the precious-metal-mining districts (to which alone it applies), should not be abrogated and the contrary rule adopted. It is not maintained that conditions have materially altered except perhaps by the entry of "Big Business" into the mining industry. No one has ever suggested that the prospectors and mine operators originating from them are not practically unanimous in favor of the apex law. These are the "people"; they are the representatives of the nation in relation to the mining industry; their customs alone originate law, their wishes should control the provisions of the statute.

There is no merit in the objection that the number of apex syllabuses does not represent the proportion of apex litigation. The apex question was just as often subordinate in cases chiefly contested on other sections of the mining laws as such other questions were subordinate in cases turning chiefly on the apex provision. In modern times these syllabuses form a very carefully prepared and accurate abstract of everything in a court decision, with a separate paragraph for each contested question or point in the case. There is no better way than this practicable to estimate the relative amounts of litigation regarding different legal questions. Wherever the apex law was involved in a decision it appears in the syllabuses. This method is infinitely more trustworthy than the mere impressions of any man, however experienced he may be, for such impressions are formed only by each individual's personal experiences, which are wholly accidental with reference to the whole field of mining litigation.

tion. The confused and erroneous ideas so widely prevailing concerning the apex law are well illustrated by the member who, in the discussion, referred to it as in force in Michigan. That State was never, at any time, subject thereto. The statement that the U. S. Supreme Court evades deciding the main questions in cases is another striking illustration of the desperate straits of the opponents of the apex law and of the lack of valid argument against it. I am still satisfied that counting the syllabuses is as fair and accurate a way as can be devised for determining the proportion of apex litigation and that, in truth and in fact, the apex section of the mining statute has caused less than its proportionate share of mining litigation and therefore should not be abrogated on account of causing excessive litigation.

What I proposed as to a new way of granting apex rights and possibly of deciding some apex controversies was merely a suggestion for starting discussion. I am not, however, convinced by what the able and experienced Senator from Montana has said that some such plan would not be an improvement over the present method. Also I did not mean to suggest that such grant or contest should be made at or conducted in Washington, D. C., but, on the contrary, would provide that the hearings be had in the locality, with an inspection of the property by the board of experts.

I have always recognized that there were difficulties in the administration of the apex law, but these exist in relation to the other and unattacked provisions of the mining law in even greater degree, as the facts I have adduced seem to me to prove conclusively. These facts have not been disproved or explained away by anything in this discussion. The law is just in principle. Therefore I still maintain that it would be economically inexpedient and ethically unjust now to repeal the apex law.

Recrystallization of Limestone at Igneous Contacts

Discussion of the paper of C.K. LEITH, presented at the New York meeting, February, 1914, and printed in this (June, 1914) *Bulletin*, pp. 1129 to 1134.

WALDEMAR LINDGREN, Boston, Mass.—I think geologists should be careful how they associate with mathematicians. Now, what these men have done is simply to take two adjoining bodies, one very hot and one very cold, and calculate the conditions resulting. That is not the idea at all. The idea is that the limestone receives the emanations from the magma, and that these magmatic emanations pass through it almost like water goes through a sponge, so that, reasoning from that point of view, I do not think that this mathematical calculation has very much bearing on it, because the emanations would find fissures and penetrate long distances

that would upset the whole calculation. The penetration of the emanation would be very much more rapid in any case than would be the diffusion between two bodies adjoining, without any action on each other except by the heat.

JAMES F. KEMP, New York, N. Y.—The igneous rocks sometimes show the reflex action of the limestone in having garnets in epidote.

I was interested in reading Professor Leith's paper in the emphasis laid on the delicate balance of the materials that enter the limestone, with the result, it would seem, that one could reply that when the material ceases to come from the igneous rock the garnet ceases to form.

Mr. Lane raised the point, inferentially, that Mr. Lindgren cannot show the effect of the limestone of the igneous rock, but I believe he has shown such effect in that he mentioned garnets and epidotes in the igneous rock itself.

Professor Leith has made an objection to the contribution from the igneous rock, because of the delicate balance between the recrystallized materials and the contributions, but, of course, if the garnets are produced from the limestone by additions of silica and iron, their crystallization would cease when the contributions stopped. A balance would necessarily, therefore, be established. I presume, however, that Professor Leith's point was based upon this consideration—he believes the limestone to be calcined by the igneous rock, but that limit itself would result in greater quantity than is present in the silicate,

JOHN D. IRVING, New Haven, Conn.—For reasons which it seems unnecessary to give at length I believe that the lime silicates which develop at the contacts of intrusive igneous rocks with limestone are formed almost entirely with the aid of contributions of silica and limestone derived as emanations from the intrusive magmas. The formation of such silicates, however, demands the elimination of a large quantity of carbon dioxide, inasmuch as this compound does not appear to any extent as a constituent of the lime-silicate zones. While there are, of course, many ways in which carbon dioxide can be expelled in gaseous form and escape through fractures and fissures to the surface, it is conceivable that under conditions of high temperature and pressure, such as have accompanied deep-seated intrusions, fissures and fractures for the escape of carbon dioxide should have been either wanting or insufficient in number. The elimination of the carbon dioxide under these circumstances offers a problem of some difficulty. I would like to learn from Mr. Lindgren whether he has any suggestions to offer in regard to this point.

Our National Resources and Our Federal Government

Continued Discussion of the Author's paper on this subject, presented at the Cleveland Meeting, October, 1912 (*Trans.*, xlv, 612), and Discussion of the paper of GEORGE OTIS SMITH on The Disposition of Natural Resources, presented at the New York meeting, February, 1914 (*Bulletin* No. 82, October, 1913, pp. 2403 to 2408).

R. W. RAYMOND, New York, N. Y.—The propositions contained in my paper above mentioned were denied, in whole or in part, in communications to the Secretary from Messrs. R. B. Brinsmade, Pueblo, Mexico; G. W. Wepfer, Berkeley, Cal.; and George Otis Smith, Director of the U. S. Geological Survey.

Mr. Brinsmade, agreeing with me as to the desirability of local, rather than Federal, administration, and as to the evils inseparable from the Federal "apex law," dissents from my "four final propositions," for reasons given in addition to those stated by Director Smith.

These four propositions were in substance: (1) that governmental interference with private enterprise is in general undesirable; (2) that, if necessary, it should be performed by a municipal or State, rather than by the Federal, government; (3) that a certain system of leasing (which I defined) is the worst that could be adopted by any government; and (4) that the argument for Federal leasing of the coal lands and water powers of the public domain would be equally strong for a similar treatment of other mineral, or even of agricultural, lands, concerning which we had heard, at the time I wrote my paper, practically nothing from the advocates of the new Federal policy. At least, such an extension of the new plan of Federal control had been explicitly disclaimed by President Taft, in a public address, advocating that plan for coal lands and water powers.

Mr. Brinsmade explicitly approves my second proposition, and does not explicitly condemn my first. His opposition to the third (apart from arguments like those of Director Smith, to be considered later) seems to rest upon an economic theory as to the private ownership of land, which I must decline to discuss here. It is an old theory, as familiarly known as it is frankly rejected by the best authorities. But it is not pertinent to this discussion, because it attacks the ownership of all land whatever.

It is "on the basis" of this economic theory, that Mr. Brinsmade disputes "the assertion of Dr. Raymond that our railroad land grants and mineral-land laws were essential to the settlement of the West." I did not use the word "essential," and I plainly said that the result referred to might have been reached by much longer processes. But the difference is trivial; and the facts are incontrovertible "on the basis" of any theory whatever.

As to my fourth proposition, Mr. Brinsmade claims that it has already been answered by many foreign countries. Perhaps I am to blame for his misunderstanding of my statement, which referred exclusively to this country, where the doctrine of the governmental ownership of all property, apparently approved by him, is advanced by a party not yet venturing to carry it to its logical result, and therefore confining it, for the present, to certain kinds of property, called National Resources. I am grateful to Mr. Brinsmade for his through exposition of the doctrine and its ultimate results.

Mr. Wepfer describes the government leasing system of Bolivia, which is quite unlike the system I characterized as bad. According to his account, any one, citizen or foreigner, can acquire in Bolivia for \$1.66 per acre a patent to mining ground of which he holds exclusive possession so long as he pays semi-annually a tax of 33 c. per acre. This is practically a purchase of the ground. The forfeiture of ownership by non-payment of the small tax constitutes no hardship. Continuous working is apparently not required. The revenue from the tax must be insignificant—scarcely enough to cover the cost of collection.

To my argument that it would be unwise for the Federal government to treat the different States differently, Director Smith replies only that there has been for the past half-century an evident tendency to expand Federal functions, and that it is only "national" resources that the national government is to administer—and specifies the mineral wealth of the public lands as an illustration. My rejoinder is simple. I admit the existence of such a tendency as he asserts; and I recognize that it has been largely created and augmented by the zeal of enthusiastic heads of Federal departments; but I believe it to be dangerous and unwise.

In emphasizing the all-important distinction between the landlord and the sovereign, and showing that in dealing with the public mineral lands, the United States has acted as the owner, not as the sovereign, I have hitherto tacitly assumed that the ownership of the United States was as complete as that of any private owner of the common-law fee-simple, covering both the surface and the minerals beneath it, and conferring the right of perpetual possession, free administration, lease or sale of both together, or of either separately. For the purpose of my argument, this was sufficiently accurate; for the mineral-land laws under discussion looked to the sale of the lands only. But now it is frankly proposed that the United States shall no longer sell these lands, but shall hold and lease them for the benefit of the Federal treasury. This warrants a closer inquiry into the history and nature of the "ownership" of the public lands; and such an inquiry discloses some interesting facts. Perhaps our fathers did not know as much as we do; perhaps their promises are not binding upon us; perhaps their institutions are not worthy to stand against our "tendencies." Nevertheless, we can afford, in our superior strength and wis-

dom, to learn what we are undermining or overthrowing, or even simply (but not less destructively) ignoring.¹

The well-known origin of our public domain, through the cession of Western lands by certain of the original States, was the result of an agreement, ending a long controversy, consummated before the adoption of the Constitution, and confirmed in that instrument. In that controversy, Virginia, the steadfast champion of State sovereignty and opponent of imperial domain, refused, against great pressure, to consent to the transfer of unconditional ownership of the lands in question; and, after a proposition made by Congress in 1780, submitted her own counter-proposition, which was agreed to by Congress in 1783; and March 1, 1784, the deed of Virginia was accepted by Congress. This deed transferred all the rights of Virginia on conditions which made the Federal government, not absolute owner, but trustee, authorized to administer the public lands for the benefit of all the present or future States of the Confederation, but bound to dispose of them so that they should be settled and formed into distinct republican States, and as such admitted into the Federal union, with the same rights of sovereignty, freedom and independence as the other States.

The deeds of other States, held back to await the decision of Virginia, were subsequently executed; and the Confederation thus became holder in trust of the public domain, on the conditions which the wisdom and courage of Virginia had imposed.

The Convention which framed our present Constitution came some years later (1787), was subordinate to Congress, and was required to report to Congress; and the provision of the new Constitution, relating to the public lands, was framed in conformity with the Congressional ordinance of 1787, and confirmed the contract with Virginia, by declaring that nothing in the Constitution should prejudice the rights either of (1) the United States, or (2) of any particular State. The former rights covered the profit which the United States might realize from the public lands before their sale or through their sale; the latter rights were those of the grantor States to insist that the one and only purpose of Federal management should be the sale of the lands and the formation of new States.

I think no competent student can deny that the title of the United States under the new Constitution was the same as under the preceding Articles of Confederation, and that, whatever its exact nature, it forbade the adoption of any system of Federal administration looking to the

¹ The following statement, based upon an address on *The Status of the Public Lands, and Its Relation to Mining Tenures*, by George T. Edwards of Berkeley, Cal., delivered before the San Francisco section of the Mining and Metallurgical Society of America, and published in the *Bulletin* of that Society, No. 57, vol. vi, No. 2, (Feb. 28, 1913), I have verified by reference to original sources.

permanent Federal ownership of the public lands. Refined subtlety of construction as to the language of these documents is out of place; for the old common law still governs us in the interpretation of contracts according to the intention of the parties; and concerning this intention there is no doubt. Virginia victoriously maintained, Congress finally accepted, and the new Constitution ratified, the principle that no great tracts of land, held by the central government, should ever prevent the formation of new sovereign States, or impair the power and rank of such States. If further confirmation be needed, it may be found in the decision pronounced by the United States Supreme Court in 1845² concerning certain public lands in Alabama, which declares:

"The United States never had any municipal sovereignty, jurisdiction, or right of soil in and to the territory of which Alabama, or any of the new States were formed, except for temporary purposes, and to execute the trusts created by the acts of Virginia and Georgia legislatures, and the deeds of cession, executed by them to the United States, and the trust created by the treaty with the French republic, of the 30th of April, 1803, ceding Louisiana. . . . When the United States accepted the cession of the territory, they took upon themselves to hold the municipal eminent domain for the new States, and to invest them with it, to the same extent, in all respects, that it was held by the States ceding the territory."

This decision goes far. It declares that even the territory acquired by the Louisiana purchase, like the land previously ceded by some of the original thirteen States, is held by the United States as a trust for the creation and endowment of new States. In the treaty of 1803 with the French republic, the United States guaranteed to the inhabitants of the ceded territory the same rights as were guaranteed already by the Constitution to the inhabitants of the existing public domain.

But in the argument which Director Smith opposes, I did not go so far as to assert that the Federal administration which he advocates would be unconstitutional. I contended only that it would be unwise, as a matter of statesmanship. And I am still content to stand upon that ground. Whatever the legal or historical reasons, the fact is, that the public domain conveyed by some of the original States has been treated according to the contract made with them; that the State of Texas was admitted by treaty, with its own public lands and land laws, and entirely free from any claim of the United States over any one of its "natural resources"; and that hitherto, the policy of the United States, as expressed in its legislation, has been to carry out, whether it was or was not legally bound so to do, the policy established in 1784, and then expected to be permanent—namely, that of administering the public domain for the purpose of having it occupied, as soon as possible, by citizens owning it; of forming out of these citizens, as soon as practicable, new States, and of leaving to

² Pollard's Lessees *vs.* Hogan (3 How., 212), cited by Mr. Edwards in the paper already mentioned, and consulted since by me.

these new States, as a source of revenue, the taxation of the wealth produced from their natural resources.

What Director Smith and his party propose ought to be acknowledged as at least new. They propose that instead of selling "national resources" to private citizens, and thereby making them taxable by individual States, the Federal government shall hold on to them and lease them—not for the appreciable direct advantage of the Federal treasury; nobody dreams any such wild dream as that—but for enough to pay the cost of Federal "administration" and experimentation, leaving the State to get what revenue it can by indirect taxation.

The State of Texas is already looming up as a formidable rival of its sisters in the Union. Its immense area; its varied natural resources; its climate, favoring all industries; and, above all, its complete immunity from Federal control (whether in the name of "conservation" or in any other name) of its own self-directed internal development, are making it more and more portentous. Whatever may be adopted as the policy for administering United States public lands in other States will not apply to Texas, which contains no such lands.

I venture to say that any legislation which handicaps other States, by reason of their history, while it excepts Texas by reason of its history, is bad statesmanship. I know that the theoretic right of secession from the Union was disproved by a terrible war; but I am equally sure that the peaceful and prosperous maintenance of the Union requires a just and equal treatment of the States which compose it. Whether the people of a given State resort to futile armed rebellion or not, is a subordinate question. The main question is, do they feel that they are justly treated?

The policy established by our fathers in the contract with Virginia and other States, whether it be or be not, with regard to all the territory subsequently acquired, legally binding upon us, solves this problem. And that is why I say, independently of all legal and constitutional argument, that the revolutionary novelty of permanent Federal ownership would be bad statesmanship.

Director Smith objects to my contention that local, instead of central, government should deal with local interests and industries, asserting his belief to the contrary. But I am of the same opinion still, though I abstain, for the present, from advancing further proofs of my proposition, which might involve invidious criticism of friends in official positions, whose loyal labors I am not inclined to disparage. Yet I must be permitted to say in general terms, and without personal references, that I have seen in no department any proof that government employees are specially qualified to supervise the operations, solve the problems, or prescribe the models for individual enterprise. This general statement covers State as well as Federal activities; but with reference to the distinction between them, I will add that, within the last ten years, at least

two of the Federal executive departments have been guilty of errors which have escaped the condemnation they deserved, and would have received, if they had been perpetrated in the service of a single State. What I know of Federal administration has convinced me that it is less desirable than State administration, and that either of them is less desirable than individual enterprise under ordinary legal responsibility, as a means of securing the benefits of public prosperity and social justice which the advocates of all these systems profess to seek.

Director Smith defends the leasing system on sundry grounds of economic theory, and by appeal to the experience of other nations. I must postpone to some future occasion the thorough discussion of that subject. The most important feature of the present situation is the circumstance that the intelligent and patriotic official head of a great and useful Executive department, now frankly advocates this system for all the mineral resources of the public domain. I, for one, welcome this declaration as a substitute for the piece-meal suggestions of exceptional treatment for special "resources," like Alaska coals, or phosphate lands, or possible deposits of radium. Let us have the full plan of Federal control at once, and know what we have to face!

It is hardly necessary to assure us that the proposed leasing of "national resources" would be done rather for public benefit than for public revenue. Past experience, as well as common sense, assure us that the expenses of inspection and regulation would quite absorb, if they did not exceed, the net revenue of the system. Indeed, those of our States which possess no resources on which the Federal government can at present lay its helping hand—a large majority, in fact, of those which constitute the Union—might count themselves fortunate in such an event, if they did not have to pay their share of the extra expense of the blessings governmentally bestowed upon their less fortunate sisters.

Director Smith corrects what he regards as an inadvertence on my part in my statement that the advocates of a leasing system contemplate the requirement of continuous operation, and adduces to the contrary a passage from his annual report of 1911, referring to needed legislation for oil lands. My statement was based on a plain declaration of the President of the United States to the effect that "natural resources" should be leased by the Federal government on the condition of continuous utilization of the privilege granted. I had not read Director Smith's report; but I do not see that his plan for oil leases offers anything essentially different. Instead of punishing the suspension of production under lease by the forfeiture of the lease, he would make the "ground rental sufficiently high to discourage the acquisition of lands except for immediate and continued development," and he says that "this indirect control of development would be preferable to the direct enforcement, by forfeiture, of continuous production, which should be avoided because of

the danger of disturbing the delicate equilibrium between supply and demand."

As I understand this statement, the high ground rent is expected to discourage the holding of oil land without producing oil from it. On the far greater and more immediate evil of reckless overproduction and consequent irrevocable waste, it would have a contrary effect, if it had any. It would be much more practical to lower the rent for a tenant who was prevented by the delicate operations of supply and demand from producing oil at a profit, and thereby encourage him to assist the government in conserving the oil until a rise in market price showed that it was needed. But most practical of all would it be, to let the aforesaid equilibrium between supply and demand (the variations of which, though delicate, are irresistible) take care of the whole matter. For it operates by the old-fashioned law of gravity, with which, when it is acting in our favor, we can often best co-operate by letting it alone. But I did not intend, in this reply to Director Smith's criticisms of my paper, to wander into a discussion of his own later utterance; and, indeed, I would not have mentioned it had he not called attention to it in his remarks. The question presented by the petroleum lands is in many respects peculiar, and deserves separate consideration.

Water Intrusion and Methods of Prevention in California Oil Fields

Discussion of the paper of FRANKLYN W. OATMAN, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 87, March, 1914, pp. 531 to 553.

ARTHUR KNAPP, Ardmore, Pa. (communication to the Secretary*).—*Cementing*.—I believe that the washing of a well with clear water is, in general, a very bad thing. There are probably formations that will stand up after being washed with clear water, but my experience in California and elsewhere has been, that to wash a well with anything but a well-mixed mud would stick the casing before cementing could be accomplished. The mud is the only thing that will wash the oil out of the hole and hold back the gas. Neat cement is heavy enough to displace the mud adhering to the casing and give the desired contact between cement and casing. Further, any well in which the circulation cannot be established in 3 hrs. is not in a condition to be cemented. The casing should be removed and the hole reamed. Circulation for a long period of time serves only to cut deeper channels around the casing and does not clear the entire circumference.

Method A.—The operator can never be sure of a successful cementing job in using this method, for the reason that, with the packer outside

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and the plug inside of the tubing, the casing is closed almost water tight, and on pulling the tubing the cement will be pumped back into the casing. No seat can be made that will withstand this suction. The tubing being plugged at the bottom, the string will have to be pulled wet. This is a very bad feature, especially in the long strings, as it puts an unwarranted strain on the tubing if an attempt is made to pull the tubing fast. It also more than doubles the time of pulling.

Method B.—To have success with this method, the packer must be *absolutely* tight, otherwise the tubing will be cemented in. As the chances are there will be at least a small leak, it is well to take precautions to prevent the casing from being cemented in. Either the tubing must be removed or the left-hand thread must be on a connection further up the hole. The bottom 4 to 8 ft. of the tubing may be made of cast iron with longitudinal flutes to make it light enough to be easily broken. This method is open to the same objection—that the string will have to be pulled wet.

Perkins Method.—This method can only be used in a few wells where conditions for cementing are ideal. Under average conditions the casing has to be pulled more than 3 or 4 ft. off of bottom, which makes the cast-iron packer too long. Also in most cases the casing has to be moved up and down as the heavy cement is being pumped up outside of the casing. The second packer, *B*, if made as shown would stick and turn sideways, allowing the cement to pass it. The cast-iron packer is not easily broken. The stem being bedded in cement, it is in a condition to resist being broken by ordinary blows. There is great danger of breaking the seat while drilling out cement alone and the iron adds materially to the amount of drilling necessary.

The Apex Law in the Drumlummon Controversy

Discussion of the paper of CHARLES W. GOODALE, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 89, May, 1914, pp. 745 to 758.

SENATOR THOMAS J. WALSH, Helena, Mont.—I should point out the one apex proposition which was determined in this St. Louis-Nine Hour litigation.

There ran along in this litigation the question of what right would exist in the case of split apex—that is, where a portion of the apex was within one claim and a portion within another claim. That question was determined in another case.

In Fig. 3 of the paper you will see illustrated what is called the 108-ft. plane and the 135-ft. plane. The apex of the vein in question is quite wide. Going now to the upper left-hand corner, you will observe that coming from the left is the apex of the vein. It reaches the Nine-Hour

or compromise ground, marked in blue, at what is indicated as the 108-ft. plane. At that point it begins to enter the compromise ground, but it is not all within the compromise ground until the 135-ft. plane is reached.

Now, the question arose as to where the rights of the St. Louis Co. ceased—whether its rights ceased when the vein first began to enter the compromise ground at the 108-ft. plane or at the plane at which it was all within the ground.

Judge Hiram Knowles, before whom it was tried, a judge profoundly familiar with the mining law, held that the rights of the St. Louis Co. ceased at the 108-ft. plane. That is, he held that as soon as any portion of the vein entered the compromise ground, the line parallel to the end line was there to be drawn defining the extent of the apex right. That view, however, was reversed by the Circuit Court of Appeals, which held that the 133-ft. plane bounded the extra-lateral rights (that, by the way, should be 133 in place of 135), and it determined the proposition that in case of a wide apex passing from one claim into another the owner of the apex could take all of the vein up to the plane which marked the most extreme point.

R. W. RAYMOND, New York, N. Y.—Having been personally engaged, as expert counsel and witness, in the first part of the litigation described by Mr. Goodale, and cognizant of its later stages, I desire to point out briefly some of its more important features.

1. This contest, from beginning to end, was waged by the St. Louis Co. against a company of British capitalists who had purchased the property, and invested their money for its development, in entire good faith, and who never undertook to do more than defend what they had bought and paid for, and honestly believed that they owned. That this company, after many years of litigation, was finally forced to submit to a practical confiscation of its supposed property, is chargeable, not to the action of its adversaries, who doubtless claimed only that to which they thought themselves legally entitled, but to the law which made such conflicts possible. The victims of the law in this case were those who had sedulously sought to obey it.

2. Mr. Goodale omits to mention in detail a number of interlocutory proceedings, which would swell the aggregate amount of the litigation involved.

3. This litigation involved several "leading cases," in which the U. S. Supreme Court settled important questions for the first time. But, before mentioning these, I would call attention to a point decided in the so-called "Triangle" case,¹ and not appealed, but subsequently decided in the opposite sense by the U. S. Supreme Court in another case.²

¹ Montana Co. *vs.* Clark, 42 Fed., 626.

² Del Monte Mg. Co. *vs.* Last Chance Mg. Co., 171 U. S., 55.

The latter decision set up a novel doctrine as to the nature of the title of a possessory owner of mineral land by location. The statute said that such locators should have "the exclusive right of possession and enjoyment of all the surface included within the lines of their locations"; and this was naturally construed by everybody to mean that this possessory title, so long as it was kept valid by compliance with legal requirements, was equivalent to ownership, so that no one could intrude upon the surface which it covered without committing a trespass; and that, consequently, no one could establish a location in the required rectangular form by putting his corner stakes on another man's ground, and then, in his application for patent, excepting the ground already owned by his neighbor, but claiming the rest as a valid location, whatever its shape. But the U. S. Supreme Court held, in the later *Del Monte* case, cited above, that the locator was not an owner, and that "imaginary boundaries" might be drawn for a later location on the surface of his location, so as to secure extra-lateral rights, not impairing *his* rights.

The Montana statute, authorizing the inspection and survey of a mine, to ascertain whether a trespass had been committed, and the extent of damage thereby inflicted, *for the purpose of getting facts for the commencement of a suit*, was a decided novelty. According to the previous practice, a party might apply for such an inspection, after he had brought a bill in equity, just as the complainant in a civil suit may apply for permission to examine the defendant prior to trial; because the commencement of the suit puts the question of the property concerned within the jurisdiction of the court, and places the complainant under serious responsibilities and pecuniary liabilities. But it was a startling proposition that a party could go before a court and say, "My neighbor is mining much ore; and I would like to know where his bonanza lies, and whether some of it is not mine by extra-lateral right or otherwise. I will not take the responsibility of beginning a suit, because I do not really know that I have any ground for legal complaint, and I do not wish to frame a complaint, without first ascertaining just what to claim, and where, so as to secure the greatest advantage by victory—or else, discovering that I have no case, so as to avoid fighting altogether. I therefore ask the court to order a survey of my neighbor's mine workings for the purpose of giving me the information I desire"—and that the court might make such a decree, without requiring, as a condition precedent, the commencement of a suit.

When I say this was new doctrine, I do not mean that similar statutes did not exist in other States. It is enough to say, that the question was carried to the U. S. Supreme Court in this case, on the proposition that the decree authorizing entry, survey, etc., upon private property in advance of any other proceeding, was not "due process of law," and therefore that the statute authorizing it contravened the Constitution

of the United States. The U. S. Supreme Court (in 152 U. S., 160) upheld the constitutionality of the statute. I think this decision was correct, though it disappointed my clients, and established a possibly dangerous precedent.³ For I hold it to be a sound principle of American equity practice that a court may order (under proper safeguard against injury, and bonds for possible damage, to property) the entry upon private property, for the purpose of ascertaining facts not otherwise to be known, and necessary to the just decision of a question before it. The Montana statute authorized a court to do this, when there was no question before it. But this extension of judicial authority was a necessary result of the apex law; and any evils which may flow from it are directly chargeable to that law, which conveys mining rights so vague, and so dependent upon geological phenomena, that nothing but an underground survey can determine them. In order to prove title, every stope in a mine must be traced to an apex; every branch of a vein must be connected with an apex right, etc.; and there is no doubt that, under this law, mining operations may be (and sometimes are) extended by cross-cuts underground into veins not within the extra-lateral right of the operator. Congress having established the (undoubtedly constitutional) apex law, State statutes, necessary to its just administration, ought not to be lightly regarded as unconstitutional; and, while I am still inclined to think it would be wiser to require the commencement of a suit before the application for a survey, it seems to me that the Montana statute should not be condemned as omitting to provide a "due process of law."

The extent to which the apex law has thus required the enlargement of judicial power is amusingly illustrated by a mining case in British Columbia, of which I give an outline, omitting names, dates, and other details, as not necessary to my present purpose. British Columbia, at one time, adopted the apex law, and, after some years of experiment and amendment, abandoned it, returning to the system of "square locations" and vertical boundary planes. Long afterward, a controversy arose between two mines, involving the extra-lateral right acquired for one of them by its original location. The evidence showed that A had on the surface of his location the plain apex of a vein, which he had followed with an incline down to a point where it reached the vertical plane through his side line. Here A was stopped by an injunction, forbidding him to trespass upon the ground covered by the location of B. Meanwhile, B, in working his mine, had followed upward a precisely similar vein, until he reached a point 12 ft. (if I remember correctly) on the dip, from the said boundary. It was this vein that A claimed, under his apex right. But the continuity and identity could not be proved beyond

³ The danger is, perhaps, not great; for the applicant for such a survey, "fishing for evidence in advance," can get it only in the discretion of the court, and would doubtless have to show reasonable cause.

possible doubt, except by driving, on B's ground, from the bottom of an excavation of A, an incline to reach the top of B's excavation; and A's application to have this connection made under the direction of the court was refused. In vain were the opinions and decisions of American courts, interpreting the apex law, produced in argument. They were received with complimentary comments as worthy of consideration, but not authoritative in a Crown Province, taking its precedents from the British Chancellor or the House of Lords. It was very difficult to find such precedents; for the British decisions had never dealt with the apex law; but finally (to skip all intermediate steps) an equity decision was discovered, which proved to cover the point. It was in a dispute, involving the responsibility for an evil odor, proceeding from a system of drains; and since the actual source of the evil odor could not be ascertained without investigating the connections of the drains under an adjacent public highway, the court ordered the digging-up of that highway, in order to obtain the information necessary to a just decision. But the highway belonged to the King, and if the King's property rights could be violated for such a reason, how much more those of a private subject! It was this fragrant precedent that ultimately secured for A an order of court, authorizing the excavation which established his extra-lateral right.

Many other points in Mr. Goodale's paper might be made the subjects of extended comment. But I will mention only one. The complicated litigation over the "compromise ground," which ultimately ruined the Montana Co., was due to the form of the agreement (and the subsequent deed, conforming thereto) which conveyed only a given area, "with the mineral therein contained", and did not name the extra-lateral right. If the "compromise ground" had been conveyed without any such specific qualification, the conveyance might easily have been construed, under the principle laid down in the old Eureka case, according to the custom of miners, as conveying all the rights appertaining to the ownership of the ground as part of the senior Nine Hour location. That was undoubtedly the original intention of the parties. But the specification in a deed of what is actually conveyed, may exclude the consideration of other things orally promised or mentally intended. It may fairly be said, therefore, that the hardship and injustice suffered by the Montana Co. in connection with its "compromise ground," were due to the error of its original grantor. Nevertheless, it still remains a pertinent objection to the apex law that such punishments of innocent error and such hardships to *bona fide* investors can take place under its honest administration.

M. K. RODGERS, Santa Monica, Cal.—My early work was in Butte, and I remember one of the first things I noticed there on the apex law was a fracture owned by a gentleman in Butte. Considerable ore was being taken out. The Anaconda Co. was interested in the ore taken out,

and found that there was a small vein on the side of the mountain. We got into litigation and we found that both were taking out ore which did not belong to either, but which belonged to the Anaconda Co.

My experience in British Columbia and in this country led me to this: I eliminated the apex law from all concerns or properties, and we would not have anything to do with a mining company that based its title on the apex law, and a bunch of lawyers and a bunch of innocent Western jurors to give you justice.

The expert's work is only good until further development works changes; change conditions, and he will then change his opinion. I found it worked well to eliminate the apex law from my work entirely. The indefiniteness of the apex conditions is such that no apex law will cover it. It is much easier to do mining business in Canada, for the reason that their laws are definite, and ours are not.

If you own a claim at the head of the Copper river to-day, and another on the British side of the line, it will cost the American anywhere from \$1,000 up to do \$100 worth of work; while on the other side you can remit your check and do the work.

In British Columbia you have something in the way of an abstracted title. As soon as you do your assessment work you must describe what you do. At the end of five years, you have something. On the Alaska side you do not even have a record of the work of two years ago, and at the end of five years your title depends upon whether there are more people there to swear you did your work or more people who will swear you did not do the work. A wise Swede out there had a claim, during the days when everything was jumped. He had seven or eight corners at his corner post. Some one noticed it and said, "Somebody has been jumping you too, have they?" He said, "No, I jump myself every six weeks."

CHARLES H. SHAMEL, Springfield, Ill. (communication to the Secretary*).—Mr. Goodale's history of the litigation involving the Drumlummon lode is a clear and interesting account of a celebrated controversy involving the apex law. It is written at this time for its bearing on the proposed repeal of the apex law; and its intended moral is that the apex law is "immeasurably bad." He asks: "Can any one describe an instance where such injustice has resulted in the operation of the mining laws of other countries?" Unfortunately, I am not familiar with the history of mining litigation elsewhere than in the United States and so cannot answer the question exactly as proposed. There are, however, only two practical alternatives for a law governing underground rights anywhere in the world: allowing the vein to be followed wheresoever it may lead (the principle of the apex law); and limitation

* Received Feb. 3, 1914.

by vertical planes through boundaries. The latter alternative rules in the greater part of the United States (a fact which appears to be overlooked by many writers) and, without going from home, the examples the gentleman asks for can be readily furnished and these entirely demolish any argument against the apex law founded on the Drumlummon case. A striking example is the bitter litigation about the zinc deposits at Franklin Furnace, N. J., lasting in various form nearly half a century. I have outlined this remarkable litigation elsewhere and it would be impossible to do so here within allowable space.⁴

Another recent example is the enormously protracted (about 3 months for one trial) and expensive (about \$60,000 for court costs alone) trial in the U. S. district court in New York city, which resulted in the conviction and sentence to the penitentiary for fraud by a mining company of Julian Hawthorne, and a doctor and an Austrian whose names just now escape me. There are very numerous examples of this kind equal to the Drumlummon litigation. I happen to know personally of a litigation arising in a rural county in Illinois about the corporate affairs of an Arizona mining company which lasted 13 years, and resulted in the wrecking of the company which previously stood in the not overcrowded rank of dividend payers.

A search would doubtless disclose in the legal history of the United States many other equally important mining litigations also not involving the apex law at all; so that it is utterly fallacious to argue that the Drumlummon case justifies the repeal of that law. From scattered hints in reading, I believe that other mining countries would furnish also many examples of just as important litigations. I have conclusively shown in my main contribution to this discussion, by an actual count of the mining cases, that the apex law causes much less than its proportionate share of mining litigation, instead of nearly all of it, as most non-legal writers seem to assume. The instances I give above prove that its lawsuits are not more protracted or expensive than other mining litigation.

If injustice resulted from the final decision in the Drumlummon case, as Mr. Goodale seems to believe, the fault lies not with the apex law, but with the existing instruments and methods of legal procedure. I am as strongly in favor as any one can possibly be of a thorough-going reformation of our entire legal system. It is foolish to try this kind of a case or, in truth, any kind of a case before a jury, if an honest and impartial judge can be found to try it; and, under a proper legal system, this could be done without fail. Our Anglo-American jury system is an archaic survival of an entirely different state of society and stage of

⁴ See Shamel's *Mining, Mineral and Geological Law*, p. 43 *et seq.*, where the citations to the original reports of the case are given.

civilization and is as crude and inefficient as the solid-wheeled carts of Mexico compared to a railway train.

True it is, as Fred T. Greene observes in discussing Mr. Goodale's paper, that if the dip of the vein changes, or, to state it more accurately, if the boundary of the ore body is a warped surface, its intersection with St. Louis's side line will be a bent line, but this is not any argument whatever against the apex law. If it has any bearing at all it is just the contrary; for it is one of the problems which will arise oftenest under the opposite legal principle because liable to occur whenever the vein passes under *any* boundary, whereas under the apex law it cannot occur more than one-half as often; that is, only when the vein passes out beneath an *end* line. If it passes out beneath a side line no jury or judge will have to worry about an irregular polyhedron, for the very simple reason that the miner can legally excavate as far as he sees fit without liability to anybody.

In reality, the situation instanced has absolutely no application in the apex controversy; and its attempted use is a striking example of the confusion of thought or desperate dearth of sound argument suffered by the opponents of the apex law. So far as the difficulty Mr. Greene mentions is an argument for anything, it is, like Mr. Goodale's, an argument for a radical reform of our entire legal system, or, perhaps, even for the abolition of courts altogether, allowing mining disputes to be adjusted by the primitive combat method. I would further observe, however, that so far as my experience goes I never saw either a judge or jury obliged to struggle with such a problem as calculating the cubical contents of an irregular polyhedron. The universal practice is, when such questions arise, that one or more competent engineers or other experts are placed on the witness stand and asked the question as to the contents of the irregular polyhedron of given dimensions. If he or they are competent, it is impossible that there would be any doubt about the testimony; and it would certainly be adopted by the judge and jury in arriving at a verdict. Consequently, such problems, even under our present legal system, would be decided according to the best knowledge of engineers or other experts; and the ignorance of judge and jury about descriptive geometry does not matter. In fact, justice can be more nearly attained, and I would rather try a technical or scientific case before a judge and jury entirely without knowledge of the technical or scientific principles involved, because if the expert who testifies and furnishes the technical information is mistaken or false it can be exposed and thrown out, but it is impossible to know and correct technical mistakes which may lurk in the minds of judge or jury supposed to be familiar with them. In my judgment, neither Mr. Goodale nor Mr. Greene advances any arguments justifying even remotely the proposal to repeal the apex law.

Good Ideas in the Mining Laws of British Columbia and Mexico

Discussion of the paper of F. L. SIZER, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 87, March, 1914, pp. 555 and 556.

SIDNEY J. JENNINGS, New York, N. Y.—This paper seems to open up what is desirable in the new law of the United States. I think that is a point on which the Institute would like to be heard when new laws are actually proposed by the government in the shape of bills before Congress. And it would be wise before any new legislation is passed to have a Commission gather the views and opinions concerning the desirable features of mining laws of other countries, and those which appeal most strongly to the operating engineers in the United States.

W. L. CUMINGS, Bethlehem, Pa.—Mr. Sizer's paper suggests something which I have not noticed in many of the papers on this subject, and that is, more comparisons drawn between the mining laws of the United States, and those of other countries, showing the good and bad features of each. We all know that there are serious objections to some features of the mining laws of the Spanish-American countries, such as, for instance, confusion arising from the classification of mineral deposits. This has given rise in Cuba to a lawsuit which compares in magnitude with many of the famous apex-law cases.

In this connection it would seem that descriptions of some of the complicated ore deposits of other countries could be used as examples showing the litigation that would undoubtedly have followed had the same been located in the United States.

WILLIAM B. MCKINLAY, New York, N. Y. (communication to the Secretary*).—1. The "square location." On account of the extremely irregular nature of some ore deposits, it would perhaps be restrictive to the point of injustice were a miner compelled to locate his ground in the form of one large square. In the various contributions which I have had the privilege to intrude in the extra-lateral wrangle, I have consistently urged that locations should be made in a form resembling the Mexican "pertenencia" system; i.e., blocks of ground bounded on the surface by true squares of moderate size (or rather by the lines of vertical projection of a true square upon the surface of the ground) and bounded laterally by vertical planes passing through the surface lines.

About 10 years ago there was introduced in the House of Representatives a bill prepared by me and containing some of the "innovations" which I advocated. The same bill, with a slight change, is at present being considered for introduction in the U. S. Senate, though at present overshadowed by matters of more urgent importance, such as

* Received Mar. 20, 1914.

the Mexican policy and the Canal tolls question. The present text of this tentative bill is as follows:

A BILL to amend sections twenty-three hundred and twenty, twenty-three hundred and twenty-two, twenty-three hundred and twenty-three, and twenty-three hundred and twenty-four of the Revised Statutes of the United States, and for other purposes.

Be it enacted by the Senate and House of Representatives of the United States of America in Congress assembled,

That sections twenty-three hundred and twenty, twenty-three hundred and twenty-two, twenty-three hundred and twenty-three, and twenty-three hundred and twenty-four of the Revised Statutes are hereby amended so as to read as follows:

Sec. 2320. Mining-claims upon veins or lodes of quartz or other rock in place bearing gold, silver, cinnabar, lead, tin, copper, or other valuable deposits, heretofore located, shall be governed as to length along the vein or lode by the customs, regulations, and laws in force at the date of their location. A mining-claim located after the passage of this act, whether located by one or more persons, shall be rectangular in form and may equal, but shall not exceed, four contiguous mining-blocks, each of which shall be a square with sides measuring, as near as may be, six hundred and sixty feet. Provided, however, if it be impracticable to procure blocks of true squares because of a previous valid location the new location shall be bounded on all sides by straight lines at right angles, except along the boundary of such location or locations. No claim shall be granted for an area less than that of a mining-block, except when, from any cause, a grant of that quantity cannot be made. No claim may be located until the discovery of a vein, lode, ledge, or deposit within the limits of one of the mining-blocks; though a single block may be provisionally claimed as a prospecting claim, in the manner described in Section 2323.

Sec. 2322. The locators of all mining-claims heretofore located or which shall hereafter be located, on any mineral vein, lode, ledge, or deposit, situated on the public domain, their heirs and assigns, where no adverse claim exists at the passage of this act, so long as they comply with the laws of the United States, and with State, territorial, and local regulations not in conflict with the laws of the United States governing their possessory title, shall have the exclusive right of possession and enjoyment of all the surface included within the lines of their locations, and of all veins, lodes, ledges, and deposits included within the territory bounded by vertical planes passing through the lines of their locations.

Provided, however, that the locators of mining-claims located prior to the passage of this act, their heirs and assigns, shall have the right to exploit or mine all veins, lodes, and ledges throughout their entire depth, which shall have been discovered by the said locators or their heirs and assigns and actually exploited or mined prior to the passage of this act, although such veins, lodes, or ledges may so far depart from a perpendicular in their course downward as to extend outside the territory bounded by vertical planes through the side-lines of the surface locations.

But their right of possession to such outside parts of such veins, lodes, or ledges shall be confined to such portions thereof as lie between the vertical planes passing through the end-lines of their locations. And nothing in this section shall authorize the locator or possessor of a vein, lode, or ledge which extends in its downward course beyond the vertical planes through the side-lines of his claim to enter upon the surface of a claim owned or possessed by another. Nor shall anything in this section be so construed as to authorize any locator or possessor of a mining-claim to exploit or mine, outside of the limits of the territory bounded by vertical planes passing through the lines of their location, any vein, lode, or ledge not discovered, exploited, or mined prior to the passage of this Act, even though the top or apex of the vein, lode, or ledge may

be considered to lie within the limits of the above-described territory. And the failure to exercise the right to exploit or mine beyond the side-lines of a claim during a period of twelve months at any time subsequent to the passage of this Act shall be considered as an abandonment of such rights.

Sec. 2323. A single mining-block may be provisionally located as a prospecting block or prospecting claim, before the actual discovery of a vein, lode, ledge, or deposit; and the locator of a prospecting claim shall have the right to sink shafts, run tunnels, and carry on other exploratory work for the discovery of mineral, within the limits of the block located. Locations of prospecting claims, conflicting with prior locations on which work is being prosecuted with reasonable diligence, shall be invalid. And failure to prosecute the work on a prospecting claim for six months shall be considered as an abandonment of the right to all undiscovered veins, lodes, ledges or deposits within the limits of the prospecting block. But the rights of the locator of a prospecting claim, as long as it is merely a prospecting claim, shall not interfere with the rights of the locator of a mining-claim to choose such location as he desires for the additional mining-blocks adjacent to the mining-block in which he has actually discovered a vein, lode, ledge, or deposit of mineral.

Sec. 2324. The miners of each mining district may make regulations not in conflict with the laws of the United States, or with the laws of the State or Territory in which the district is situated, governing the location, manner of recording, amount of work necessary to hold possession of a mining-claim or a prospecting claim, subject to the following requirements:

The location must be distinctly marked upon the ground so that its boundaries can be readily traced. All records of mining-claims hereafter made shall contain the name or names of the locators, the date of the location, and such a description of the claim or claims located by reference to some natural object or permanent monument as will identify the claim. On each mining-claim located after the passage of this Act, and until a patent has been issued therefor, labor shall be performed or improvements made during each year, to the value of not less than twenty-five dollars for each mining-block or fraction thereof. On all claims located prior to the passage of this Act, not less than one hundred dollars worth of labor shall be performed or improvements made within eighteen months from the passage of this Act, and each year thereafter until a patent has been issued therefor; but where contiguous claims are held in common, such expenditure may be made upon any one claim. And upon a failure to comply with these conditions, the claim or mine upon which such failure occurred shall be open to re-location in the same manner as if no location of the same had ever been made, provided that the original locators, their heirs, assigns, or legal representatives, have not resumed work upon the claim after failure and before such re-location. Upon the failure of any one of several co-owners to contribute his proportion of the expenditures required hereby, the co-owners who have performed the labor or made the improvements may, at the expiration of the year, give such delinquent co-owner personal notice in writing or notice by publication in the newspaper published nearest the claim, for at least once a week for ninety days, and if at the expiration of ninety days after such notice in writing or by publication such delinquent should fail or refuse to contribute his proportion of the expenditure required by this section, his interest in the claim shall become the property of his co-owners who have made the required expenditures.

It is not expected, nor is it entirely desirable, that the bill in its present form should be finally enacted as a law. Certain details have been incorporated in it simply for the purpose of exciting discussion, so that when the final draft shall have been prepared it will contain the

features which promise to afford the healthy stimulation which the mining industry needs.

The present laws establishing the method of location of mineral lands situated in what are known as the public lands of the United States of America were based upon mistaken ideas as to the nature of mineral veins and deposits. It was apparently assumed that all deposits existed in true fissure veins extending downward into the earth for an indefinite distance and rarely departing any great distance from the perpendicular.

It was deemed just, by the makers of the laws, that a miner should be entitled to mine throughout its depth any deposit whose apex or highest point he might discover in the course of working within the limits of his own claim; even though the deposit might extend beyond the bounds of perpendicular planes passing through the side lines of his own claim.

In the course of many years of mining work carried on since the enactment of the present laws, it has been learned that many mineral deposits have a character totally unlike that presumed in the original premises. As a consequence, many long and bitter fights have been carried on in the courts (and out of court), involving claims to ore deposits mined and unmined. The trend of the deposits, often difficult to determine even under peaceful conditions of working, has in some instances been greatly obscured by the dynamiting of worked-out stopes by miners who desired to give themselves the benefit of the doubt.

Were the limits set by the definite bounds of vertical planes through lines of surface ownership, any disputes as to the ownership of ore could be resolved by simple surveying.

In the States where there is no Federal public land the Federal mining laws do not apply. Thus, for example, in the State of Michigan, in Houghton county, it was possible for the Tamarack Mining Co. to sink a shaft through several thousand feet of barren rock in order to tap the celebrated conglomerate copper deposit worked from the surface by the Calumet & Hecla Co. Had the Federal mining laws applied in that district, the Calumet & Hecla would have enjoyed a monopoly and have left as future ore reserves the deposit worked by the Tamarack Co. But, since rational laws governed the title to the said deposit, employment was given to many thousands of men and the sinking of the deepest mine shaft in the world was later justified by the results obtained by the Tamarack Co. Also, the sinking of the great Whiting shaft was undertaken by the Calumet & Hecla Co., partly due to the encouragement induced by its neighbor's success.

While it is desirable to limit the rights of possession of ore deposits by providing that all future locations shall give definite possession within the definite bounds of vertical planes, it is desirable that vested rights (previously granted by laws which, however irrational they may have

been, nevertheless were the laws of the land) should not be taken away during the enjoyment of said rights. But it is not compatible with the principles of our government to continue to sustain monopolistic privileges of an indefinite character for the benefit of those individuals or corporations who neglect or refuse to exercise the indefinite privileges either for their own benefit or for the benefit of the community in general.

In my opinion it would not be correct to assume that the annulling of an unexercised indefinite privilege would constitute *ex-post-facto* legislation, because from the very nature of the extra-lateral right privilege the fact of the existence of ore beyond the side lines of a claim is not demonstrated to be a fact until the ore itself is uncovered and mined.

It is not claimed that the amendment of the mining laws will prove a universal panacea for all the ills which at present afflict the mining industry. But amendment along the general lines as suggested by the draft which I have submitted would certainly tend to stimulate prospecting activity and would force the owners of idle mines to either work their own mines or give some one else the right to search for mineral, with reasonable protection covering the enjoyment of such deposits as they may find. It is extremely disconcerting to spend money in the development of a property and ultimately discover that the vein has its highest point under the surface of another, with the result that the apex owner seizes all.

In any event, legislation should not be undertaken too hastily. Men cannot be made honest by legislation. Nor should legislators assume that all men are dishonest by inclination. In cases where this latter assumption is made, the new freedom will tend to resemble a modified application of the British criminal system of ticket of leave.

2. The suggestion of the appointment of official inspectors or Commissioners is an excellent one. Where Federal lands are to be inspected, it is rational that the inspectors should be Federal officers rather than county or State officers. The enlargement of the Bureau of Mines, with a possible elevation to the status of a Department of Mines, seems to me to be the best solution.

I know of no person better fitted to plan the adaptation of the Bureau of Mines to this class of service than the present Director of that Bureau.

When the Congress shall have solved all the urgent matters of trust regulation, inter-oceanic transportation tolls rates, foreign relations, etc., perhaps they who dominate that joint legislative body will see the wisdom of authorizing the Director of the Bureau of Mines to select a Commission of practical miners and mining engineers to assist him in drafting the ideal bill.

Why the Mining Laws Should be Revised

Discussion of the paper of HORACE V. WINCHELL, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 88, April, 1914, pp. 645 to 664.

HENNEN JENNINGS, Washington, D. C.—I have been so long out of the country that I have not had an opportunity for getting any mining company into title difficulties, or urging new laws, or giving any expert advice upon the subject of the laws of the apex; so I can make my statement very dispassionately and impartially.

I was connected with mining in South Africa some 16 years, and thus had an opportunity to become acquainted with the mining laws of the Transvaal under the Boer government.

I was greatly impressed with the main principle of these laws: namely, the conveying of certainty of title. All claims were held on the basis of a surface area pegged out, and with the downward continuance in accordance with vertical-plane projections of the boundary lines.

There were two classes of claims, known as digger's and prospector's. They were held under a leasing system of payment of so much a month, but no limitation was made as to time of holding so long as the necessary payments were made. The payment for digger's license was larger than for the prospector's, the higher price being for ground that was under exploitation. One person could peg off but a limited area; but there were no laws preventing amalgamation of claims, which was done on a big scale; but greedy and speculative holding of claims was minimized by the constant necessity of paying licenses for all claims held, whether worked or not.

It is true that in this way most of the claims got into the hands of strong corporations, but it is also true that the original locators had a good chance to sell at a good price, and the nature of the deposit was such that it required a great capital outlay to profitably equip and work a mine.

From what I have heard and read, it would appear that the law of the apex was framed to a large extent by the prospectors of California and Nevada, who no doubt believed it was a system that was advantageous to themselves.

Now, has this really been the case? How many of the prospectors who pegged out the ground have worked their claims to a finish, and benefited by the extra-lateral rights given under this law? Have not litigation and uncertainty of title been detrimental features in selling out to capitalists? Have the capitalists benefited as a whole by these rights when the many and costly lawsuits are taken into consideration?

I want to know also, whether the public has benefited by this law, and what has been its influence upon morality in mining. Has it not

had the tendency to lower the moral tone, by making secrecy almost a necessity, and to tempt the honest man to make location on a very flimsy basis in order to guard against blackmail?

Who, then, has it helped? Possibly a few professional mining experts may have obtained remunerative employment, but the best of them have at the same time decried the law. I believe it has benefited the blackmailer and the faker; and the lawyer has not lost by it.

In Europe an American mine that has not gone through the fiery furnace of a lawsuit is considered a risky possession.

Thus it would seem that even to the original locators and first holders, the law has been something of a boomerang.

SIDNEY J. JENNINGS, New York, N. Y.—There is one point I would like to bring out. We have heard a great deal about apex. We have not heard so much about the definition as to what is a vein, and that is a point with which I have had some considerable experience in my work in the United States.

You would think that a definition of a vein was a reasonably easy thing, but in a case in which I was interested, not as a witness nor as an expert, but simply as a worker of the property after it had been acquired by a favorable decision, this matter was up.

A fissure was found on the surface carrying mineral in payable quantities. It was followed down, and at a certain horizon ore was found to make on a bed of limestone. The location of the fissure on the surface was held by one individual, and the location of the apex of the favorable bed of limestone was held by another individual. Then the question was, which was the vein, the limestone bed, or the fissure where the ore was originally discovered and followed down? This was fought for many years, and the decision of the Supreme Court was made to the effect that the limestone was the vein, as it was the place where large bodies of ore were always found. There were numerous fissures going through this bed of limestone, and wherever those fissures got into the limestone large payable bodies of ore were found, consequently the limestone and not the fissures was the apex. That strikes me as ingenious, but it is difficult to predicate how far some other ingenious theory may carry another individual in upholding what he supposes to be his right under the extra-lateral theory.

I am of the opinion that, as Dr. Raymond said, it is wise to concentrate your opposition on one point. If you have a concerted attack on one point it is not apt to be overthrown, whereas if you have a scattering attack against various wrongs, you cannot make as big an impression. But there are so many points which, while not as objectionable as the apex right, are as well worthy of being changed, that I would be inclined to include in our opposition one or two other features mentioned by Mr. Winchell in his paper, rather than restrict ourselves

entirely to the attempt to persuade the law makers to do away with the apex theory.

HENNEN JENNINGS.—In the Transvaal companies were formed on the dip, one behind the other, and called deep civil companies. You would think that there would be great dissatisfaction among the first people who pegged out claims when the deposit went out of their boundaries. As a matter of fact, this was not the case, for although they would have been glad to have had a greater length of line, there was no bitterness of feeling on this score either among corporations or individuals, as they knew their rights and risks at the start. There were no lawsuits or ill feeling, as has been the case with the apex law. I do not think the deep levels would be developed to-day as they have been if held only by the companies that first started.

It would have lessened the progress of development, and caused uncertainty and litigation. Our legislators seem to fear that by limiting downward rights, they are injuring the original locators. It is not so, if you give them at first latitude to peg out liberally and as much as their resources will allow them to work. The extra-lateral right stopped mining instead of helping.

GEORGE OTIS SMITH, Washington, D. C.—It may seem presumptuous to add anything to what Mr. Winchell has said about the reasons for revision; but I would suggest reason No. 10, as it has occurred to many of us, and as it was suggested to me by C. B. Dutton, of the Bureau of Mines, namely: That we have confusion worse confounded by reason of the fact that the States and the local districts have added to the Federal codes their special State laws and district regulations, so that the result of some of these has been really to obstruct what might be expected as the favorable operation of the Federal law.

HORACE V. WINCHELL.—When you come to the consideration of the question of a single, definite reform, laying all emphasis on the one particular revision desired, you will find it so interlocked with other provisions of the law as to make it almost impossible. You spoke of abolishing the apex law, and so you must give a wider piece of land. Senator Smoot introduced a bill providing for the abolition of the extra-lateral right. What would have been the result? You cannot abolish the extra-lateral right without abolishing the right of discovery. They are all tied up together. The only way is to investigate the whole field, and frame a harmonious and consistent theory of mineral law, and then try to get it enacted. You want to get a theory which will give the prospector the security of a possessory title while looking after his mineral, and unquestioned, undoubted rights afterward.

Why should it be a matter of question as to what a prospector or miner is entitled to? If he knows he has what is immediately under the

surface, everybody knows it. But as it is to-day, there is trouble, doubt, question about it; questions of uncertainty, hesitation, and the consequent lack of development. We all know that something is the matter with prospecting. We all know that mines will not live forever. We know we are exhausting our mines at a tremendous rate. The United States, with a population of 6 per cent. of the people of the globe, occupying 6 per cent. of the land area, produces 30 per cent. of its minerals. Now we cannot keep that up unless we continue to find mines, and we cannot find mines unless mining titles are made secure.

JAMES KIRKPATRICK,* Dillon, Mont. (communication to the Secretary†).—Many people will be glad to see better laws enacted and will hope for speedy results. That the prospector is an important individual and that he naturally has much to contend against goes without saying. In his discoveries he should be safeguarded to the utmost from the rapacity and greed of those who are ever ready and watchful to take advantage of his omissions or neglect. The ordinary requirements of mining law seem sufficiently burdensome to his pioneering spirit and he is usually unable and unwilling to fight a legal duel against heavy odds. The result is too apparent for comment.

It is probable that half or more of the present unpatented mining claims in this State are subject to re-location because of inaccuracy of measurement in their boundary lines. The average prospector is under the impression that, should his boundary lines extend beyond legal allowance, he runs no risk but will have the privilege, if necessary, of diminishing them; but the court has held in at least two instances (*Hauswirth vs. Butcher*, Montana Reports, Vol. 4, p. 299; and *Leveridge vs. Hennessy*, Pacific Reporter, Nov., 1913, p. 907, that more than a slight over-measurement of boundary lines voids the location. Thus, if the locator of a Montana lode claim, through inability or ignorance, fails in accuracy of measurement, his claim, regardless of time or money expended, is exposed to the cupidity of any shark. It has come to be common practice for the big fish thus to gobble up the little ones. Justice as well as regard for the welfare of the mining industry of this State dictates that, in case of excess, the locator should still be protected in the amount of ground contemplated by the mineral-land laws.

Claims are frequently located in difficult, even almost inaccessible, places where, without assistance or facilities for measurement, the locator is obliged to approximate his directions and distances and, even in favorable localities, cannot always procure a rope, or afford to employ a surveyor.

It is to be hoped that the mining societies, congresses, and committees will succeed in impressing upon the Congress of the United States the

* Non-member.

† Received Feb. 10, 1914.

necessity and importance of at once revising our moss-grown mining laws. The results of the efforts will certainly be watched with great interest throughout the West.

E. B. HOWELL,* Butte, Mont. (communication to the Secretary†).—I do not believe in the extra-lateral rights doctrine any more than do mining engineers generally, but prospectors usually do, and if you try to effect too revolutionary changes in this respect you are sure to fail.

I would make the following suggestions:

1. Abolish the distinction between quartz and placer claims by making the government price for each \$5 per acre. This will terminate the present "known lode" infirmity of placer titles. Any mineral land that is worth locating at all is worth \$5 per acre. The price of land, paid to the government, is a mere bagatelle in the cost of mining, anyway. There was never any good reason for making placer land only one-half as much as lode claims and to make the price uniform will result in simplification of mining law.

2. Make all claims 600 by 1,500 ft. in maximum dimensions and make no limitations as to how the claim must lie with reference to the apex of the vein, and restrict mining operations to the vertical planes of the boundaries. In other words, if a prospector is sure his vein pitches south he can lay his claim so that the apex will lie just inside his northerly side line. Or if he wants to be sure of more of the vein on the dip let him turn his claim at right angles to the apex, letting the outcrop be close to one end line. In this way he can have nearly 1,500 ft. horizontally within which to pursue his vein.

3. There should be uniform provisions as to the method of locating, marking boundaries, and recording claims. I think we have the best law relating to these topics in the State of Montana. The Federal law makes no requirement relative to the marking, except that the claim must be so marked upon the ground that its boundaries can be readily traced. The Montana law specifies, as *prima facie* evidence that the claim is so marked, the placing at each of its corners of some one of several prescribed forms of monument—for instance, a post 4 by 4 in. in size and 4 ft. 6 in. long, set one foot in ground, with mound, etc. A post firmly set with a mound of rock about it makes the best corner; but it should be remembered that in some mining districts of the West it would be very difficult for a prospector to secure posts. Hence, alternative methods of marking should be provided. Where posts of a smaller size than those prescribed in Montana are used, it becomes a question for the jury as to whether the claim is so marked upon the ground that its boundaries can be readily traced, but the locator does not necessarily lose his claim.

* Non-member.

† Received Feb. 10, 1914.

4. I would abolish the present method of locating quarter-sections of placer land by associations of individuals. That is unnecessary for legitimate placer mining operations. My observation has been that such locations are made by seven-eighths dummy locators and one-eighth land grabbers. If placer claims are 600 by 1,500 ft. in size they can be laid along the channel of the stream—which is quite necessary for placer operations.

5. There ought to be something done to prevent corporations from going into new districts and locating all the land solidly with claims having common end and side lines. I have known of this being done in some instances. It is a perversion of the spirit of the mining law, which might be prevented by prescribing that no locator shall, either by himself or through the agency of any other person, locate more than, say, two claims within a radius of one mile of his primary discovery.

I think that these suggestions, if followed, will result in a simplification of the law without any revolutionary changes. In some respects it would resemble the present law in the Philippine Islands.

The Initiation of Title to Mineral Lands

Discussion of the paper of ALBERT BURCH, presented at the New York meeting, February, 1914, and printed in this (June, 1914) *Bulletin*, pp. 1165 to 1168

M. K. RODGERS, Santa Monica, Cal.—I would like to call attention to a very important thing in this paper, and that is the survey of the claim being made immediately after location, or as soon as the prospector cares to do it. In British Columbia that is permissible, and it is allowed as one year's assessment work.

GEORGE A. PACKARD, Boston, Mass.—Speaking of a survey, I want to refer to the change in the manner in which prospectors and locators are doing work on their claims. Two months ago I went to Montana to look over claims for which patent was asked two years ago; and there were two discovery holes, about 50 ft. apart. One of those claims had been passed for patent by the Interior Department, and the other had not.

I will admit that in one claim there was good quartz-bearing ore, and in the other only some stringers; but at the same time there was a mining claim within 2 miles which had produced \$20,000,000 according to a record I had seen. One of these claims had been turned down for patenting as "non-mineral land." It had been relocated and the owner was driving a useless cross-cut tunnel in an endeavor to find some quartz or copper to satisfy persons sent out by the Department to make the examination.

There were some eight claims, so located that a tunnel on one of the veins, with cross-cuts, would have developed all very advantageously.

But the owner had been obliged to spend \$500 on each individual claim, and to throw it away, as he had been told by the engineers who made the survey for the patent that if he made an expenditure of \$4,000 all in one place, which would be greatly to his advantage in developing the property, undoubtedly the claims would not be passed for patent. That is one thing which has arisen as a result of change of policy in the Department.

Now, taking up the matter of survey of the claim: I agree with the gentleman who has just spoken, that that is an extremely desirable thing, and it should be permissible that the claim should be surveyed and the cost of survey be applied as a portion of the assessment work, of the \$100 worth of work, if we are to have such a feature in our law. But I do not think survey of the claim should be made obligatory. There are many prospectors who have not the money to pay for a survey of the claim, but they can go out and do \$100 worth of work on the property; and that is of some value in making the property salable, while the survey is not as valuable.

We have not enough prospectors in the West to find as many mines as the mining investors of the East would like to buy if they could see something worth buying; and I think it would be a great mistake to put into the law any requirement which would make it more difficult for the prospector to go out and establish title to his claim. I think the law should be such that the *initiation of the possessory right should be as simple and inexpensive as it possibly can be, and yet at the same time give us a record, and a title, which would be unquestionable.*

HORACE V. WINCHELL, Minneapolis, Minn.—I am reminded that there are surveys and surveys. Last summer I visited Patagonia. I had a map furnished by the Argentine government in Buenos Aires. I traveled 3,000 miles, and each applicant for an oil concession had to advance 1,500 or 2,000 pesos for surveys. They did so. I organized an expedition and went out, and as I approached I saw white stakes, but there was not a single mark on a single stake to show what had been surveyed.

An Oil-Land Law

Discussion of the paper of GEORGE OTIS SMITH, presented at the New York meeting, February, 1914, and printed in this (June, 1914) *Bulletin*, pp. 1169 to 1172

EUGENE COSTE, Toronto, Canada.—I am interested in what Dr. Smith has said, and in the fact that you are working earnestly in the United States toward a good oil-land law, which is certainly very much needed. Nothing could foster the oil industry in those parts controlled by the Federal government more than a good law.

I would like to make one or two objections, if I may call them so to Dr. Smith's statements. He is, I think, in favor of making a too complicated law. I would prefer a simple law, rather than one that is complicated, even though it is not so good. The fact of its simplicity makes up for its imperfections. I do not see the use or necessity of first issuing a permit to prospect oil lands and then afterward a lease. You cannot explore oil land, except in the one way, by drilling wells, and therefore I think it would be better for everybody concerned if the law were such that an applicant could get his final lease promptly without going through the form of getting a permit first, the lease for which land may not be issued later. You cannot expect any one to spend much money on simply a prospecting permit, his title not being sufficiently secured.

I will cite an instance of that. In Canada we have regulations, not law regulating Federal land, but Orders in Council; that is, the government makes certain regulations, called in this case "Petroleum Regulations." Up to a short time ago an applicant could tender \$5, and he could hold up 1,920 acres of land for one month. During that month he had to pay 25c. an acre, and then he would get his lease. A lot of speculators took advantage of that provision, and paid \$5, never intending to pay the 25c. an acre, just to have a sort of a title for one month, and they would hold up the *bona fide* oil men who wanted to get the land for drilling. Thousands upon thousands of acres were plastered in that way.

The Interior Department got into such a tangle that they finally saw the point which I had brought to their attention vainly years before, and they now have changed the regulations; now you must pay 25c. an acre at the time you make your application for the land, and this payment insures one an absolute title for 21 years, and is renewable for another 21 years, provided, of course, one meets the other obligations of the lease. There is no doubt that this new regulation, which insures one a good title lease right in the start, and which will prevent the speculator from coming in and holding up oil lands, will be much to the benefit of the *bona fide* operator who intends to spend thousands of dollars on his lease. All he wants is to get a simple but safe title, with no provisions left to the administrative officers to decide whether he will be permitted to go on after spending much time and money. Law is made by Congress here, and I should have no such matters left to the discretion of departmental or governmental officers; and anything that arises in the way of dispute should be settled by judges in the courts.

While this is true, the law should incorporate all the safeguards possible. And whether there should be a royalty or a rent paid, it matters not. It works well both ways, and makes very little difference. But there should be a strong condition of drilling, so that there would be no mere speculation. In Canada the government remits the second a

third years' rentals, which are 50c. an acre, on proof that more money than the rentals due has been spent in drilling. There is no royalty; on the contrary, the government is offering a bounty. The remittance of the second and third years' rentals, if you can show that your expenditures have at least equaled these rentals in the work of actual drilling, is a good provision, as it lets you put your money into wells, which are, of course, the ultimate source of all wealth in oil fields. Besides there is a compulsory provision in the leases that you must be drilling a well on every three sections after 15 months from the date of the lease.

DAVID T. DAY, Washington, D. C.—I want to ask one question of Mr. Coste. In this country we are becoming embarrassed by the foreign ownership of American oil lands, and it is generally understood that some protection is being arranged in the disposal of Canadian land by regulation or by law. I would like to hear something on that point.

EUGENE COSTE.—The changes made in that respect are so recent that I am not quite certain of their purport. But I think the only change that has been made lately is that there are provisions that the Board of Directors of a company acquiring oil lands from the government should have a majority of their members residing in Canada, and a few matters of that kind. In the recent regulations the majority of the Board of Directors must even be Canadians or Englishmen. A similar provision was already in the laws of the Province of Ontario. While I am speaking about oil laws and mining laws I might remark that in Canada we are in about the same fix as you are here in regard to our mining laws. That is, I mean the Federal laws. We have had good laws in the Province of Ontario and the older Provinces, but in Saskatchewan, Alberta, Manitoba, and the Northwestern Provinces, which are still under Federal law, we have as yet no mining law, and we have only regulations by Order in Council. That has been most detrimental to the industry, and very hard on mining men, as these regulations are made by a clerk in the office of the Department of the Interior who knows nothing about mining and are then passed on through the Minister of the Interior to the Council, and the result is that you have something that is absolutely ludicrous just like the provision mentioned above where by paying \$5 one could get 1,920 acres of land from the government, and hold it for a month, and after that month lapses he could pay another \$5 and hold another 1,920 acres for another month. Then five or ten people could get together and hold these lands from month to month simply by paying \$5 a month for each 1,920 acres. Some definite, clear and simple title lease for the *bona fide* applicant who really wants to go to work, at so much an acre, with drilling provisions, is all that is wanted. But for these operators the title should be absolute, without any possibility of impeachment. If I had my way the leases issued would be only a half page in length, giving the applicant

title, absolute title, and requiring from him so much rental per acre and so much work in drilling, and that is all.

HENNEN JENNINGS, Washington, D. C.—I think Mr. Smith was with us yesterday when we discussed mining laws, and the confusion and uncertainty that have arisen in connection with the laws of the apex. I think unintentionally he is putting elements of uncertainty in these suggestions which may develop troubles akin to those which were discussed yesterday.

I am entirely sympathetic with a lease that is in the nature of a practical ownership, that people can rely upon. But I do not believe in backing and filling or giving the government any power to back and fill in dealing with people who start to develop a large enterprise. You should make up your mind as to what the government is willing to do at the start in a given locality; and if you want a lease system, make it fixed and determined, and not leave it to the discretion and periodic adjustment of officials; it was shown yesterday in our discussion, that even with honesty of purpose and ability the scientific opinions of men vary as knowledge increases and facts accumulate.

I entirely agree with the fundamental aim set forth by Mr. Smith, that the consumer should obtain the oil at a minimum price, but with minimum waste.

The gentleman who has just spoken [Dr. Coste] of the Canadian practice seems to be perfectly sound in his recommendations, and they are in accordance with the practice which I have been accustomed to see work most satisfactorily on a large scale in South Africa. The leasing system of the gold mines in South Africa is a definite thing. You know your rights and risks from the start and your lease is negotiable. You are prevented from tying up land for speculative purposes by a definite fixed monthly charge per claim area. The area under exploitation pays a higher license than the ground unworked, but this has a lease tax that makes it a burden and detriment to hold unless provision is being made for its early exploitation. Why is not a similar system possible and desirable in petroleum leases? Make the area of holding generous, but with constant fixed, unalterable term payments for all ground held, whether worked or not, and with a definite but not excessive royalty per barrel for all oil extracted. The royalty might vary for each district, but when once fixed it should not be changed.

I will not go further into detail, but I will ask Mr. Smith to consider the possibility of amending his suggestions to meet some of the views which have been expressed.

GEORGE OTIS SMITH.—With regard to the suggestion by Dr. Coste and also the suggestion from Mr. Jennings, I agree that we should have a simple law which should lead to a simplified contract. I have suggested a law that in certain terms might vary with different conditions.

I have not suggested a contract which would vary after the contract was entered into; except that I have suggested a sliding scale of royalty in order to protect, not the government but the operator.

It would be a sliding downward scale, in order to continue operation under more favorable circumstances, rather than to force the stoppage of oil production because it would not pay the cost of production plus the royalty, as fixed in the contract. I do not believe there is any suggestion made in Washington to have anything less definite than the ordinary contract which is made in business.

I absolutely agree with Mr. Jennings. I said in my paper yesterday that any uncertainty involves an unnecessary expense in operation and financing, and therefore it should be avoided. The reason for having a permit, and having it followed by a deed, is because a prospecting permit should be for a larger acreage than that which is to be written into a lease.

The suggestion under consideration by the Secretary of the Interior, and by the Chairmen of the two Congressional Committees, referred to by Senator Walsh in yesterday's session, is to provide a permit for an area which will be four times the grant to the successful prospector. In other words, if you take 640 acres under a permit, you will receive in fee simple title to 160 acres; the remaining three-quarters of the square mile being available for leases under definite terms, fixed by the government, by or before the time of the lease.

Under those conditions I think it is necessary to have a permit as well as the later granting of title, both in fee simple and on a leasehold. I say so because it is more favorable to the prospector. I think that with the government as the lessor it will work out better than with a lease from a private individual.

I have faith in the integrity and the sincerity of government officials, as ordinary American citizens. I will say further that I believe the oil men of California have as much confidence in government officials as they have in some of the landlords who have secured title from the government simply to impose hard conditions on the real oil men who operate under leases secured from oil men who are not producing oil men.

Mining Law Revision. How to Obtain It

Discussion of the paper of E. B. Kirby, presented at the New York meeting, February, 1914, and printed in this (June, 1914) *Bulletin*, pp. 1159 to 1162.

J. B. TYRELL, Toronto, Canada.—It is rather an onerous duty I am asked to perform here in telling you how our attempt was made to draft a new mining code for the Dominion of Canada. The Dominion of Canada

does not own all its wild land. Most of the various Provinces, like many of your States, own their public lands, so that the Dominion only owns a certain proportion of the more distant vacant land. These lands are covered by the Dominion mining law.

A few years ago the Canadian Mining Institute suggested to the Dominion government that it might be allowed to co-operate with the government in the drafting of a new mining law for the Dominion, in the hope that such a mining law might not only be useful to the Dominion of Canada itself, but might also serve as a model for the various Provinces of Canada, and that sooner or later the Provinces would pass laws which would closely approximate this model law which we were to draw up.

The Dominion government threw the responsibility back on the Canadian Mining Institute and said to it: "You draw up the Act, and if possible we will pass it for you." That was under a government which two years ago went out of power, and since the new government has come into power, while it has considered our draft sympathetically, it has not passed it into law.

We hope it will take that draft into consideration during this year and enact it into law, because it embodies, as closely as we can determine, the kind of law which we think would be best for Canada.

Now, the constitution of the committee that was appointed to draft this law was about as follows: We chose three or four of those actual mining members of the Canadian Mining Institute whom we thought had most experience and were most competent to express opinions on what a mining law should be. We also added a couple of lawyers to the committee who were to furnish the legal assistance in doing the work and in putting the drafts into proper legal form. It was not their duty, nor did they assume, to dictate what that law should be, but they were to assist us in framing the law; so that it would be a law which would be considered as being properly drawn by the courts.

I am not going to give you any synopsis of that draft. It is still a draft, though we hope to have it enacted into law this year. Whether it will be or not, I do not know. But what I have said may give you some idea of the way in which we undertook the preparation of it. We have endeavored to draw up a mining law for the Dominion of Canada covering the Federal lands, which we hope will serve as a model, and be copied by the Provinces as soon as possible, so that we will have fairly harmonious mining laws throughout the entire Dominion.

In Canada, we do not sympathize with the apex law, which as I understand it, if I understand it aright, was a product of an erroneous interpretation of Cordilleran geology. Leaving out the ore-bodies in the Rocky mountains and considering only those occurring in the Pre-Cambrian and other great formations throughout the eastern portion of the country, I do not think that the apex law would have suggested itself to anybod

HORACE V. WINCHELL, Minneapolis, Minn.—I wish to call attention to Senate File No. 4373. Feb. 5, 1914, Mr. Smoot introduced the following bill [not included in this report], which was read twice and referred to Committee. Now, in connection with the bill, I would say that it has been suggested that this Commission shall consist of three members, all of whom shall be lawyers, experienced in the practice of mining law. And there is a provision regarding a third party, constituting a member of the Geological Survey, that is eliminated. And the provision that said code shall not deal with coal and other minerals has been crossed out. The Commission *may* hold hearings, instead of being compelled to, as in the original law. Will Senator Walsh advise us as to the interpretation of this amendment, and others which he may have in mind at this time?

SENATOR THOMAS J. WALSH, Helena, Mont.—The exactions incident to my official duties have been so continuous in character as to deny to me any opportunity for reflection concerning any subject that might be of special interest to this assemblage and in respect to the discussion of which my views ripened by study could be regarded as a contribution. I shall, in this situation, content myself with presenting a brief review of legislation now pending in Congress, which from its nature will evoke the attention of the members of the American Institute of Mining Engineers and the men with whom they come into contact professionally.

The Committee on Mines and Mining of either House of Congress has not heretofore been regarded as much more than one of the numerous paper committees, created and preserved chiefly to afford to the Representative designated as its Chairman, a needed clerk. It bids fair at present, however, more because of the important legislation affecting the mining industry that crowds upon the attention of Congress than by reason of the personnel of its membership, to be transformed from a more or less moribund organization to an active legislative force. The haste in which this *résumé* is necessarily prepared impels me to confine my remarks to measures that have been referred or which it is expected will be referred to the Senate Committee.

The Revision Commission Bill.—It has had under consideration a bill introduced by Senator Smoot, for the creation of a Commission, charged with the duty of revising and codifying the laws in relation to the appropriation and occupation of mineral lands of all kinds, a subject which, as I am advised, has had repeated consideration by this body. As the bill came to the Committee the Commission was to consist of three members, one of whom should be a lawyer of experience in the practice of mining law; one a mining operator, and one an officer of the Geological Survey. Being referred to the Department for an expression of its views, it was proposed that the membership be increased to five, giving the Bureau of Mines a representative and an additional representative to the operators. It was, however, urged before the Committee that as the work

devolving on the Commission is the preparation of a draft of a code of laws, that in the past have given rise to interminable litigation and that are likely to be a fruitful source of perplexity to the courts for years to come, it ought to be entrusted exclusively to lawyers of eminence and character, conversant with the troubles that have been encountered in the past, and most competent from this experience and learning to devise a system under which unnecessary controversies may be avoided in the future. While the suggestion might appear to have originated in a desire upon the part of the members to take care of their brothers of the law, a little reflection will exhibit much merit in it. It was advanced that the Commission being authorized to hold public hearings throughout the mining regions affected, and elsewhere, at will, it would unquestionably invite an expression of views from operators, engineers and prospectors, from representatives of the Geological Survey and the Bureau of Mines. The bill contemplates that a report shall be made before Jan. 1, 1915, and provides that each member of the Commission shall be paid a salary of \$500 per month for the time actually devoted to the work. It is now over 40 years since our system of laws in relation to the disposition of mining lands was devised. Generally speaking, it has met the conditions in a most satisfactory way, excepting always the part dealing with the disposition of lands containing coal. That part of the law never would have been operative at all had not the practice now condemned as criminal been pursued. As the appropriation of coal lands has all but ceased in consequence of the failure of the law to recognize that an entry of 320 acres will not justify the expenditure necessary to the mining of coal on a commercial basis, a revision in respect to lands valuable for coal is imperative. Aside from that feature, however, the conviction is quite general that the extra-lateral right gives rise to complications so numerous and serious, it is such a prolific breeder of litigation, not infrequently characterized by imposition and perjury, that it ought to be abandoned. The idea in which it had its origin—a just purpose to stimulate the prospector by assuring to him the vein discovered through his sagacity and his self-denial—was most commendable. But in practice it often served to deprive the enterprising miner of the legitimate fruit of his toil and his expenditures. Every lawyer of experience is familiar with instances in which ore bodies of great value have been uncovered by expensive development made by the owners of the claims in which they are found, and who have been obliged to yield them up upon the claim of an apex in adjacent territories, the owners of which had no suspicion of their existence and who never exhibited enterprise enough to find them.

It will be found profitable to compare the working of the system to which we have become attached, with those that have been developed during the past two score years in the English colonies, the Latin

American republics and other nations that have been required to legislate concerning the public mineral lands. An enlightened public sentiment concerning our mineral-land policies can be formed only in the light that is afforded by knowledge of the kindred systems of the progressive peoples of the earth. A compilation of such laws, accompanied by a brief explanation of their character and the measure of success which has attended their working, is all but essential to comprehend the active value of our own, to make appreciable such defects as have not obtruded themselves upon our notice, and to perfect our system in the respects in which it appears faulty. However competent any one or more members of Congress might be to undertake or perform such a task, it is simply impossible, except they neglect duties no less imperative, to devote to the work the time necessary to accomplish it. In all probability the Committee will report this bill favorably.

The Alaska Coal Land Bill.—Some of the questions with which the Commission will be called upon to deal, should one be appointed, brook no delay. They must be met at once and solved in some way looking to revision later in the light of facts brought out or views advanced by those deputed to study the subjects as a whole. Of this character is the question of how to open the Alaska coal deposits. The bill providing for the construction by the government of railroads in that Territory from the seaboard to the inland waterways has passed the Senate and will speedily, it is believed, be approved by the House. President Wilson's sanction of the general purpose of the Act has been given in advance in his message to Congress. It must be accompanied by an Act which will permit and invite the workings of the coal fields. It is idle to imagine that the Executive Department will advance one step toward the construction of railroads in Alaska unless assured in advance that the coal mines will be producing to afford fuel for construction purposes and tonnage when the road is built. The Senate Committee has under consideration a bill which is the result of repeated conferences participated in by the Secretary of the Interior and his staff and the Chairmen of the House and the Senate Committees respectively on Public Lands, Territories, and Mines. It is a frank adoption of the leasing system, to which none of those having any conspicuous part in the preparation of the bill confess any attachment. The Congressional members are all convinced, however, that the concession is essential to the necessities of the case. They entertain the view that to enact a law providing in any terms for the disposition in fee of coal lands in Alaska would be in effect to doom the country to another decade of inaction. I am myself of the opinion that any Secretary of the Interior who should permit Alaska coal lands to be patented under any kind of a law passing the fee would be driven from public life, and it would be good fortune if he did not pull down the temple of the administration of which he formed a part as he passed out

of it. It is to be regretted that a great public question such as this is cannot be debated dispassionably and without impugning the motives of those who adhere to what may be said to be the Western view, except when and in so far as the conditions point to selfishness in those upholding it. The promoters of the leasing plan never admit the possibility of an honest difference of opinion concerning the wisdom of the departure they propose. All who oppose them are involved invariably in the common denunciation which is leveled at the plunderers of the public domain and the defenders and apologists for such. There has been no deliberate judgment by the American people on this supremely important question. They have been turned by invective from the consideration of it. The leasing system was tried in connection with the lead lands of the Mississippi valley, and abandoned after proving a dismal failure. Probably the essential differences in the conditions under which it was tried and those now prevailing deprive the experience we had under it of anything like controlling force, but if so the public has not been advised as to why the same troubles are not in store for us. Calumniation has been so long the portion of those who have sought to gain a fair hearing for the system of alienation in fee that few remain who have not despaired of securing considerate attention to the merits of that plan. Anyway, the friends of Alaska are without hope of convincing the public mind, at least not speedily, that as to coal deposits, popularly believed, and doubtless with justice, to be exceptionally rich, the leasing system is not a most doubtful experiment. The people of Alaska pray for any kind of a law which will permit them to mine coal. They have ceased to debate the merits of rival systems. It must be confessed that the plan of reserving title in the government is growing in favor. The Western Representatives are much more tolerant of this idea than they once were. The legislation of such States as Colorado and Montana concerning their own coal lands, forbidding their sale but authorizing the operation of them under leases, is appealed to with embarrassing effect. It is true that the policy of the State is primarily to make as much money as it can out of its school lands, while the policy of the Federal government is, or ought to be, to make its lands serviceable in the development of the sections in which they are situated. But the State is not altogether unmindful of its larger interests in the sale of its granted lands, and the policy of local development is as well subserved under a leasing system, if all the avails are devoted to improvements within the State, instead of going to meet the general demands upon the Federal treasury. The Northern Pacific Railway Co. has the most vital concern in the rapid development of the territory adjacent and tributary to its lines. But it has also adopted the leasing system as to its coal lands and declines to sell such at any price.

Those responsible for the Alaska coal-land bill referred to find sufficient justification in the foregoing to tender a measure recognizing the leasing

system. Under it the Secretary of the Interior is authorized to withdraw eight sections in the Bering River coal field and twelve sections in the Matanuska field for the use of the navy, or to be used in connection with government construction work, or for disposition by Congress in case of oppressive conditions arising from monopoly, it being contemplated that the government might in the future deem it the best solution of difficulties which might arise similar to those encountered in connection with the anthracite coal situation in the East, itself to supply the market from its own fields. Leases are authorized for indeterminate periods, with provisions for readjustment of prices every 20 years, no lease to be for more than 2,560 acres to any person or corporation. Drastic provisions are inserted in the law to prevent evasion of the provision limiting the area in which any individual may be interested. Forfeiture and imprisonment are the result of conviction. It is believed the risks are so great that few will care to take the chances involved in an effort at monopoly. The royalty is fixed by the lease, but cannot be less than 2c. per ton, and it all goes toward the redemption of the bonds issued for the construction of railroads in the territory and for purposes of a similar character when the obligation created by them is discharged. The lessee is entitled to sue the government on any cause of action arising out of his lease in the courts of Alaska—so that any arbitrary or destructive policy on the part of the Department may be restrained, and questions arising upon the construction of the lease may be determined as they would be between private parties under like circumstances. To compel the operation of any ground leased, a rental in addition to the royalty is exacted at the rate of 25c. per acre the first year, 50c. an acre annually for the next four years, and \$1 an acre for each year thereafter. For local use the Secretary is authorized to issue permits without any charge for the working of tracts not to exceed 10 acres, the purpose being to permit the homesteader and miner to secure coal for his own use at an adjacent bed.

The Radium Bill.—A third bill before the Committee which has given rise to much discussion appertains to lands containing radium-bearing ores. These are mainly pitchblende and carnotite. The former is often, perhaps usually, found in metal-bearing veins and in association with the precious metals as well as with zinc and lead; the latter, so far as known, appears only in veins or pockets in sedimentary rocks. The carnotite ores are the chief source of supply. They abound in an area reaching from Colorado into Utah, 150 miles long and varying in width from 5 to 25 miles. Discoveries have been reported in Arizona, Montana, Idaho, and South Dakota, but no authentic information is available of any workable fields save those of Utah and Colorado. The wonderful advance made in the use of this remarkable agency in the eradication and cure of cancer and other malignant growths has directed the attention of the

world to the sources of its supply. The exceptional riches of our Western fields have made the output of them eagerly sought after by the laboratories and reduction works of Europe. Two great plants located in the State of Pennsylvania are now treating the ores and claim to be supplying the trade each with a gram a month, salable at about \$120,000. Thus far their production has gone very largely to foreign markets, as has practically all the ore not treated by them. They own the claims from which their supply is drawn—one of the companies having acquired about 140 claims. Another association has 80 odd; a third, 40 or more. Though the crude ore which is the source of by far the greater part of the products of the works in Europe comes from this country, our surgeons are forced to go there to procure their supply. Unmistakable evidence is at hand of something like a race to purchase or locate every deposit of any prominence—and surgeons of eminence who are intensely interested in the success that has been achieved in the use in therapeutics of this singular substance, and the still more marvelous possibilities that experimentations are constantly revealing, became genuinely alarmed on visiting the region from which the ore comes lest the whole of it should speedily fall into the hands of a monopoly which might maintain the price of its product so high as to make it available for the treatment of those only who could afford to pay anything that might be demanded. Before the American ores entered the European market the chief source of supply for the Continent was the pitchblende ores of Joachimsthal, but the Austrian government prohibited the exportation of any ores containing radium—preferring to preserve the supply of the Empire for domestic use. The German Emperor is making extraordinary exertion to procure an adequate supply for the hospitals in his country.

These conditions prompted the introduction of a bill which obligates the locators of all claims containing radium-bearing ores to sell their products to the United States at the market price to be fixed by the Secretary of the Interior. Development work or mining during a period or for periods aggregating four months are required, and even after patent, if the property is not worked with diligence, the Secretary is authorized to enter upon the property and mine it, paying to the owner market value of the ore less the cost of extraction. Rights to claims located prior to the passage of the Act are, of course, not affected, but the government is given the preference right to buy the output of all claims hereafter located. The Bureau of Mines is perfecting a process which it is claimed will enable it to reduce the ores at a cost not to exceed that now attending similar work by private companies.

Two questions are of first importance in the consideration of this measure—first, whether it is justifiable to impound all ores produced in our territory to supply the needs of our people; and, second, whether the

peril of monopoly is sufficiently imminent to justify the government in itself undertaking the work of extraction.

Oil and Gas Bill.—Various other bills related in character to those referred to will be considered in conference, such as has been mentioned, and then introduced. First in importance is an oil and gas bill. The basic feature in this bill is one which permits the discoverer to purchase a limited portion of a tract for the exploration of which an exclusive license is issued to him, the remainder then being leased. It is proposed to give the adventurer the exclusive right for two years to prospect over four sections, if they are distant 100 miles from a producing well, and over one section if it is less. Should he discover oil or gas, he is entitled to locate, in the one case 640 acres, in the other 160 acres, of the land included in his permit, acquiring title under the procedure prescribed in the case of placer mines.

Potassium and Sodium Compounds.—The same principle is to be made applicable to the disposition of valuable deposits of chlorides, sulphates, borates, or nitrates of potassium or sodium.

Phosphates.—Phosphate deposits it is proposed shall be leased by competitive bidding in tracts not to exceed 320 acres. The royalties from all leases of lands outside of Alaska, except so much as is deemed necessary to defray the cost of administering the laws (for this principle applies to all leasing measures), go into the reclamation fund.

Coal.—A general coal-land bill has been prepared by the Department on the same lines.

The Committees of both Houses, the Department, and the Congress will be thankful for any consideration you may give these various measures and for any helpful suggestions you may be moved to make concerning any of them.

HORACE V. WINCHELL.—Now, gentlemen, you will see something is doing over in Washington—many things we do not know anything about, and which we should know about. If we are in the mining business, as some of us suppose we are, these laws are of vital import to our business, as well as to the welfare of the country. The first thing of interest, it seems to me, is in connection with this Commission, under the resolution introduced, and I would like to know if there are any resolutions to be introduced here for our consideration at this time.

E. B. KIRBY.—The number of members suggested for this Commission by the American Mining Congress was five. It was not attempted to specify as to their character, except in private character the people who were considered the best; the idea was to have the President appoint a Commission of five, and then it was expected when it reached the President there would be an opportunity for mining men and members of our Institute, and the Congress, to present their own recommendations as to the character of members suitable for the Commission. There was a

general fear that any attempt to dictate to the Congress or to the States would arouse opposition. The original idea was for a Commission of three lawyers.

GEORGE A. PACKARD, Boston, Mass.—It seems to me it would be a mistake to allow this bill or the suggestion as to this Commission to be read before this meeting, and have nothing said in regard to the suggestion that the bill be amended, and that the Commission should consist of three lawyers.

I think there should be some expression at least to the effect that mining communities should be recognized. It seems to me the original scheme was a most delightful one. I do not know how they would like to have four mining men and two lawyers, when the two lawyers are only expected to advise as to the legal method of preparing the bill. There are lawyers in this country who would be glad to go as members of the Commission, but I do not think we want it to be considered that in the sense of this meeting that we would like to be represented entirely by lawyers. I think the majority should be mining operators, miners or mining men.

DENIS M. RIORDAN, New York, N. Y.—My predilections were in favor of a Commission of five, and I would be in favor of one of the Commission to be a lawyer. I think the three mining societies, one representative of the government representing the Bureau of Mines, and the Geological Survey, would make a well-balanced and well-considered Commission. But I think the time has come when such a Commission is indicated, and I am in full accord with anything that will result from the ambition of the man who has done so much to make the country of the Missouri what it is to-day, namely, the prospector.

HENNEN JENNINGS, Washington, D. C.—I think the mining men should immediately admit that the lawyer is more competent to draft a bill than himself. But as to the substance of the bill, it seems to me that the locators, owners, engineers, the operators, should know better than the lawyer. Therefore in a Commission of that kind it should say that a larger committee would be better than a small one, being of course understood that the legal representative should draft the bill after the substance had been arrived at by questioning of witnesses by those who best should know the work, and after a conference and discussion of capable and honest representatives of various sides of the question. I think this would meet with more general satisfaction and stand better the future test of time.

It is a vast subject for three men with no practical experience. No matter how able they may be to grasp everything connected with the subject and law.

SENATOR WALSH.—I feel it to be my duty to advise this gentleman that I am not at all responsible for the suggestion of the amendment in regard to the character of the membership of the Commission, and

I am free to admit that the merit of the proposition appealed to me very strongly. It was suggested to me by a member of the Committee, of well-known ability and wide experience, and he told us why he suggested it. He said he was appointed a member of a Commission in his State for the purpose of revising their laws with reference to mining. He was made a member of the Commission with two other gentlemen, who were not lawyers. He saw or thought he saw, from a lawyer's standpoint, some very serious obstacles to the working of the plan they proposed. He said in fact that he was confident—and the result showed the soundness of his judgment—confident of his position, but his views were overruled by the other two members of the body. The reason for that is that men not engaged in the profession of the law could readily suggest a line of action which it would be impossible to put into force, because, possibly, of constitutional objections. And the lay member would not always appreciate the force and applicability of decisions which had been rendered by the courts, and which would be controlling on the interpretation of the proposed statute's provisions. And it was suggested that it would be construed in a way to defeat the very purpose for which it was drafted.

Now, it is really a work of drafting, and inasmuch as the Commission will have available for its services the views and opinions of all those who are interested, it would seem not to be necessary at all that any of us should be represented on the Commission. But you will bear in mind about that, with respect to that matter, you gentlemen represent only one particular branch of the mining business. The prospector, who is perhaps greatest in number, in the revision work would have no representation upon the Commission at all, in all probability. And you are likewise to bear in mind that he does not always look on you gentlemen as having interests absolutely identical with his own. And so there is a political aspect to this thing which must be borne in mind. I must say I haven't any very fixed conviction at this time with respect to the wisdom of the change proposed, but I rather feel that perhaps some gentlemen feel it was a rather invidious amendment directed at them. I do not think that is the case at all.

HENNEN JENNINGS.—In case the Commission had conflicting views, who would decide? You have to take testimony, and you must adjudicate as to its worth. Now, if you do not have interested people on the Commission, how are you going to arrive at a judgment that will be universally satisfactory?

SENATOR WALSH.—The decision would be arrived at in the same way, whatever the composition of the Commission, as the report, if it is reported, will be a majority report; the only trouble is that with three lawyers, there would be three separate reports.

The Segregation and Classification of the Natural Resources of the Public Domain

Discussion of the paper of FREDERICK F. SHARPLESS, presented at the New meeting, February, 1914, and printed in *Bulletin* No. 88, April, 1914, pp. 713.

R. W. RAYMOND, New York, N. Y.—The very clear and logical position of Mr. Sharpless is based on premises not yet established. It assumes, in the first place, that the legal purpose of the classification of public lands, as carried on hitherto, has been to fix their character separately for purposes of *administration*; whereas, the so-called "mining" of the United States is simply a set of rules for *selling* the public mineral lands, and has no more direct relation to mining than the agricultural-land law has to agriculture. It creates particular classes of lands, as mineral lands containing lodes, placers, salines, coal, petroleum, and prescribes the way in which tracts of each class may be temporarily occupied, and ultimately purchased, and defines the nature of the title to be conveyed to the purchaser. Its purpose, like that of the agricultural-land law, is to transfer the public domain outright to its citizens in such a way as to encourage private enterprise and discourage monopoly. Since it fixed different units of area, different forms of procedure, different limitations of tenure and title, and different prices per acre for different classes of public lands, it was highly important for the explorer or investing citizen to know in which class the United States as land owner had placed the particular tract he desired, so that he might know what kind of title he could acquire, and by what method, under what conditions, and at what price. Obviously it would have been quite sufficient for this purpose if the United States had simply said "This we call mineral land, and will sell it under the mineral-land law. If you find it more valuable than the price you paid, that is your lookout. A bargain is a bargain." That is what an honest private land owner would have done. And a part of it is actually done by the United States. It requires of the mineral-land locator that he shall discover valuable deposits before acquiring title; but if it turns out afterward that the deposits were not valuable, so that the purchaser was deceived (as is actually the case with the majority of mining claims patented hitherto), the United States does not dream of returning to him the cost of surveys and proceedings which he has sustained. In that case, a bargain is a bargain. But if a tract of supposed agricultural land, granted as such, turns out to be more valuable for mining, the United States tries to take away the title from its own grantee, on the ground of this later discovery, the title it has granted him. An illustration may be found in the suits now pending against the Southern Pacific R. R. Co., attempting to annul the title of that company to certain lands granted to it, and of all the people who have purchased parts of such lands, on the ground that petroleum has lately been found

in them, or so near them as to warrant the classing of them as petroleum lands. The basis of these suits is the provision in the grant to the railroad that it does not include mineral lands, and the claim that the mineral character of the land, no matter how, or how much later, established, should be held to vitiate the original grant. No doubt this proposition has plausible legal force, since nobody can deny that a thing explicitly reserved in a deed was never granted; and if mineral lands were thus reserved from railroad grants, the only defenses which the railroad company can now set up would seem to be (1) that petroleum is not a mineral, and hence that petroleum-bearing lands are not mineral lands; and (2) that, whatever be the value of the first defense, the word mineral, as used all over the Pacific slope, at the time when the grant was made, meant *ore*—i.e., a solid, including mechanically, or in chemical combination, a valuable metal—and that, at that time, petroleum-bearing lands, though known to exist in California, as well as in Pennsylvania, etc., were never included among “mineral” lands. Upon the common-law principle, that a deed or contract expresses the mutual understanding of the parties, and must be construed according to that understanding, it may be, in my opinion, justly held that petroleum-bearing lands, not being regarded at the time of the grant as “mineral” lands, were not included in the aforesaid reservation. This I regard as a sufficient defense of the railroad company and its innocent grantees in the cases I have mentioned; but I think it is an outrage that, after forty-odd years of peaceful, *bona fide* possession, and after innumerable sales of so-called agricultural land to innocent purchasers, the original title given by the United States should be assailed on such grounds. Simple honesty, such as obtains between man and man, would require that the United States, as grantor of the land, should determine, before its sale, in what class it had decided to place it, and then stand by its own decision.

I could give many other illustrations, all carrying one moral—namely, that after all precautionary measures and forms have been duly complied with, the title to land, given by the United States, should be definite and indefeasible. The new policy, advocated by some, of perpetual government ownership of mineral rights, is not in question here. Even the advocates of that policy must agree that, so long as the present system of selling mineral lands outright is continued, the government ought to give clear and final title to purchasers. For this purpose, it is relatively immaterial whether its “classification” be scientifically correct or not, if it only binds the government and protects its grantees.

In a word, the argument of Mr. Sharpless has noteworthy force only on the assumption that the United States is expected to administer its natural resources permanently as the property of the whole nation, and not to transfer them to private citizens or to the several States. Concerning that policy, I have elsewhere expressed and supported my views in other contributions to the Institute.

Chlorides in Oil-Field Waters

Discussion of the paper of C. W. WASHBURNE, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 87, March, 1914, pp. 375 to 381.

EUGENE COSTE, Toronto, Canada.—I agree entirely with Mr. Washburne's view that the strong salinity of these waters in the oil fields is due to solfataric volcanic emanations; the composition of these waters is not only the large excess of chlorine in them but the excess of magnesium and calcium, differentiates them entirely from sea waters and they can be compared with magmatic waters. In the Texas and Louisiana oil fields we certainly have the clearest examples of solfataric action which we have in the oil fields of the United States—I mean in the salt domes of Louisiana and Texas, where secondary products, not only of salt in solution, but enormous bodies of dolomite, gypsum, and sulphur, have been deposited in vertical chimneys by solfataric hot waters accompanied by hydrocarbons and sulphuretted hydrogen; the salts being impregnated with natural gas, sulphuretted hydrogen, and oil, and the salt waters of the oils being quite hot yet in some cases.

A. C. LANE, Tufts College, Mass.—Mr. Washburne's paper brings forward some interesting points; and his main thesis, that these are derived from within (juvenile), is one well worthy of careful consideration. Does he not, however, assume that the early sea water had a character too much like that at present? Judging from the average water now received by the ocean, either according to Murray or Clarke, there must have been a gradual change in its composition, as Hunt, Goessman and others have pointed out, not only in increasing salinity but in the relative proportions of the different salts.

It would be interesting for him to extend his collections of analyses to a wider range of geological horizons. Those cited are Paleozoic. His paper on the chemical evolution of the ocean, yet unpublished, I have not seen. I have no reason to believe that while underground circulation tends to increase up the ratio of Na : Cl, the character of truly stagnant water varies little with its geological age.

The trouble with following his suggestions and connecting the calcium chloride waters of the Lake Superior mines with the highly calcic magmatic waters of the greenstones associated is that the calcium chloride waters are not just the same in the felsitic conglomerates interbedded and also in the removed sandstones, e.g., near Whitefish lake and Ontonagon, in Michigan iron mines, and in the Storm King granite. If we are to look for an earth sweat for these universal waters, then I think it would be better to assume the chloride merely to be slowly diffused from the interior, picking up its bases *en route*. But a serious difficulty there is that we have, for instance, under the oil-field waters referred to by Washburne, the St. Peter sandstone, with a water which is much fresher, appearing

cutting off any prospect of a juvenile source, which otherwise I should be delighted to accept.

The objection to Washburne's theory is the fact, if it be a fact, that different geological horizons have different chemical characters, and that if not too much disturbed by circulation they can be recognized thereby.

Cementing Oil and Gas Wells

Discussion of the paper of I. N. KNAPP, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 87, March, 1914, pp. 471 to 488.

ARTHUR KNAPP, Ardmore, Pa. (communication to the Secretary*).—*Cementing Short Strings of Large-Size Casing*.—The necessary fittings, such as back-pressure valves, nipples, by-passes, etc., for 12½-in., 15-in.,

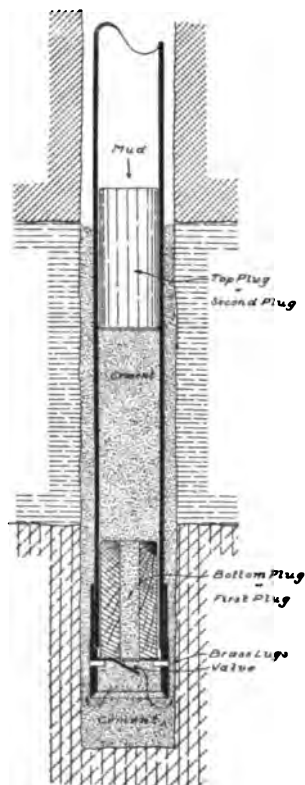


FIG. 1.

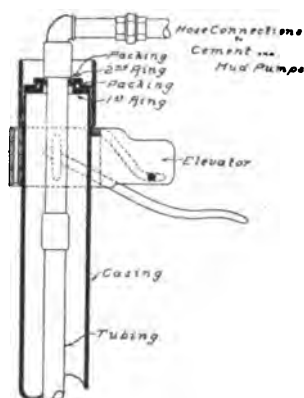


FIG. 2.

or larger size casings are very expensive. These strings are seldom more than 500 ft. long, and cementing may be safely accomplished as follows (see Fig. 1):

* Received Jan. 29, 1914.

First Plug.—This is a cylindrical wooden plug with a length about twice the diameter. A hole whose diameter is one-third that of the plug is bored through the plug. The bottom of this plug is fitted with a circular piece of belting the same diameter as the casing. This belting is so cut as to form a flap valve over the lower end of the bored hole in the plug.

Second Plug.—This plug is a cylinder with a length about twice the diameter, and the customary belting is nailed on its lower end.

Lugs.—Four brass lugs about $\frac{3}{4}$ in. in diameter are inserted in the casing shoe in such a manner as to project into the casing about $1\frac{1}{2}$ in.

Operation.—The flap valve on the first plug is held shut by several small nails. After mud circulation is established around the casing this

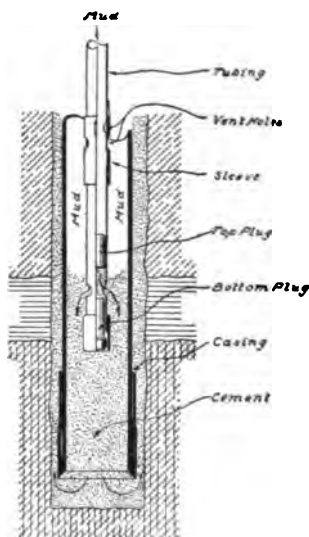


FIG. 3.

plug is inserted and the cement pumped in on top of it. The second plug is then put in on top of the cement and the pump again started. When the first plug reaches the brass lugs inserted in the shoe (see Fig. 1) it can go no further. The pump pressure then opens the valve and forces the cement through the plug and into position outside of the casing. When the second plug reaches the first plug the pump pressure immediately increases, showing the cement to be in place. The casing is then lowered to bottom and cementing has been accomplished. It is good practice to leave the pump pressure on the casing for some time to prevent any back flow, should there be a leak around the shoe.

Cementing Very Long Strings of Casing.—The use of tubing ($2\frac{1}{2}$ to 4 in.) to pump cement through in setting long strings is good practice. The contamination with mud which is liable to occur in the larger casing is

materially reduced in the tubing. The total inside surface of 2,500 ft. of 8-in. casing is 5,250 sq. ft., and of 3-in. tubing 2,000 sq. ft., the ratio being as their respective diameters.

Fig. 2 shows the top casing collar. A steel (never cast-iron) ring is screwed into this collar, with a hole through it large enough to pass a tubing collar. A second ring is fitted to a tubing nipple with packing inserted to pack against the tubing collar. Packing is inserted into the first ring and on it the second ring sets, thus effectually sealing the top of the casing and preventing back flow inside of the casing.

Fig. 3 shows the bottom of the tubing and casing. The tubing is fitted with a collar into which is screwed a bushing. Two holes are drilled into the tubing far enough above the collar so that when the bottom plug hits the bushing there will be just space for the cement to pass out above it. This figure shows the top plug at the instant before it reaches the bottom plug and shuts down the pump, thus indicating that all of the cement has passed from the tubing. When the cement has all been pumped into place the casing is lowered to bottom and cementing has been completed. The tubing is now plugged at the bottom and if pulled would be full of mud. It is often pulled this way. Fig. 3 also shows a sleeve which may be used to vent the tubing by setting the string on bottom and allow the string to be pulled dry.

After the casing is on bottom the hose connections are removed and a longer nipple is screwed into the top of the tubing. The string of tubing is then lowered to bottom. This causes the sleeve to slip up on the tubing and bring the vent holes into line. If now the hose connections are again screwed on, the cement left in the bottom of the casing may be pumped out. This leaves a minimum of cement to be drilled out after the casing is set and is an advantage over the method of using full-sized plugs. With this method of using tubing there is no danger that an accident to the cement pump will leave the cement where it will set in the casing or tubing. The vent at the bottom of the tubing will allow the circulation of mud to wash superfluous cement from the well.

O. P. Hood, Pittsburg, Pa.—In the rotary system of drilling the circulating mud performs several functions. It removes the cuttings from the path of the bit and gives a clean surface for the bit to work upon, it brings the cuttings to the surface where they can be examined, it may plaster the wall of the hole and also penetrate some of the formations.

Mr. Knapp calls special attention to the plastering effect, but seemingly overlooks the matter of penetration into porous formations. If mudding a well refers to plastering the wall of the hole, as implied in the criticism of *Bureau of Mines Technical Paper 42*, then a circulation is indeed necessary to distribute the mud thickened by the cuttings along the length of the hole. There is, however, an action of muddy water quite apart from plastering the walls of the well and which does not depend upon

circulation. A fluid may flow from a formation into a bore hole of a reduced pressure at the hole, but the direction of this flow reversed by creating a fluid pressure in the hole greater than that of the formation. The amount of radial flow will depend upon the pressure difference, the porosity of the formation, and the viscosity of the fluid. A muddy fluid of such character that fine clay will stay in suspension and finally clog the small and tortuous passages by deep penetration into the formation.

Maintaining a sufficiently high static pressure on a column of fluid, even without circulation, will therefore produce an effect in the formation. By releasing the pressure sufficiently the passages can be cleaned by the superior pressure within the formation. This is independent of any plastering of the surface and of any wobbling of the drill pipe.

The use of mud fluid dates back to at least 1889, but has been restricted to such conditions as described by Mr. Knapp, where it is believed fundamental that such mud should circulate down the drill pipe, up the walls of the well, and overflow to a mud pit at the surface. Reliance on the radial flow of the mud, and particularly as applied to the hole method of cable-tool drilling, is believed to be recent and has had numerous successful applications in the Oklahoma field.

RALPH ARNOLD AND V. R. GARFIAS, Los Angeles, Cal. (communicated to the Secretary*).—Under the heading "Difficulty of Getting Information," Mr Knapp makes the following criticisms on *Technical Paper No. 32, Bureau of Mines, Cementing Process of Excluding Water as Practised in California*, by ourselves. He first takes exception to the following recommendation made by us for the accomplishment of a successful cementing operation:

"The well should be cleaned of all débris and mud, and enough clear water be pumped in to wash out any sulphur or alkaline water."

We do not know what is considered good practice in Pennsylvania, but the above recommendation seems to be warranted in California. It is followed generally by up-to-date cementers here; at all events, if Mr Knapp is correct, we err in common with many well-posted California operators. In corroboration of this and for ease of reference, we quote only from the very interesting paper by F. W. Oatman¹ presented before the Institute at the same time as the paper just discussed.

"The water string is raised a foot or so from bottom and water pumped inside the casing and up to the surface around the outside of the casing. The

* Received May 4, 1914.

¹ Water Intrusion and Methods of Prevention in California Oil Fields, No. 87, Mar., 1914, pp. 531 to 553.

of this is: To assure the operator that there is free circulation; to wash away the mud, oil, etc., from the sides of the casing and the hole, so that cement may adhere thereto; to open the space around the casing so that a complete ring of cement may form. This water circulation may be continued for a considerable period, *sometimes as much as 30 hr. or more*, to clean the hole thoroughly."

Of course, one readily can imagine a combination of unusual conditions which might render the pumping of clear water always injurious, and perhaps such prevail in Pennsylvania, but one should not lose sight of the fact, as indicated by the title of our paper, that the information contained therein applies only to California.

Mr. Knapp also takes exception to that part of our fifth recommendation reading as follows:

"Previous to cementing, some definite information should be gained relative to the action of the water in the hole on the setting qualities of the cement,"

and he regards as "not to the point to talk about the action of salt, alkaline, sulphur, or gypseous water or oil in the hole on cement . . ." The writers fully agree with Mr. Knapp upon the uselessness of only talking about these matters but they strongly favor the advisability of gaining some definite information as to the action on the cement of the water in the well.

Again quoting from the instructive paper of Mr. Oatman, p. 542:

"Considerable experimental work has been done by many investigators to determine the effect of various salts on rate of setting. Calcium chloride, CaCl_2 , decreases time of set, and if over 2 per cent. by weight is used is likely to so hasten the set as to make the cement difficult to handle. The use of $\frac{1}{2}$ per cent. of sal soda, Na_2CO_3 , has been found effective in hastening the set in wells troubled with gas.

"Since hot water is encountered in many oil fields, its effect on cement is worthy of notice. In general it may be stated that the time of setting decreases with increasing temperature."

If the physical and chemical properties of the water in certain cases, however few, affect the behavior of cement, as noted by Mr. Oatman, it is obviously important to ascertain these facts before cementing, particularly when such an investigation usually requires little expenditure of time and money.

Mr. Knapp also objects to the following from the same paragraph:

"An excess of gypsum in the water might delay the setting of ordinary cement three or four months."

The statement as made is correct, for an excess of gypsum in the water *did* delay the setting of ordinary cement three or four months, as explained below by Prof. C. W. Wing, head of structural engineering at Leland Stanford, Jr., University, in a letter to the junior author:

"In the basement of the Royal Insurance Building in San Francisco some difficulty with the setting of the cement was experienced. This basement was some six or eight

feet in places below the ground water lines. . . The floor was a foot or more in and when the time came for placing the water proofing coating it was found concrete in the floor had failed to harden, although at the time I visited it it was than a month old. In some of the depressions seepage water was standing. In investigating the causes of the slow setting of the concrete it was found the building was in line with a gypsum spring that used to exist on the hillside building. An analysis of the seepage water showed that it contained large quantities of gypsum, and briquette samples of cement stored in this water would not harden and reach the same strength that identical samples stored in the water gave. . . . While this hardening had been delayed there was no reason to think that in due course of time the concrete would not reach its full strength. . . . The basement, on my recommendation, was finished as originally designed, and it has since I know has never given any trouble. The effect of the gypsum, therefore, was to delay the hardening of the concrete and not to destroy its value. . . . The results of tests with Portland cements in which the only gypsum is that contained in the cement would determine the effect of gypsum in the water upon the rate of setting of cement."

It should be noted by the wording of the paragraph in question that this condition was not intended to represent an every-day occurrence but one which might be repeated, and the writers deemed it advisable to caution operators against such a possibility. We were very glad to be shown "to be shown" when Professor Wing brought this to our attention.

Mr. Knapp then takes issue with statements made by us in *Paper No. 42*, published by the Bureau of Mines. As the quoted statements do not convey the complete meaning intended by the writers, we quote in full the paragraphs in question:

"Sealing of Artesian-Water or Gas Strata"

"It sometimes becomes necessary to exclude from the drill hole water or gas in an overlying sand in order to control the well or to recover or test for the gas. This end can be easily accomplished in drilling with a rotary rig by forcing water to the bottom of the well through one of the lateral openings of the casing, the pressure being regulated by closing the valve connected to the other opening. The back-pressure valve in the drill pipe will prevent the flow of the mixture through the casing. By drilling a short distance and forcing as much mud as possible through the porous sands penetrated, a plastered wall of clay is built throughout the gas or water-bearing strata. After the entire thickness has been penetrated and the mud has landed in an impervious bed, another string may be used to tap the oil sand. The gas thus excluded can be recovered afterward by drilling shallower wells through the upper sands.

"This method can also be employed in drilling the entire thickness of a sand under tremendous pressures, an operation that might otherwise prove impossible. The mud can easily be removed by releasing the pressure or bailing out the water in the hole, a strainer or perforated casing being placed in the sand to prevent the wall of the well from caving when the pressure is released."

Mr. Knapp's opinion to the contrary notwithstanding, the above description represents actual conditions which have been put into practice by J. A. Pollard and other petroleum engineers.

I. N. KNAPP (communication to the Secretary³).—The discussion of Messrs. Arnold and Garfias of my paper on Cementing Oil and Gas Wells has been carefully read to me. Also, I have read F. W. Oatman's paper and I do not think it has been interpreted correctly by them.

They quote Mr. Oatman as washing out the well, presumably with *clear water*, "to open the pace around the casing so that a complete ring of cement may form." No one can by any practical means direct the action of the clear water so that it will produce this ideal condition, which must be approached to successfully cement in well casing. Clear water by its solvent, erosive, and undirected action will tend to wash away the walls of the well at points of least resistance and work away from the ideal condition.

My paper explains in detail how to bring a well into condition for cementing, so I will not repeat. If Mr. Oatman is working in territory where the walls of a well will stand 30 hr., washing with water, he is to be congratulated on not having difficult territory to handle. He is again quoted (p. 542) as to experimental work to determine the effect of various salts, and troubles with gas, oil, and hot water in cementing wells.

As I have shown in my paper, all ground water, oil, and gas may and should be excluded from a well before cementing, so it is not necessary to consider their action on cement unless the driller deliberately elects to let them bother him.

Now as to hot water I did not discuss this in my paper, but I have drilled into a water-bearing horizon of a known temperature of 99° F. with the rotary, using a mud circulation. This mud, as coming from the well until drilling stopped, did not exceed 72° F. as noted by a standardized thermometer. There would have been ample time to have cemented without much increase in this temperature. This was clearly shown by the fact that when the well was bailed and the hot water warmed up the casing it expanded fully 10 in. in 1,880 ft. Thus hot-water conditions admit of easy control.

The quotation from Prof. C. W. Wing's letter does not distinguish between "setting" and "hardening" of cement or concrete. It is not made clear that the gypsum spring was the cause of the concrete failing to harden, as the same result might have been produced in other ways. I would refer to my paper (p. 471) as to the difference between "setting" and "hardening" of cement. I still object and desire "to be shown" that:

"An excess of gypsum in the water might delay the setting of ordinary cement three or four months."

Let us not forget the main point, as yet uncontroverted, that all ground water, gas, and oil may and should be excluded from a well before cementing.

* Received May 14, 1914.

The Origin of Petroleum

Discussion of the paper of HANS VON HÖFER, presented at the New York February, 1914, and printed in *Bulletin* No. 89, May 1914, pp. 869 to

A. F. LUCAS, Washington, D. C.—I have long held the view that igneous rock in the form of laccoliths, batholiths, or sills make up the salt domes of Louisiana, Texas and elsewhere, and have advanced the hypothesis of the volcanic origin of these domes. In the face of the apparently conclusive arguments presented by such authority as H. v. Höfer, it takes much courage or almost convincing proof to advocate such an hypothesis, but when one travels over the Great Plains of Texas, Louisiana, and Mexico, one needs still more courage, as there are certainly no visible evidences of vulcanism for hundreds of miles, only perfect and level plains of unconsolidated sedimentary materials.

Over these plains there rise some elevations of various heights. By comparison with the wide extent of the plain may be likened to a prick on the floor of this room. A series of these elevations apparently in perfect alignment northwest to southwest are now known as the Salt Islands in Louisiana, and others scattered through the Coast Range are known as hills, mounds, or domes. On some of these elevations there exude from low depressions solfataric emanations from acidulated water, while on others there are no indications of distillates coming from below.

I pioneered the drilling on all these islands and most of the hills, and invariably found them to be underlain in whole or in part by salt masses. This is the case also at Belle Isle, Anse la Butte, Spindletop, Bryan Hights, etc., on whose domes oil and sulphur was also found.

On Belle Isle strong indications of solfataric emanations were apparent, more so than on other domes of the series, and therefore this island was more extensively exploited than any other, with the exception of Spindletop, first for salt, and then for oil.

I have held the opinion with Mr. Coste that the emanations from the salt through these domes constitute evidences of volcanic origin, and in consequence have been desirous of making a practical demonstration to prove, if possible, whether or not the salt masses of the Coast Range rest on volcanic plugs. Accordingly I induced I. N. Knapp to drill a deep test well to ascertain, first, the total depth or thickness of the salt mass, and second, the character of the rock formation upon which the salt rests.

This was easily said, but not easily done, as in one of the test holes drilled by me some 18 years ago at Jefferson Island, for example, at Jefferson, drilling was carried to a depth of 2,100 ft. without

through the salt. The total thickness of pure rock salt, as far as explored, was nearly 1,900 ft.

Since the drilling of the discovery well by me on Spindletop, a salt dome, in 1901, creating a new precedent in oil exploration on the Coastal Plains, many attempts have been made to find oil by drilling on all sorts of elevations, however slight, in hope to strike another Spindletop, which was only 12 ft. above the surrounding prairie. In most instances the drill encountered either rock salt or dolomite which continued to considerable depth. As oil was not supposed to be present below these substances, the drill was stopped and the well abandoned.

There was therefore at that time no precedent at hand and no geologist willing or able to advance a possible hypothesis as to how deep the salt mass may continue, or upon what kind of rock it might rest.

Many theories and hypotheses have been advanced by students or geologists to account for the origin of these wonderful salt masses. In studying the interior of salt mines which have been carried to a depth of several hundred feet, and also in considering the perimeter of the salt domes, one cannot help concluding that the salt was not deposited by evaporation, but must have been deposited by saline waters ascending from great depths. In looking over a vaulted chamber in a salt mine of the Coastal Plains one is struck by the dazzling whiteness of the salt, which is interspersed, however, by dark streaks resembling the grain of oak or chestnut. These dark streaks are caused by very fine grains or stringers of gypsum, making it appear as if the whole mass was at one time in a plastic condition.

The Knapp well at Belle Isle passed through the salt at about 2,900 ft. and then continued in calcareous rock to a depth of 3,171 ft. At this point the drill encountered a formation so hard that no impression could be made with the ordinary fishtail bit and consequently no sample of the rock could be obtained. While endeavouring to obtain a diamond drill outfit, the well (which was at that time in great distress owing to gas pressure and caves) was lost and no other deep well has been drilled since that time.

The information this well gave is or should be gratifying, for although it failed to furnish a sample of the hard rock at 3,171 ft., it is likely that it would have proved to be an igneous rock, probably basalt, for in all my experience no sedimentary rock, even a hard limestone, would have worn the drill as this rock did. A new and very hard drill was repeatedly lowered and efforts made to drill into it, but with the same result.

This well has, however, established certain facts, to wit: (1) the total depth of the rock salt, a thing heretofore unknown; (2) that oil and gas under heavy pressure exist under the salt; and (3) the existence and beginning of a very hard formation, necessitating better drills than the

best fishtail bit then in use (possibly a Sharpe & Hughes drill might cope with the situation, and if not that, the Calyx drill, or even a diamond drill); (4) that the depth so far attained is not at all beyond limits of economic drilling as practiced to-day, and that with a proper drilling outfit and heavy casing a station could be made to set the casing on the extreme bottom in order to proceed unhampered by gas and caves through a greater depth, say at least 6,000 ft. This would give a latitude of nearly 3,000 ft. of territory still to be explored; and in a zone of such wonderful activity very important facts may be brought to light which may prove conclusively whether or not petroleum comes from igneous rock and consequently is of volcanic origin.

Through the many holes drilled on this dome to ascertain the dip of the salt, there were invariably encountered, overlying the salt, lenses of limestone containing from 10 to 35 per cent. of sulphur. In some instances the drill dropped several feet, thus showing that the limestone is cavernous. This is also evidenced by the fact that in some wells many wheelbarrows of clay were dumped in the hole in the effort to get return of the drill water.

Nearly the whole mass of the salt seems to have been impregnated with gas and oil globules, which are often visible without the aid of a lens. In emptying a bailer of its fragment of salt into a trough, the fragments in liquefying would pop like so many jumping jacks, proving that the salt containing the gas and oil is under heavy pressure. Throughout the drilling of this well gas under heavy pressure, oil, impure gypsum, anhydrite, and sulphur were obtained, and at a depth of 2,745 ft. a ledge of magnetic iron. Where did this magnetic iron come from? Certainly not from the outside of the dome, which is surrounded and hemmed in by muck sand, gumbo, gravel, etc.

The cost of a well like this was of course high, because one did not know what to expect next, but now within the limit of depth attained by this well the cost would be considerably less, and if proper preparations were made the drilling of a well to say a depth of 6,000 ft. would not be excessive, considering the possible results that might be attained.

WILLIAM C. PHALEN, Washington, D. C.—What is the approximate thickness of calcium carbonate and gypsum after you passed through the salt?

A. F. LUCAS.—We have the data; to wit, from 2,900 to 3,171 ft.

WILLIAM C. PHALEN.—It seems to me that these are normal solutions. We should have a calcium carbonate, then gypsum and salt on top. You mentioned the presence of these substances.

A. F. LUCAS.—Yes, within the confines above given, although some may have come above the salt, as the well was not cased at this point.

I. N. KNAPP, Ardmore, Pa.—We established the fact that there was a ledge of magnetic iron. I think there was 4 ft. of magnetic iron at a

depth of 2,745 ft. The bit wore faster than the ore. The magnetic iron consisted of small particles cemented together, and it fairly hissed when it came out of the hole.

GEORGE S. RICE, Pittsburgh, Pa.—Was an analysis made of the gas?

A. F. LUCAS.—No, sir; we were after more valuable or important data than the composition of the gas.

C. W. WASHBURNE, Washington, D. C.—In calling attention to this salt, which Captain Lucas said hopped about when it was thrown on a shovel, that is what they call knister salt. If this salt is put in water it practically explodes, and blows the water around. The gas in it is under very high pressure. It has been measured, and the pressure is under several hundred pounds. These gases could not have gotten in there under steam or water pressure. That salt crystallizes under ground and the gas gets into it at the time it crystallizes. It would crystallize in a shallow dune, and it formed there under pressure of the rocks, and the gas, of course, is under pressure still.

EUGENE COSTE, Toronto, Canada.—I am sincerely grateful to Dr. v. Höfer for this paper discussing the origin of petroleum, and opposing the view of that origin which I have supported on many occasions since 1900. It is only by such discussions that all the facts and arguments in the case will be brought clearly forward; the members of the Institute, mining engineers and geologists, are the best of jurors to decide on both the facts and the arguments; and thus we may hope to arrive at the solution of this scientifically and economically most important problem.

Before answering Dr. v. Höfer's points against the solfataric volcanic origin, I may be permitted to *résumé* what I understand from his paper to be his own views, and what he frankly states as his position on the question. He narrows the origin of petroleums to the direct transformation of animals or fatty plants (such as diatoms) without cellulose; and he considers that the organic matter was originally in the "sands" in which the petroleums are now found, unless in the cases where petroleums have passed afterward through fissures to other sand strata. Dr. v. Höfer also considers these sands to be coastal marine formations, deposited in shallow bays of the sea, where, under special favoring conditions, the oxygen of the air did not destroy as usual the animal or fatty plant matter, which was therefore entombed, and afterward through the agency of long time was gradually distilled at low temperature and under high pressure, and became petroleum.

The clear statement of these views forcibly suggests at once the following objections to them:

I. Why is not this process in active operation in the world to-day? Why can we not abundantly verify it, and witness it in numerous cases in

some of the millions of shallow bays of the sea teeming with life, sands are being deposited to-day, and have been deposited in ages under similar conditions? It is not enough to cite in support of this hypothesis a very few cases, in which empty shells, or matter partly decomposed, were evidently impregnated with petroleum by seepage through fissures or seams from underlying reservoirs.

II. It is also erroneous to say that this hypothesis (p. 871 of Höfer's paper) was accepted by eminent authorities, "mostly in the circumstance that the bituminous rocks carry the fossil remains of animals." As a matter of fact, the fossil remains of animals are found mostly in shales which are, as a general rule, absolutely barren of petroleum. It is only very occasionally, surely not in 1 per cent. of the cubic contents of the strata, that bituminous rocks, or rocks (shales, sands or limestone) containing petroleum, are found; and, as a rule, these spots are comparatively small and are very poor in petroleum. The other 99 or more per cent. of the strata really contains no petroleum beds; and these fossil beds, as is well known, are barren of petroleum. Although some of these shales may be carbonaceous, they are not bituminous or petroliferous.

III. This brings one to the third serious objection to the view of Dr. v. Höfer, namely, that the "petroliferous sands" are so poor in petroleum and the petroliferous sand-reservoirs are so limited in extent and so few in number, with impervious rocks all around them (since we find high pressures in these reservoirs), that the enormous quantities of petroleum which they have produced cannot possibly be accounted for in that way. I will cite only one instance: viz., the example of the small dome of Salt Domes at Beaumont, Texas, where from a little over 200 acres of land 50,000,000 barrels of oil have been produced up to date. The oil is found only under the dome area of a little over 200 acres, the surrounding strata being impervious clays and sands and "gumbo" beds containing fossils but without oil. The secondary crystalline limestones or dolomites under the dome, containing these enormous quantities of oil, are not petroliferous; but even if they were, the oil in them could not be induced in such quantities, and undoubtedly came up the chimney under the dome from below, since it cannot have come from the impervious sides.

This reasoning from indisputable facts, patent in many fields of geology, ago forced the American geologists to the conclusion that the petroleum cannot have been produced in the sands they now occupy. On the other hand, most of the American geologists, and many others, conclude that regional migration of oil out of the impervious surrounding strata into the sands—which, of course, is also impossible. Dr. v. Höfer agrees with me (p. 878 of his paper) that there is no possible regional migration of oil through the pores of such impervious clays and shales as surround

the "sands," and "that the migration of oil can take place only in cracks, joints, and fissures;" but his primary deposits, "the porous sands," are evidently altogether too small in cubic capacity, and too poor in organic contents, to furnish the enormous quantities of petroleum which have actually been produced from them.

Moreover, in the different fields of the world we can trace these primary sand deposits of Dr. v. Höfer lower and lower down in the geological scale, until we find them not only in the Devonian and Silurian but also in the Cambrian (Potsdam sandstone in N. Y. State) and in the crystalline rocks (Newhall, Cal.). This forces us to admit a still lower source, namely, the volcanic magma; and when these volcanics everywhere give so much evidence of containing large quantities of hydrocarbons either in their associated solfataric gases or in the lavas themselves, why should we reject that source to which we are forcibly led by the full consideration of the geological evidence mentioned above?

IV. If the petroleum deposits were coastal marine formations, deposited in shallow bays of the sea, they would be found under geographical alignments entirely different from the straight oil belts *in which they are actually being found*. The oil belts are evidently connected with the tectonic and orogenic disturbances of each region, and not with the ancient shore lines of the different formations. Moreover, along the same belt we find the petroleum impregnating sands of many different ages. In California, for instance, from and including the crystalline rocks to the Quaternary, there is a thickness of some 30,000 ft. in which productive sands are found. Yet, outside of the productive narrow belts along the Coast Range these 30,000 ft. of strata are barren of petroleum. Surely it cannot be imagined that marine bays of the ancient seas could align themselves in that way along fault lines or straight disturbed zones, and juxtapose themselves, one on top of the other, in formations of so many different ages, according to just the same tectonic zones of disturbance.

V. In shallow bays of the sea, in which sands are deposited, the organic matter is generally observed to have totally disappeared, having been destroyed by the oxygen of the air. Dr. v. Höfer admits this (p. 878); but he speaks of vague special favoring conditions which *occasionally* permitted the preservation and entombment of the organic matter. Would such special favoring conditions explain the enormous quantities of petroleum in the world? And why should these special favoring conditions occur at repeated periods during long ages in the same district along fault lines or disturbed zones; and what are these special conditions, anyhow? If petroleum were deposited in shallow bays, what about the deep vertical chimneys of Texas and Louisiana with several thousand feet of thickness of salt impregnated with petroleum?

VI. Admitting, for the sake of argument, that the soft organic

tissues of animals, or the fatty tissues of plants, were occasionally tombed, how did the transformation of these into petroleums take place? Dr. v. Höfer says it was by the action of long time, which permitted a slow distillation at low temperature; and, strange to say, as a scientific proof of that proposition he gives the experiments of Engler, in which oils similar to petroleums were produced from organic fats by heating in a retort at temperatures from 300° to 400° C.—experiments made under conditions of temperatures entirely different from those which obtain in nature, and therefore not in the least to the point. If long time could distill some of the organic matter of the sediments into petroleum, is it that it did not produce any other effect on these sediments, such as the “coals” contained in them, which are unaltered and undisturbed? And if long time could replace temperature in bringing about distillation, should not everything on this earth be in a gaseous state, as it has been all the time imaginable in the eternity behind us to bring about the same effect as the highest imaginable temperature? Phenomena of physical or chemical changes of state in elements require certain temperature points and will not take place at a lower temperature, no matter the length of time. One might as well say that by leaving a piece of meat long enough in cold storage it would cook itself!

I will now take up in their order Dr. v. Höfer's criticisms of my paper as contained in his paper.

1. I, of course, never intended to state anywhere in my paper that there was no methane in petroleums; what I did say was, that the gas formed from the decomposition of plants is quite apart and distinct from the methane of petroleums.

2. Whether the crystalline schists or gneiss, from which a light gravity oil is produced near Newhall, Los Angeles county, is a metamorphosed sedimentary or not, and is of Jurassic age or Archean age, makes absolutely no difference in the point which I am making about this occurrence of petroleum, namely, that the petroleum is indigenous to the crystalline rocks and therefore cannot possibly be indigenous to the sedimentaries. If the crystalline rocks must come from the San Gabriel granite or the magma below. If the crystallines are ancient sediments, they must certainly have lost their organic matter during the metamorphosis, and especially the light gravity oil as is found there must have an extraneous origin. To attribute that origin to the neighboring Tertiary oil field is altogether impossible; since light oils of that nature, full of gas, never go down but always ascend.

3. To suppose that the oil and bitumen in the quicksilver deposits of California and elsewhere may have been extracted and transported from deeper bituminous rocks by the ascending ore-bearing solutions is to reverse the problem without the shadow of a proof. One might as well suppose that the quicksilver itself in these veins had its origin

wall rocks, instead of the ascending ore-bearing solutions. It is well known to mining geologists that ore-bearing solutions, circulating in veins and fissures, sometimes impregnate the wall rocks and become diffuse in them; but they cannot do the reverse, and receive their contents from these wall rocks.

4. I must differ entirely with Dr. v. Höfer when he says that the occurrences of bitumen, petroleums, or graphite in metalliferous veins, pegmatite dikes or volcanic rocks, are scanty or sporadic occurrences. I maintain, on the contrary, that they are frequent all over the world and constitute positive and overwhelming proofs that these products, in all such cases, have an inorganic origin. To suppose that volcanic or eruptive rocks can distill and absorb into their own mass petroleums or natural gas from bituminous materials in the wall rocks, is again to reverse the question without the semblance of a proof, and moreover, involves an impossibility. One cannot look for distillates inside of the hot mass which produces the distillation. The very word "distill" means "driving away."

5. The occurrence of oil around volcanic necks in Mexico is questioned by Dr. v. Höfer, who says that it is widely distributed. From all the records of reputable geologists who have examined the occurrences of oil in that country, and even from the records of Mr. Villarello, quoted by Dr. Höfer (p. 875), it is quite clear that the petroleum deposits are always intimately connected with the volcanic necks. In fact, this is one of the clearest evidences in the world of the solfataric volcanic origin of oil in enormous quantities.

6. Dr. v. Höfer says that, as a logical consequence of my views, deposits of petroleums should always be found in the vicinity of volcanic eruptions. As we have just seen, when it is found in such vicinity in enormous quantities, as in Mexico, the proximity and connection of the volcanics and the petroleums are denied. But when the volcanics are not plainly to be seen, then their occurrence in the petroleum fields is demanded. Faulting and fissuring connected with volcanic manifestations take place all over the world, not only in mountainous regions but also in regions of plains, and may be, and are, often accompanied with solfataric emanations, though the lava or volcanic rocks themselves do not appear at the surface. It is clear upon careful consideration of these phenomena that any belt of country very rich in eruptives, such as the "inner bend" of the Carpathians cited by Dr. v. Höfer, might be too much faulted and fissured to permit the storage of the gaseous emanations in the greatly disturbed and broken surrounding sediments, while another belt of the same country such as the "outer bend," which is sufficiently fissured to permit pent-up vapors and gases to force their way through to the porous portions of the sediments, and yet not so much as to permit their complete escape to the surface, would naturally

furnish the best and richest petroleum fields. Even in such oil fields as those of Pennsylvania and northwestern Ohio, where the strata are apparently undisturbed, we find such well-marked breaks as the Lodi Volcano break and the grahamite vein of solid petroleum near it in West Virginia, and the famous Findlay break in northwestern Ohio, so well described in several of Orton's reports as the most pronounced disturbance in that State. In the greatest number of oil fields the elongation of the different pools or fields, all in one direction, demonstrates that they are connected with fissuring and faulting.

I believe it is unnecessary to prolong this discussion and to trouble Dr. v. Höfer's remarks on the objections advanced by me in my *Petroleum and Coals*, to the theory of organic origin, as I consider these objections still stand and have not been sufficiently answered. Most of these points are also covered by my remarks in this discussion or in my new paper, read at the same meeting of the Institute with Dr. v. Höfer's, and written before I had seen the latter.

PROF. DR. H. VON HOFER (communication to the Secretary*) translated by R. W. Raymond).—The remarks of Captain Lucas, the Chairman of the Committee, are highly interesting for the geologist and miner of saline deposits; yet none of the facts he mentions can be regarded as evidences of the volcanic origin of petroleum. The emanation of gas from salt deposits has long been known, as Mr. Washburne, in the discussion, has already observed. At various places, in Galicia (Kamienica); in the Alps (Hallstadt); and in the German potash district (Desdemona mine near Ahlfeld), natural gases in salt deposits have been the cause of gas explosions. In the salt mine of Szlatina, Hungary, natural gas was encountered in 1783, and used for several years to light the workings. The natural gas occurring in connection with the salt deposits in the Chinese province of Sz'tschwan served for centuries as a source of heat.

Petroleum also is found in salt, though mostly only as an impurity in solution, or in pockets, in the Carpathians and the salt deposits of northern Germany. In the latter, it has occurred sometimes in considerable quantity. For instance, oil to the value of 35,000 marks was obtained last year from an intrusion of petroleum into the Hope salt mine near Schwarznsteck. Yet, at the present time rock salt and potash are won without any admixture of oil, are won at this mine.¹

At all the European localities above named, there is no trace of volcanic activity, and no plausible ground for inferring it.

During the Third International Petroleum Congress, held in Bucharest at Burcharest, I first proposed, as generally applicable to deposits

* Received Apr. 18, 1914.

¹ *Petroleum*, Berlin, vol. ix, p. 981 (1914).

troleum and natural gas, the "dome theory," which has since received manifold confirmation, and has proved highly useful in practice. I was able at that time to support my new theory with facts observed in three continents, and showing without exception the favorable influence of the dome-shaped structural form. Most of these cases could be referred to one type—that of a locally elevated anticline. The salt domes of Louisiana and Texas constitute a class by themselves. The cause of their genesis has remained unknown hitherto, and will be difficult to discover, so long as the salt deposits have not been explored outside as well as inside of the domes. Captain Lucas assumes a laccolite, which has elevated the dome; but of this he has no evidence to offer. If a volcanic laccolite, as such, were the source of the oil formation, we would naturally expect the occurrence of petroleum likewise in connection with other laccolites, such as those of Utah, Colorado, New Mexico, Dakota, etc. But nothing of this kind is known, so far as I am aware. In fact, I think that nothing at all is yet known, as to origin of these salt domes. I can only refer to the "Ekzem-hypothesis," which is becoming more and more popular in Europe, as stating the simplest way in which, under the influence, perhaps, of other abnormal genetic conditions, these domes may have been formed.

The Knapp boring proved conclusively in the dome the regular saline series; first (at the bottom) lime; then gypsum; and then rock salt—as it occurs in all such *stratified* saline deposits. Hence it is impossible to conceive a volcanic process for the formation of the salt deposit.

The solid and tough rock reported at the bottom of the abandoned bore hole was never determined, and can therefore furnish no basis for a hypothesis. Perhaps it may have been, not even a rock, but a piece of iron which had fallen into the hole.

I am sorry that I must answer my friend, Captain Lucas, by saying that I cannot change my mind according to his suggestion.

From Mr. Coste I expected a more detailed reply, dealing, point by point, with my criticisms. But he has but a few things for remark.

Regarding his comments, I would observe that in a genetic question it is not a matter of indifference whether a given gneiss is, in Rosenbusch's sense, on orthogneiss or a paragneiss; that is, whether it was formed from granite, or from a post-Archean sedimentary rock. Mr. Coste, for the benefit of his volcanic hypothesis, makes the California gneiss granitic, whereas it is evidently sedimentary in origin, and can therefore furnish no evidence of the volcanic origin of petroleum.

The origin of the petroleum found in the quicksilver deposits of California, I leave, as I have already said, to be discussed by my American colleagues. I could say it came from some oil deposit not far below the place which it now occupies, with as much reason as Mr. Coste has for

saying it came from great "volcanic" depth. One ignorant would simply contradict the other.

It is notorious that in Mexico extensive oil fields are now known which are far from any volcanic eruptives.

The various attacks of Mr. Coste upon the theory of the origin of petroleum and natural gas, I leave unanswered, until he has kindly offered a thorough and coherent criticism of my arguments such as I made in the interest of science, concerning his.

The Geology and Ore Deposits of the Bully Hill Mining District, California

Discussion of the paper of A. C. BOYLE, JR., presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 85, January, 1914, pp. 57 to 60.

E. GYBBON SPILSBURY, New York, N. Y.—There is one point in Boyle's paper on the Bully Hill Mining District which is very interesting to me as I happen to have seen the occurrence myself, and was therefore inclined to call attention to its probable significance there; I refer to the development in the lower levels of the mine of the anhydrite and gypsum deposits, which have been increasing gradually from the 700-ft. level down to the present bottom, where the deposit has obtained great thickness. The purity of the deposit is remarkable, and it occurs in the crushed zone as the pyrite and pyritic lenses, and seems to have a great deal of influence on the continuance of the ore below the level where it first occurred.

Professor Winchell in his examination appeared to think that the occurrence marks the bottom of the mineralization, but Mr. Boyle does not agree with this conclusion. I think probably he is right, because there is no sign of mineralization in the anhydrite and gypsum beds themselves, still there is a persistent continuance of small seams along the contact of the gypsum with the andesite of the country rock, and it is quite probable, therefore, that the gypsum will be found to be only a local intrusion of limited depth, and that the mineralization along the contact will be found below that point. I would like to ask whether anyone else knows of any of these copper deposits which have bottomed out on gypsum or anhydrite? It is the only occurrence of the kind that I have ever observed.

G. A. PACKARD, Boston, Mass.—The deposit of the Ludwig mine in the Nevada Douglas Co. in the Yerington mining district is somewhat similar, as I remember it. It has been described by Mr. Ransom, and since then by A. F. Rogers.² There is a difference of opinion be-

these two men as to the method of formation of this gypsum, whether or not it is the result of the action on the limestone of sulphuric acid from the oxidation of the sulphide ores. The deposit is partly gypsum and partly anhydrite, and there is also a question as to which mineral is the original form and which the result of alteration.

L. C. GRATON, Cambridge, Mass.—A few years ago, I spent some time in the Shasta county copper region and found the occurrence of masses of gypsum in the Bully Hill and Rising Star mines of particular interest, because so unusual. In the body of some of these larger fine-grained gypsum masses there appeared some more coarsely crystalline material, which, when examined in microscopic sections, proved to be anhydrite.

As yet, I have not had opportunity to read Mr. Boyle's paper carefully. It appears, however, that he is not satisfied as to whether the gypsum was derived from the anhydrite or the anhydrite from the gypsum. Numerous occurrences have of course been described in which gypsum has been converted into anhydrite by dehydration. The structural relations of the material that I collected, however, seemed to leave no doubt that anhydrite is the original mineral and that the gypsum is altering from it.³

This conclusion gathers strength from the fact that at the Cactus mine, at Frisco, Utah,⁴ that rather unusual type of ore contains veinlets and masses of gypsum in it, and in the deeper levels the gypsum gives way to anhydrite, from which the gypsum is being derived. Large masses of pinkish anhydrite are to be found associated with plentiful barite, a mineral that is also present in the Shasta county occurrence. Since seeing these two occurrences, I am inclined to believe that the gypsum associated with gold in the White Oaks district,⁵ New Mexico, may likewise have been derived from original anhydrite gangue stuff. Professor Lindgren has told me that on a recent trip to Cuba he found at El Cobre mines anhydrite intergrown with sulphides, and altering to gypsum. I understand, also, that Professor Palache has found anhydrite intergrown with prehnite and datolite in the traps of Meriden, Conn., an association that points to an origin connected with vulcanism, if not with ore deposition.

The situation at the Ludwig mine, Yerington, Nev., seems to be less clear. The occurrence of gypsum in association with limestone on the foot-wall side of a thick bed of deeply oxidized sulphides suggested⁶ that the sulphate of lime might have been produced, as in small amounts

¹ *Bulletin No. 380, U. S. Geological Survey*, p. 113 (1909).

² *Economic Geology*, vol. vii, No., 2, p. 185 (Mar., 1912).

³ *Bulletin No. 430, U. S. Geological Survey*, p. 100 (1910).

⁴ Lindgren, W.: *Economic Geology*, vol. v, No. 6, p. 522 (Sept., 1910.)

⁵ *Professional Paper No. 68, U. S. Geological Survey*, p. 181 (1910).

⁶ Ransome, F. L.: *Bulletin No. 380, U. S. Geological Survey* (1909).

in other regions, by reaction of sulphuric acid waters from the ore upon the limestone. Professor Rogers,⁷ however, who later mined the mine, found that at depth the gypsum gave way to anhydrite. He concludes that the association of the ore and the lime sulphate is wholly accidental. In any event, it is interesting to note that he found anhydrite to be the original mineral and that gypsum is being derived from it.

With regard to the significance of gypsum and anhydrite in the Bully Hill district, I have understood that the large masses of these minerals encountered in the deeper levels have been regarded by some geologists as unfavorable occurrences, indicating the likelihood that ore of commercial grade will not be found in quantity at depths below these levels. One may be perhaps a little suspicious of these minerals compared with their unusual in ore occurrences. But if one grants that anhydrite is the primary mineral, remembers that this calcium sulphate and the barite sulphate, barite, are mineralogic twins, so to speak, and recalls that not only is barite common in many ore deposits, whether rich or poor, but also occurs in intimate association with much of the best ore of the Bully Hill mine, then it is difficult to find reason why the mere occurrence of calcium sulphate in this district should be regarded as either particularly favorable or particularly unfavorable for the persistence of ore.

⁷ *Economic Geology*, vol. vii, No. 2, pp. 185-189 (Mar., 1912).

To What Extent is Chalcocite a Primary, and to what Extent a Secondary, Mineral in Ore Deposits

A discussion at the New York meeting, February, 1914.

L. C. GRATON, Cambridge, Mass.—The subject of chalcocite occurrence and its geological significance has, of course, a very important commercial bearing, as shown by the recent remark of a hard-headed mining man: "This secondary enrichment business is one thing you geologists have put up which we can hang on to, and come out right."

Most enrichments of copper ores depend upon the formation of chalcocite from leaner sulphides. Recently it has been shown that not only does there exist this secondary or derived chalcocite, but that in certain ores the mineral is present as an initial or original constituent, which therefore we may call primary. The desirability of distinguishing between these two kinds of chalcocite is evident and has recently been emphasized. That constitutes the first problem in this connection. I believe many who are working upon secondary enrichment of mineral deposits are giving much attention to that particular question.

It is now possible to distinguish primary from secondary chalcocite in many cases. In such cases, what do we know of the conditions under which this secondary or derived chalcocite was formed? This is the second main question in connection with the subject. It is now plain, at least in many instances, that the formation of this derived chalcocite represents a phase of oxidation if we have in mind the primary ore as the thing acted upon. The next important question—and it is most important—is: does such oxidation go on above the zone where the rocks are saturated with water—that is, above the water level—or below, or indiscriminately above and below?

Many of you have, or have had, opportunities to study at first hand, day after day, occurrences that enable valuable conclusions to be drawn upon these and other important questions concerning the formation and significance of chalcocite. It is greatly to be hoped that your testimony will be made available and undoubtedly it will go far toward clearing up some of these doubtful points.

JOHN D. IRVING, New Haven, Conn.—A short time before this meeting was called, H. V. Winchell, who was unable to be present, requested me to exhibit and describe these two samples of pyrite. (Exhibiting specimens.) They consist of fragments of pyrite inclosed in two small bottles. The first bottle shows fragments of pyrite which were immersed in water carrying copper sulphate but without SO_2 for a period of two years. They remained comparatively clear and bright. No chalcocite has been deposited on them. The second bottle shows fragments of

pyrite which were immersed in a solution in every way similar except it was saturated with SO_2 . Immersion was for a period of one year. Removing the pyrite it was found to be coated with gray metallic chalcocite. The coating covers all parts of the pyrite grains, so that unless the grains are broken or scratched they appear to be solid chalcocite.

Members of the Institute may recall that in 1903 Mr. Winchell contributed an article¹ in which he described certain experiments in the precipitation of chalcocite by pyrite from mine waters. The specimens which have been exhibited are those of the pyrite which was used in the original experiments described by Mr. Winchell. The portion of the paper which deals with these experiments is found on pages 272 to 276 of the article above mentioned. In order that the meaning of the specimens may be clear to the members the following paragraphs are quoted from Mr. Winchell's paper:

"Artificial Formation of Chalcocite"

"After considering the geological history and physical structure of the deposits, the writer came to the conclusion some three years ago that the chalcocite was formed by a chemical reaction between copper sulphate in solution and descending waters and the iron pyrites and other primary sulphide minerals below. In order to ascertain the truth or falsity of this theory, laboratory experiments were undertaken by the writer and carried on by Messrs. S. J. Gormly and Tolman in the laboratories of the Anaconda Copper Mining Company.

"The first experiments were conducted with a relatively small amount of ferrous pyrite and a dilute solution of copper sulphate. The results, as reported, showed the formation first of SO_2 and then of H_2SO_4 ; the solution of both copper and iron, the precipitation of the iron as ferric hydrate, and the formation of copper sulphide.

"Analyses of the mine waters showed no ferrous salt in the strong copper solution, but disclosed the presence of quantities of cuprous salts, in acid solution.

"The experiments repeatedly showed that SO_2 is formed by the action of iron and chalcopyrite upon CuSO_4 , and that the SO_2 reduces some of the copper ions of CuSO_4 to the cuprous form. According to theoretical chemistry, a relatively insoluble compound may be precipitated by very small amounts of a salt containing one of the ions of the insoluble compound, if a large amount of the salt containing the other ion is present. To test this, a solution of copper sulphate was treated with the sulphides of arsenic, lead, copper, iron, zinc, and with pyrite; and in each case copper sulphide was precipitated, proving that these sulphides may precipitate copper sulphide from a solution of a copper salt. It is probable also that the more insoluble the precipitating sulphide, the more concentrated must be the solution of copper sulphate.

"To produce a solution containing cuprous ions, the above mentioned sulphides were treated with a solution of copper sulphate (CuSO_4) and SO_2 , and precipitates were formed in each instance. An analysis of the precipitate formed by iron sulphide showed a precipitation of 12 per cent. of the weight of the original Cu_2S , indicating the formation of chalcocite under these conditions.

¹ Synthesis of Chalcocite and Its Genesis at Butte, Montana. *Bulletin of the Geological Society of America*, vol. xiv, pp. 269 to 276 (1902); also *Engineering and Mining Journal*, vol. lxxxiv, No. 23, pp. 1067 to 1070 (Dec. 7, 1907).

"It was not ascertained whether the iron salts will reduce enough copper to form Cu_2S in presence of pyrite or other sulphides, or whether the SO_2 formed by solution of the pyrite and other sulphides is the active agent.

"Knowing full well that it might be urged that the formation of a precipitate of a certain chemical composition is quite a different matter from the production of a mineral having the same composition, the experiment now about to be briefly described was undertaken and carried to completion with exceedingly gratifying and satisfactory results.

"In a slightly acid solution containing sulphurous anhydride (SO_2) was digested pyrite (FeS_2) at ordinary temperature and pressure for three months. The pyrite taken was ordinary 'jig concentrates,' about one-fourth of an inch in diameter, from the Parrot concentrator at Butte, and of the following composition:

SO_2	8.30
CaO	Trace
MnO	None
Fe	41.20
Cu	1.50
Zn	0.20
S	48.70
	99.90

"Dividing these results by the molecular weights, the molecular constitution is represented as follows:

Fe	0.736
Cu	0.024
Zn	0.003
S	1.522

"After standing for three months in an ordinarily well lighted room, inclosed in a sealed jar to exclude the atmosphere, the formerly yellow grains of pyrite were completely plated with a solid coating of a dark blue-black mineral, and so closely resemble grains of solid chalcocite that they can only be distinguished from the latter by breaking them open, while in another jar which stood alongside, similarly sealed and exposed to light and ordinary temperature, containing pyrite and copper sulphate solution (but no SO_2), the grains of pyrite were just as bright and yellow as before they were immersed. Indeed, there has been no visible alteration on the surface of grains which have now been thus immersed in copper sulphate without SO_2 for two years, while in an adjacent jar containing SO_2 there has been formed what appears to be, and undoubtedly is, a fine coating of chalcocite.

"From the first jar there were taken some of the larger grains for analyses, with results as follows:

SiO_2	9.60
CaO	
MnO	
Fe	40.10
Cu	3.60
Zn	
S	43.60
	99.60

CaO , MnO and Zn not determined

"The molecular constitution is now Fe, 0.716; Cu, 0.057, and S, 1.448. are .016 equivalents of sulphur left over (after calculating the iron as FeS_2) with the copper. The exact theoretical amount to form Cu_2S is .014, and the sulphur may be combined with zinc, lime, or manganese. That the mineral thus formed on the jig-concentrates is clearly chalcocite can not be doubted mere inspection of the samples and comparison with ore from the mines."

So much has been said in discussions on secondary chalcocite concerning the prevalence of a black powder, or what is commonly known as "sooty chalcocite," that Mr. Winchell desires to call the attention of members to the fact that the chalcocite secured in his experiments was not of the sooty variety, but rather of the usual gray variety with metallic luster. It forms, as may be seen in the sample which stood for some time in water carrying an excess of SO_2 , a thin, glistening, grayish coating on the outside of the pyrite and bears no resemblance to the usual "black" variety.

While the "sooty" type is therefore to be considered as secondary, the gray massive variety may likewise originate in the same manner.

THOMAS T. READ, New York, N. Y.—In 1905 I did some experimental work along much the same lines as that which Mr. Winchell has done, because it was so obviously of great significance, and found that the presence of SO_2 was apparently exceedingly effective in bringing about the desired condition. The action was not at all marked in the absence of SO_2 , where the SO_2 was present it was very marked.

A difficulty in drawing conclusions from this fact, however, is pointed out by W. H. Emmons reports that he has been unable to detect the presence of SO_2 in mine waters. Geologists commonly regard SO_2 as a reducing agent, possibly because SO_2 is the agent in which we used to reduce gold from cyanide in the laboratory, in order to precipitate gold. But SO_2 is obviously, exceedingly rich in oxygen, and toward FeS_2 might act as a strong oxidizing agent. That induced me, at that time, in a paper presented to the Institute, to suggest that the action of copper sulphide on the sulphides is ascribable to ordinary oxidation and that secondary sulphides might be considered to be oxidation products, just as, for example, in the copper converter, from a sulphide which is quite lean in copper you can, by oxidation, get one which is high in copper. Similarly, rich sulphides of copper may be and in some cases doubtless are, produced simply to the effects of oxidation on lean sulphides.

L. C. GRATON.—In connection with the synthetic production of chalcocite, it may be pointed out that results like those in which Mr. Winchell and Reed produced coatings on the primary sulphides were indeed positive, but that negative results, such as their failure to produce such reactions without the presence of SO_2 , cannot be relied upon for entire safety as indicating that in nature some reducing agent like SO_2 is required for the operation of the secondary enrichment process.

function of time in a process of this sort as carried on in nature must be given due consideration and it may very well be that a chemical reaction which exhibits no measurable progress during a period of a few weeks or months or years might advance to a very important degree in the course of a short time as measured in geologic units.

It is of interest to note that the chemists of the Geophysical Laboratory of the Carnegie Institution in Washington, who have been at work for something over a year on the general problem of the copper sulphides, have sought to steam up or intensify the processes of nature by agitation and by increased temperatures but without increasing the concentration of the solutions over those of the natural reagents. Under these circumstances it is possible to produce measurable results in probably much shorter time than nature would have required. The particular point of interest in this connection is that by treating pure pyrite with a dilute acidified solution of cupric sulphate, they have been able to secure coatings of a substance which both chemical and microscopic analysis shows to be actually chalcocite—and this without the intervention of any reducing agent, or at least any other reducing agent than the original sulphur-rich sulphide, itself—viz., pyrite.

Professor Kemp, I believe, was the first to suggest that the excess sulphur in the mineral attacked, such as pyrite, pyrrhotite, etc., might be responsible for the evident reduction of the copper sulphate to sulphide. If one grants that the copper to produce enrichment has come from overlying portions in the form of sulphate, its ultimate precipitation as sulphide can be brought about in no other way then by chemical reduction, if one has in mind the copper so precipitated; if, on the other hand, one thinks of the primary sulphides acted upon, the process is one of oxidation.

The axiomatic fact that no oxidation can take place without equal reduction appears to have been temporarily overlooked by some in the discussion of this subject. The secondary deposition of chalcocite, therefore, may be called the result of either oxidation or reduction, depending on the viewpoint. But since the ultimate reagents involved are the atmosphere and the original primary sulphides, and since we are accustomed to think of the changes experienced by the latter rather than by the former, it follows that the development of chalcocite as an intermediate step in the reaction may most naturally be regarded as due to oxidation.

Another topic has been mentioned upon which I should like to say a word, and that is the matter of sooty chalcocite. My experience to date leads to the belief that sooty chalcocite may as well be produced from primary chalcocite as from secondary chalcocite, provided it comes within the reach of the oxidizing, disintegrating influences near the surface. The original precipitation or beginnings of precipitation of secondary

chalcocite upon pyrite, as may be best studied in the case of a pyrite crystal undergoing conversion to chalcocite at its surface, producing a coating that always, so far as I have observed, is of dense, massive chalcocite—the steely chalcocite or glance of the miner; and so long as chalcocite continues to form, it is of this massive, steely variety. However, when the conditions favoring the deposition of chalcocite cease, and oxidation or solution of the chalcocite begins to ensue, then around the edges of the individual chalcocite grains decomposition or solution begins, the individual grains are reduced in size, and the whole mass becomes lusterless and incoherent, just as a cake of ice becomes dull and crumbly when left in the sun. This dull, friable, decomposing, and disintegrating material is, in reality, the so-called “sooty” chalcocite.

In the paper on the New London mine, Maryland, now offered for publication in the Institute records by B. S. Butler and H. D. McGowan of the U. S. Geological Survey, it is interesting to note that they find primary chalcocite and bornite associated in a manner similar in all essential respects to that now known at various places in the Piedmont region. This reference perhaps affords an opportune occasion to review certain conclusions regarding primary chalcocite stated by Mr. McGowan and myself² at the meeting a year ago. In one instance was determined what we regarded as primary chalcocite in the ores of Bisbee. There was but little of the mineral present in the material we studied, but at the time the evidence seemed satisfactory. Since then, in that identical material, we have found in the chalcocite exceedingly minute residues of bornite distributed in such fashion and showing such character as to leave practically no doubt that the chalcocite had been derived from bornite, that it was, therefore, secondary and not primary, and presumably had been produced in the ordinary course of enrichment. We also state our belief that the chalcocite of the Bonanza mine, in Alaska, is primary, and is, the result of initial deposition. Very recently, in going over the material upon which we reached that conclusion, we find there are also exceedingly minute and possibly doubtfully identified bornite, but sufficient to raise the same doubt as to the character of the chalcocite, and this doubt is somewhat strengthened by the finding, in material taken from about the deepest workings of that mine, bornite in considerable quantity, altering to chalcocite and chalcocite-pyrite, with smaller amounts of some other minerals. So our minds are very much unsettled as to the nature of the Bonanza chalcocite, and we are a little inclined to feel that our earlier conclusion regarding it may be erroneous.

F. L. RANSOME, Washington, D. C.—I wish to express my agreement with Professor Gratton that much of the so-called sooty chalcocite is a disintegration product of the ordinary chalcocite with bright metallic luster; but some caution should be exercised in the interpretation

² Trans., xlv, 26 (1913).

term "sooty chalcocite" as found in literature. In a mass of fine-grained friable pyrite that has undergone partial change to chalcocite, each grain is coated with chalcocite and each little coating, examined microscopically, may be lustrous dense chalcocite. Yet the general appearance of the whole is that of a dull black, friable material. Much of the sooty chalcocite referred to in literature is material of this sort.

WALDEMAR LINDGREN, Boston, Mass.—I want to say that we are accumulating new observations so fast we are in danger of forgetting some of the old ones. The other day I came across something I suppose every mineralogist knows perfectly well, and that I think most mining geologists have forgotten—although they knew it at one time—and that is that beautiful crystalline chalcocite occurred at the Bristol mine, in Connecticut, formerly the greatest producer in the United States—that is also news to you. Specimens of these ores which are in the Massachusetts Institute of Technology, and I suppose they are in other places, show perfectly clearly, as far as my examination has gone, primary chalcocite in beautiful crystals, and primary beyond any reasonable question.

In the hunt for primary chalcocites that occurrence seems to be overlooked, and I thought I would call attention to it, in case it should happen that anybody knows anything about the occurrence, who has studied it. I think it is, except the crystals from Cornwall, of which I do not know much personally, the most beautiful and sharply crystallized chalcocite that has ever been found.

JAMES F. KEMP, New York, N. Y.—At the meeting of the Geological Society this winter, in Princeton, Professor Van Hise, of the Chase Scientific School, exhibited some crystals of bornite that only two or three of us had ever seen, and they also came from the Priestly mine, in Connecticut. I do not know whether we are justified in saying that there is an association of bornite and chalcocite, just at the moment, but this association seemed to hold good. Bornite is a rare thing to find in crystals, although common enough in mass form. Chalcocite, likewise, we seldom get in crystals, but the forms that were obtained from Priestly, Conn., were mentioned in the earlier mineralogy of this country, as to the forms of chalcocite. It would be interesting if Professor Ransome can develop some new matter from these old forgotten sources of copper.

The Oil Shales of Elko, Nev.

(A Discussion at the New York meeting, February, 1914)

R. M. CATLIN, Franklin Furnace, N. J.—I hardly think the "oil shale" is strictly applicable to this deposit. It is a small far as I know, extending, perhaps, for half a mile northerly and southerly, and there are five outcrops. I think instead of being an oil shale comparable with the Scotch shale, it should be more properly called a wax shale. There is a core or a sheet of rhyolite, which shows no oil, and on either side are bands of shale, but differing from the Scotch shale in that it is largely saturated with paraffin wax, and occurs as a mixture, and not in the form known in Scotland as "kerogenite." This leads one to the supposition that in geological ages past there was an oil spring there, and when the shale became saturated and the spring became clogged with wax, the oil found other exits, so that no oil is now found there; though at various times bore holes have been drilled down, neither oil nor gas is found. There are numerous thermal springs through that region, probably coming from great depths, crossing the strata, and any fluid oil in the strata would probably have long since escaped. On top of the rhyolite seam (which is about 12½ ft. thick) lies some 24 ft. of oil shale seams, separated by seams of what the Scotch call "blaes." This shale is of low grade, yielding not more than 12 per cent. by weight of oil. Below the rhyolite, the shale is more massive, 8 or 10 ft. thick, and yields 25 to 30 per cent. by weight of which 30 to 40 per cent. is wax. It apparently contains no tars and is very easily refined.

F. M. ANDERSON, New York, N. Y.—These shales have been described as dense and compact shales. A few years ago the Department of Chemistry in the University of California got very much interested in these shales, and they got samples and experimented with them. Dr. O'Neill, who was chiefly interested in the experiment, took me to the laboratory to show me what he was doing. He was putting the shale in a retort, and retorting off oil, and he made the statement to me that he had tried various solvents of bitumen on these shales—chloroform, carbon tetrachloride, and other solvents—and he was unable to obtain or detect any bitumen, and yet by putting them into the retort he could get 13 per cent. by weight of light oil. He asked me what I thought of it, and what I supposed might be the origin of the oil. I had a little lens in my pocket and examined the shales and found them to be a mass of organic matter, known to live in fresh water. I showed them to him, and he was much interested. I have seen the same thing in southeastern and southwestern Wyoming, and in the Green River shales.

The volcanic matter associated with the shales in both instances

volcanic ash. It seems to have buried organic remains, such as fish and many fresh water ostracops and crustacea; and the organic content of these shales forms a large percentage. I do not know exactly the amount, but the examination I made of the Elko shale left the impression that probably 75 per cent. of the weight of the shale was of organic origin.

EUGENE COSTE, Toronto, Canada.—The shales of Elko, Nev., may be composed not only of 75 per cent. but of 95 per cent. of the hard remains or shells of organisms, and yet for all that there might never have been entombed in these shales any of the soft carbon, or really organic tissues, of these organisms, whether vegetable or animal.

Organic chemistry, so called, is the chemistry of carbon compounds, and on that account I think the words organic and carbon are constantly mixed up one for the other and used interchangeably, as well as the words bituminous and organic, but one cannot settle the problem of the origin of bitumens or oils or petroleums by just playing on words and simply admitting that organic is synonymous with carbon or bitumen.

Now, Mr. Anderson says that he examined with his lens specimens of these shales, and found them to be composed mostly of hard skeletons of shells. Well, that is hardly organic. It is either silica or limestone; it is only the house, as it were, of the organism, not the organism itself. I have here a little quartz crystal, inside of which one can examine with a lens a globule of oil. Dr. Ledoux gave me this one day, and it is much more a proof of the inorganic origin of oil than shales formed of the detritus of the hard parts of organisms and containing some oil are a proof of organic origin, as the oil might easily have filtrated from inorganic sources into the shales after their deposition, but could not be found inside of the quartz crystal if from organic source. But to solve the problem of the origin of petroleums one really must look not at the oil in a little piece of shale, or in a quartz crystal, but at the petroleum deposits all over the world, in Pennsylvania, Texas, Colorado, California, Russia, Roumania, Java, Borneo, etc., and he must be able to explain the different phenomena connected with all these deposits, before he can draw conclusions as to the origin.

Now, going back to those shales: Mr. Catlin says that they are somewhat different from the Scotch shales in the way they contain the petroleums. I do not agree with him in that. The Scotch shales, containing the kerogen or petroleum, are located in Midlothian, near Edinburgh. They are intruded there with volcanic rocks, as are the Elko shales in Nevada. In Midlothian where the oil shales are mined there are some 35 masses of volcanic rocks intruding through the shales, as shown on the geological map of Henry M. Cadell accompanying his paper on the district (*Trans. of the I. of M. E.*, vol. xxii), and not only the shales near the volcanics are impregnated with petroleum in that district, but right in the volcanic rocks at a number of places a wax is

also found with the composition of ozokerite, viz., 14 per cent. hydrogen and 84 per cent. carbon; very much the same as the composition of the wax from Galicia, which is mined there largely and found in vertical veins. Some of the defenders of the organic theory attribute the wax, or other petroleum found in igneous or volcanic rocks, to the fact that the hot volcanics distilled the pre-existing bituminous matter in the shales or other surrounding rocks into vugs or cavities of the volcanic rock. Now, distillation means driving away of gases and vapors, and therefore how can the hot volcanic rocks distill these vapors away into themselves? That is another of the upside-down views which are being taken by some on this question. It is like putting some oil on the lid of a hot stove, and then looking for the distillates inside of that lid. In the Midlothian oil-shale fields of Scotland, as well as in the Elko field of Nevada, the shales are impregnated with petroleum in local spots only (where the volcanic rocks are) and not all over. In speaking of this feature Mr. Cadell in his paper before cited shows that although the oil-shale measures of Scotland, called calciferous sandstone series and 3,000 ft. thick, are well developed in other parts of Scotland, they have not yielded any oil shale outside of the Midlothian area, which was subjected to intense vulcanism after the deposition of the rocks of that series. The connection of the petroleum with the volcanics is therefore apparent, even in the oil-shale fields of Scotland.

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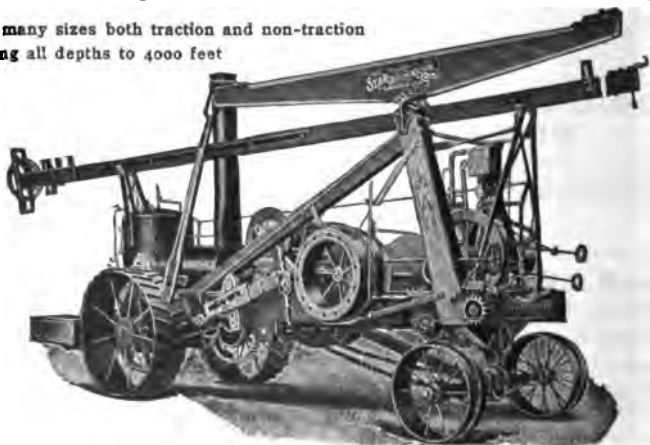
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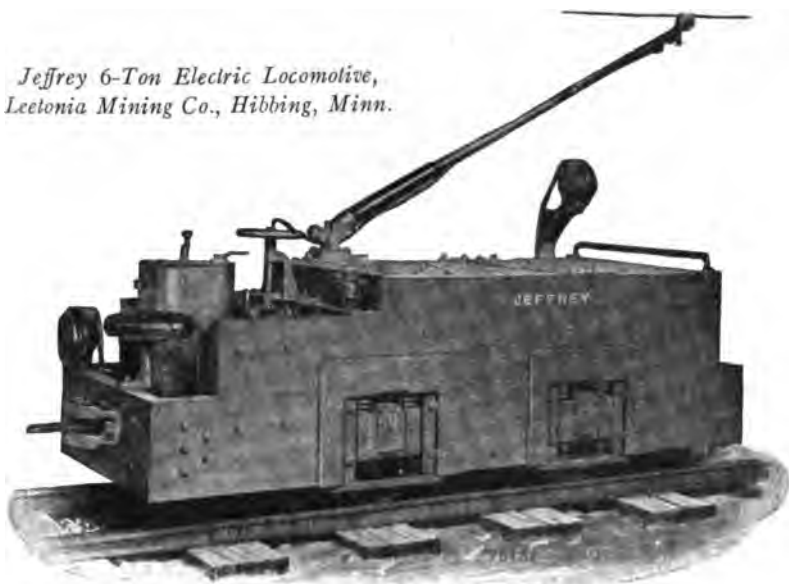
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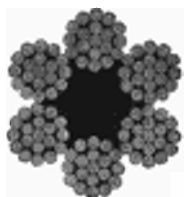
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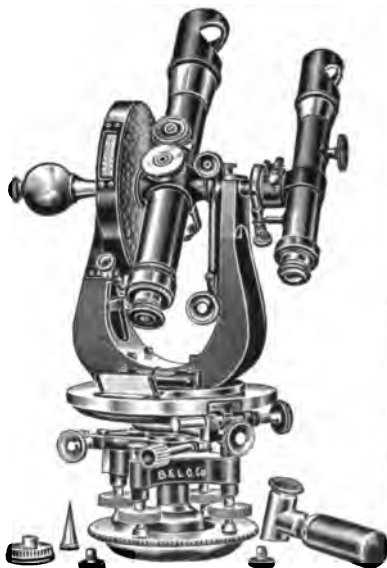
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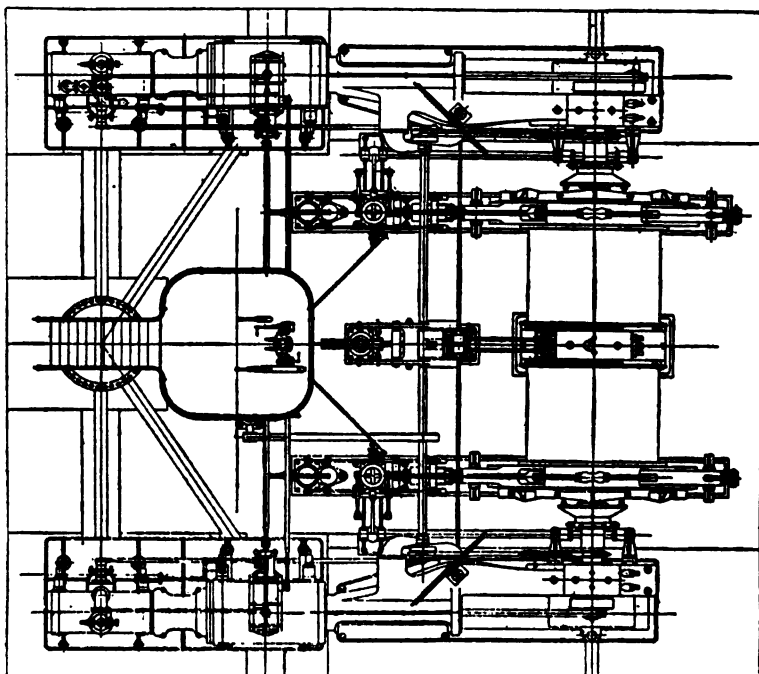
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JULY, 1914



Bulletin of the American Institute of Mining Engineers

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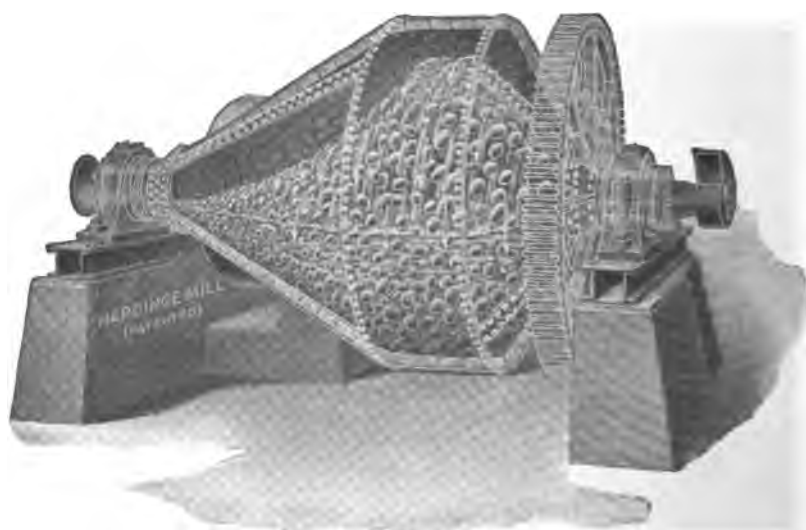
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Associates.

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Education, general and technical, when, where and how acquired,
with degrees, if any.

Dates	

[illegible]

_____ Signature

Dated _____ *191*

EXTRACTS FROM THE CONSTITUTION.

ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. **Members**; 2. **Honorary Members**; 3. **Associates**; 4. **Junior Members**. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

JUNE BULLETIN

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BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

No. 91

JULY

1914

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Entered as Second Class matter January 28, 1914, at the Post Office at
York, Pennsylvania, under the Act of March 3, 1879.

SALT LAKE CITY MEETING

The One Hundred and Eighth Meeting of the Institute,
for the presentation and discussion of technical papers,
will be held at Salt Lake City, Utah, August 10 to 14, 1914.

Institute Headquarters will be at the Hotel Utah, where
the technical sessions will be held.

In view of the heavy influx of tourists to Utah in August,
members who contemplate attending the Salt Lake meet-
ing are urged to communicate with the Secretary of the
Institute or the Chairman of the Local Committee, R. C.
Gemmell, P. O. Box 1169, Salt Lake City, Utah, at as
early a date as possible, in order that provision may be
made for suitable hotel accommodations.

BOARD OF DIRECTORS

Meeting of May 22, 1914.—President Thayer announced the appointment of the following Committee on Arrangements for the San Francisco meeting of 1915: Charles W. Merrill, Chairman; F. W. Bradley, Abbot A. Hanks, S. B. Christy, and H. C. Hoover.

President Thayer announced the appointment of the following Committee on Organization for the Utah-Nevada Local Section: R. C. Gemmell, Chairman; C. D. Blood, Lafayette Hanchett, Ernest Gayford; Duncan MacVichie, and George W. Riter.

The President of the Institute offered \$100, to be known as the "President's Prize," to be used as prizes for the best papers submitted by members of the Affiliated Student Societies, to be divided as follows: First Prize, \$50; Second Prize, \$30; Third Prize, \$20. A committee, consisting of J. F. Kemp, Chairman; W. L. Saunders, and T. T. Read, was appointed to report on the conditions to govern the award.

The President announced the appointment of the Committee on Mining and Milling Methods, the personnel of which is shown on a subsequent page of this *Bulletin*.

The following were appointed on the Advisory Committee to the U. S. Bureau of Standards, to represent the manufacture of the various non-ferrous metals as indicated: Copper, F. L. Antisell; Nickel, David H. Browne; Lead, H. O. Hofman; Aluminum, J. W. Richards; Zinc, G. C. Stone.

It was voted to hold a meeting of the Institute under the auspices of the Iron and Steel Committee, the Committee on Petroleum and Gas, and the Committee on Coal and Coke, in Pittsburg, Pa., Oct. 8, 9, and 10, 1914.

E. Gybbon Spilsbury was appointed as representative of the Institute on the special committee recommended by the Library Board to consider the general question of the duplication of exchanges of publications and the possibility of retrenchment of the expenses incurred by these exchanges.

W. L. Saunders was appointed a delegate to represent the Institute at the meeting of the National Foreign Trade Convention, May 27-28, 1914, at Washington, D. C.

Forty-four Members and five Associates were elected to membership in their respective classes.

The resignations of three Members were accepted.

The deaths of six Members were reported.

The By-laws of the Pennsylvania Anthracite Local Section were approved.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Recent graduate desires position as surveyor or assistant to mining geologist. No. 112.

Member, technical graduate with varied experience covering milling, cyaniding, plant design, mill and dam construction, underground work, and executive, examination and reporting on mining properties, desires position as superintendent of gold and silver property in United States or Latin America. No. 113.

Experienced cyanider, chemist, metallurgist, and mining engineer desires position as metallurgist or superintendent of mine and mill. No. 114.

Member, graduate engineer, aged 26, with experience in engineering with coal and iron mining companies, desires position as chief engineer in charge of surveying, drafting, construction. No. 115.

Member, technical graduate, aged 35, with experience as mining engineer, metallurgist, and manager, and as an instructor, desires position in practical work in mining or metallurgy, or as instructor in mining, metallurgy, and chemistry. No. 116.

Member, experienced superintendent and manager, successful in development problems or in operating mines in United States and Mexico, at present employed, desires change. No. 117.

Member, aged 27, married, at present assistant to general superintendent of bituminous coal mining company, technical graduate, familiar with metal mining in Michigan, Colorado, and Idaho, desires teaching position. No. 118.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period May 10 to June 10:

Rene de Sallier, New York, N. Y.

H. S. Chamberlain, Chattanooga, Tenn.

F. R. Pretzman, Oyster Bay, N. Y.

R. Eustice, South Australia.

F. C. Darby, South Australia.

Grant H. Tod, Livermore, Cal.

J. Murray Clark, Toronto, Canada.

J. H. Adams, Birmingham, Ala.

A. Faizon Dixon, Encontrador, Venezuela.

C. W. Merrill, San Francisco, Cal.

Francis Church Lincoln, Boston, Mass.

Karl Koelle, San Luis Potosi, Mexico.

W. A. Wilson, Salt Lake City, Utah.

P. E. Barbour, Candor, N. C.

R. C. Blanchard, London, England.

S. K. Tong, Shanghai, China.

W. T. Burns, Great Falls, Mont.

F. W. Bacorn, Butte, Mont.

H. B. Pulsifer, Chicago, Ill.

Dwight E. Woodbridge, Duluth, Minn.

Heinrich Ries, Ithaca, N. Y.

H. L. Slosson, San Francisco, Cal.

Frank R. Corwin has severed his connection with the Arizona Copper Co., Clifton, Ariz., and is now shift superintendent at the smelting plant of the Copper Queen Consolidated Mining Co., Douglas, Ariz.

Percy E. Barbour has resigned as manager of the Uwarra Mining Co., Candor, N. C., and will spend the summer in Maine.

Ralph E. Palmer, consulting engineer for the Rio Tinto Copper Co., has opened an office for consulting work at 3 Lombard Street, London, England.

Charles P. Perin, of New York, consulting engineer for the Tata Iron & Steel Co., India, has sailed for India, and will make extended stops in China and Japan.

Prof. H. R. Robbins, of the State College of Washington, will spend the summer making explorations in Yukon Territory.

On June 10, Dr. F. G. Cottrell, of the Western Division of the United States Bureau of Mines, and formerly of the University of California, delivered an illustrated lecture at Salt Lake City, under the auspices of the Utah Society of Engineers, on The Process of Smelter Smoke Control.

D. A. Lyon, of the U. S. Bureau of Mines, who during the past winter has given a series of lectures at Princeton and at Harvard on the subject

of electrothermics, has recently completed a series of lectures on the same subject at the University of Utah.

John A. Fulton is engaged to be married to Miss Maria Anita Bertheau, of San Francisco. The wedding will take place some time during the coming summer.

P. A. Seibert, who is connected with the Concordia mine, La Paz, Bolivia, is taking a vacation, his address being 35 bis rue de Villiers, Neuilly-sur-Seine, près Paris, France.

W. L. Remick has been transferred from the Alaska Gold Mining Co., Juneau, Alaska, to the Eagle River Mine, Eagle River, Alaska.

Edmund von Maltitz has resigned his position as Metallurgical Engineer of the Jones & Laughlin Aliquippa Works, Woodlawn, Pa., to become manager of the Hess Steel Castings Co., Bridgeton, N. J.

COLUMBIA LOCAL SECTION

Executive Committee

F. A. THOMSON, *Chairman*

GEORGE W. RODDEWIG, *Vice-Chairman*

L. K. ARMSTRONG, *Secretary-Treasurer*,

P. O. Drawer 2154, Spokane, Wash.

R. S. McCaffery

S. H. RICHARDSON

The quarterly meeting of the Columbia Local Section of the Institute was held in Pullman, Wash., May 16-17, 1914, as a joint meeting with the Spokane Sections of the American Institute of Electrical Engineers and the American Society of Civil Engineers, each holding separate sessions on the afternoon of the 16th with a joint meeting in the evening.

The Section meeting was held in Science Hall, the Chairman, Prof. Francis A. Thomson, presiding. E. Jacobs, Secretary of the Western Branch of the Canadian Mining Institute, acted as Joint Secretary. About 35 members and guests were present.

H. C. Bellinger, formerly Metallurgical Manager of the Great Cobar Co., Ltd., N. S. W., Australia, described the Horwood process.

Papers as follows were presented and discussed:

A Smelting Furnace for Cyanide Precipitate, by P. S. Anderson.

Mining Methods in Rossland, by C. H. Loux.

Geological Notes on a Part of Central Idaho, by Prof. D. C. Livingston.

Some Notes on the Ore Deposits of Republic, Wash., by A. M. Balfour.

E. Steidl, of the U. S. Bureau of Mines Rescue Car, spoke on the mine rescue service of the Bureau, and E. Jacobs described what is being done along similar lines in British Columbia and elsewhere.

The evening was occupied in joint session, J. B. Fiskén presiding. Mr. Fiskén read a short paper on the benefit to the young engineer of an association by membership in the local section of either of the societies represented in the meeting, and C. F. Uhden gave a stereopticon lecture on Power Development Along the Spokane River.

The social features of the meeting included a reception at the Gymnasium of the Washington State College and a dinner given by the Faculty

and served by the Domestic Science department of the College, with music by the College orchestra, at which about 175 persons were present. The joint session was followed by a dance at the Armory.

L. K. ARMSTRONG, *Secretary*.

BOSTON LOCAL SECTION

Executive Committee

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ALFRED C. LANE, *Vice-Chairman*

AUGUSTUS H. EUSTIS, *Secretary-Treasurer*, 131 State St., Boston, Mass.

TIMOTHY W. SPRAGUE

HENRY A. WENTWORTH

Twentieth Meeting

The Twentieth meeting of the Boston Local Section of the Institute was held at the Engineers' Club, 2 Commonwealth Avenue, Boston, Monday evening, May 4, 1914, at 7 o'clock. There were 28 members present. The Chairman, Henry L. Smyth, presided. Prof. Albert Sauveur delivered an address upon the nature of steel and its treatment, illustrated by lantern slides, and R. W. Wolfard showed a mathematical chart on which it is possible to represent the total work done by a steam engine, or an air compressor, or other similar machine, by a plotted scale on a straight line. The chart may be used to determine temperature-pressure-volume relations of gases and for other purposes.

AUGUSTUS H. EUSTIS, *Secretary*.

SAN FRANCISCO LOCAL SECTION

Executive Committee

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H. C. HOOVER, *Vice-Chairman*

ABBOT A. HANKS, *Secretary-Treasurer*, 630 Sacramento St., San Francisco, Cal.

F. W. BRADLEY

C. W. MERRILL

The second meeting of the San Francisco Local Section of the American Institute of Mining Engineers was held on the evening of March 31, in the quarters of the Engineers' Club, Hotel Sutter, Prof. S. B. Christy in the Chair and 40 members and guests present.

Gardner F. Williams delivered an interesting address on Current Views of the Origin of the Diamond, which was illustrated with lantern slides.

The total enrollment to date is 44.

ABBOT A. HANKS, *Secretary*.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS
AMERICAN SOCIETY OF MECHANICAL ENGINEERS
AMERICAN INSTITUTE OF MINING ENGINEERS
UNITED ENGINEERING SOCIETY

WILLIAM B. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining-maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining-reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

New Books on Mining

COAL MINING PRACTICE. Bull. 4, Illinois Coal Mining Investigations Co-operative Agreement. Urbana, 1914. (Gift of Illinois Coal Mining Investigations, etc.)
ELECTRIC BLASTING APPARATUS AND EXPLOSIVES: with special reference to colliery practice. By William Maurice. London, n. d.
MINE SIGNBOARDS. Technical Paper 67, U. S. Bureau of Mines. Washington, 1913.

New Books on Metallurgy and Chemistry

ELECTRIC FURNACES FOR MAKING IRON AND STEEL. Bull. No. 67, U. S. Bureau of Mines. Washington, 1914.
GRUNDLAGEN DER KOKS-CHEMIE. Ed. 2. By Oskar Simmersbach. Berlin, 1914.
HANDBUCH DER MINERALSCHEMIE. Band II, pt. 5. By C. Doelter. Dresden, 1914.
METALLURGY OF COPPER. By H. O. Hofman. New York, McGraw-Hill Book Co., 1914. Price \$5. (Gift of Publishers.)

[NOTE.—This is another volume in Professor Hofman's comprehensive series, of which I recently noticed his "General Metallurgy." Three further volumes are

announced as in preparation, on the metallurgy of lead (a revision and enlargement of his former treatise on that subject); the minor metals; and gold and silver. No metallurgist needs to be told how excellent are the qualifications of Professor Hofman for work of this kind. His career as a practitioner, investigator, instructor, critic and author, marked him long ago as an authority of the first rank. It is a boon and blessing to his students and professional colleagues that he is crowning his labors with this classic series of metallurgical treatises.

Professor Hofman's preface neatly characterizes the present volume as intended to meet the demands of the metallurgist of to-day, by presenting the principal physical and chemical facts about the metal, its alloys and its important compounds; recording whatever is of lasting value in the older practice; and describing in detail the methods of the present day. He is warranted in declaring that there does not exist another modern book which covers the entire ground as this one aims to do.

Moreover, the plan has been admirably executed, within the compass of 576 pages, with the aid of numerous illustrations, and of two other features, which deserve special mention. The first of these is the omission of those statements of general principles, etc., common to all metallurgical operations, which have been set forth already in Professor Hofman's "General Metallurgy." In this way, the book has been relieved from the annoying necessity, which has spoiled so many technical summaries, of giving so much space to inadequate preliminary disquisitions as to have too little space for adequate accounts of practice. The other feature is the abundant supply of footnote and bibliographical references, with the aid of which the reader can pursue, with regard to any subject treated, a thorough research, testing, as well as complementing, the opinions of the author. Among these sources of information, a surprisingly large number are private letters from persons named, proving with what intelligent and tireless industry Professor Hofman has collected the data for his conclusions by personal travel and correspondence, as well as by exhaustive study of technical literature. Students of metallurgy are well aware that treatises compiled from published essays only are too likely to omit important information, which could be secured by personal inquiry, but has not been put in print. Indeed, the leaders of practical progress in this art have found out a good deal more than they have told; and this is the result, not of their unwillingness, but of their lack of time, to tell. Usually, moreover, I have found that they dislike to "rush into print" with mere "reports of progress," preferring to wait until they can offer something more conclusive. Yet I have not found them, as a rule, unwilling to answer questions frankly; and I am grateful when an authority like Professor Hofman, enjoying their confidence, and knowing what he wants which they can give him, taps this reservoir of unpublished knowledge for the benefit of us all.—R. W. R.]

SCREEN SCALE SIEVES MADE TO A FIXED RATIO. By G. A. Disbro. N. p., n. d. (Gift of W. S. Tyler Co.)

TASCHENBUCH DER CHEMISCHEN TECHNOLOGIE. Vols. 1-2. By Wilhelm Bersch. Wien, 1914.

New Books on Geology and Mineral Production

CERTAIN PHASES OF SUPERFICIAL DIFFUSION IN ORE DEPOSITS. By R. A. F. Penrose, Jr. N. p., n. d. (Gift of Author.)

CONTRIBUTIONS TO ECONOMIC GEOLOGY, 1912. Part 1—Metals and non-metals except fuels. Bull. 540, U. S. Geological Survey. Washington, 1914.

ENGINEERING GEOLOGY. By Heinrich Ries and T. L. Watson. New York, J. Wiley & Sons, 1914. Price \$4. (Gift of Publishers.)

[NOTE.—The authors of this book are professors of economic geology, the first at Cornell, and the second at the University of Virginia; and the book is based upon the instruction given by them, in special courses to students of civil engineering, on geology as applied in that art. The title is not felicitous, since the subject is not a particular kind of geology. But the work itself is a useful compendium of such facts and principally of geology as may advantageously be taught to hard-worked students of engineering who cannot be expected to master all the departments of that now complex and comprehensive science. It deals in seventeen chapters with the rock-forming minerals; the general characters, mode of occurrence, and origin of rocks; their structural features and metamorphosis; their weathering and the resulting soils; the action of rivers; the circulation and work of underground waters; the effects of landslides; wave-action, shore-currents, and their relation to coasts and harbors; the origin of lakes and their relation to engineering work; glacial deposits, and their economic bearing; building-stone; limes, cement and plaster; clay and clay-products;

petroleum, natural gas and other hydrocarbons; road-foundations and road-materials; and, finally, ore-deposits. It will be seen that, with the exception of the last chapter, the book deals with materials which the civil engineer must use, or encounter as obstacles, in his work of building, excavating, dredging, irrigating, etc., and the origin, nature and consequent behavior of which he ought therefore to understand. The last chapter presents a condensed statement of the science of ore-deposits, not sufficient for mining engineers or economic geologists, yet not directly useful to anybody else, except as a stimulus to further study. Yet this chapter, like all the rest, is complemented by bibliographic references, not too numerous or antiquated, but exceedingly well chosen, to the literature of the subject. On the whole, the authors must be praised for giving so small a space to this department of economic geology, which, of course, they could not have omitted altogether. They are to be congratulated upon a work which occupies a sphere of its own, and will be found useful by teachers and practitioners alike.—R. W. R.]

GEOLOGY AND MINERAL RESOURCES OF THE YILGARN GOLDFIELD. Part 1—Southern Cross. Bull. 49, Western Australia Geological Survey. Perth, 1913.

MINERALS OF THE BLACK HILLS. Bull. 10, South Dakota School of Mines Rapid City, 1914.

MINERAL RESOURCES OF SOUTHWESTERN OREGON. Bull. 546, U. S. Geological Survey. Washington, 1914.

New Books on Non-Metallic Minerals

CEMENT MATERIALS AND INDUSTRY IN THE STATE OF WASHINGTON. Bull. 4, Washington Geological Survey. Olympia, 1913.

CLAY AND SHALE DEPOSITS OF THE WESTERN PROVINCES. Part II, report on. Memoir 25, Canada Department of Mines, Ottawa. 1913.

CONSERVATION OF COAL IN CANADA. Toronto, 1914. (Gift of Canada Conservation Commission.)

DIFFERENTIATION OF NATURAL AND OIL ASPHALTS. By E. C. Pailler. N. p., n. d. (Gift of Clifford Richardson.)

DRILLING WELLS IN OKLAHOMA BY THE MUD-LADEN FLUID METHOD. Technical Paper 68, U. S. Bureau of Mines. Washington, 1914.

MEXICAN FUEL OIL. New York, Anglo-Mexican Petroleum Products Co., 1914. (Gift of Publishers.)

THE SILICATES IN CHEMISTRY AND COMMERCE. By W. and D. Asch; translated by A. B. Searle. London, 1913.

New Books on General Subjects

CAMBRIA STEEL. A Handbook of information relating to structural steel manufactured by the Cambria Steel Co., 1914. Johnstown, 1914. (Gift of George E. Thackray.)

Trade Catalogues

BRACKETT, F. W. & Co., Colchester, England. The "Elmore" sampler. 1914.

BUCHANAN, C. G. Co., New York, N. Y. Bull. 9, Buchanan all-steel rock and ore crusher, Type "B."

CHICAGO PNEUMATIC TOOL Co., Chicago, Ill. Bull. 154, The "Chicago Stoper." Feb., 1914.

COLORADO IRON WORKS Co., Denver, Colo. Catalogue 12-F, Some details as to smelting practice and equipments.

DENVER FIRE CLAY Co., Denver, Colo. Case furnaces. 80 pp.

ELECTRIC STORAGE BATTERY Co., Philadelphia, Pa. Bull. 146, The "Ironclad-Exide" battery for storage battery locomotives. May, 1914.

LAIDLAW-DUNN-GORDON Co., Cincinnati, Ohio. Bull. L-523-A, Cincinnati gear duplex Corliss steam-driven air compressors, Class WA and XA. Jan., 1914.

LESCHEN & SONS ROPE Co., St. Louis, Mo. Leschen's Hercules. May, 1914.

SPARTA IRON WORKS Co., Sparta, Wis. The Sludge Bucket. June, 1914.

SULLIVAN MACHINERY Co., Chicago, Ill. Core drilling by contract with Sullivan diamond core drills. 1914.

SUPPLEE-BIDDLE HARDWARE Co., Philadelphia, Pa. Monel metal. May, 1914.

UNDERPINNING & FOUNDATION Co., New York, N. Y. "Hercules" steel pile foundations.

NEW MEMBERS

Member's

- McLELLAN, JAMES J., Mine Supt. Webb City, Mo.
 MILLER, JOSEPH EDWIN, County Surveyor. Lewistown, Mont.
 NELSON, CHRISTOPHER WALTER, Supt. Granite Hill Mines, Box 105, Reno, Nev.
 NORDENHOLT, GEORGE DIETRICH, Min. Engr., 1219 Hollingsworth Bldg.,
 Los Angeles, Cal.
 OHASHI, TAKISHI, Prof. of Mining, Kumamoto High Tech. School, Kumamoto, Japan.
 PEACOCK, DAN CONLEY, Min. Engr. 515 N. 5th St., Brainerd, Minn.
 PELTON, HAROLD A., Chem. Care Union Oil Co. of Cal., Avila, Cal.
 POGUE, JOSEPH E., Geol. U. S. Geological Survey, Washington, D. C.
 POMEROY, RALPH EUGENE H., Smelter Supt., Steptoe Valley Smelting
 & Mining Co., McGill, Nev.
 PORCH, EDWIN L., JR., Min. Engr. Verona, Mich.
 REILLY, WILLIAM C., Genl. Supt., Youngstown Sheet & Tube Co., Youngstown, Ohio.
 RIEDE, FRED EDWARD, Met. Care Bertha Mineral Co., Austinville, Va.
 ROBERTSON, JASPER T., Mine Supt. Golden Cycle Mining Co., Goldfield, Colo.
 ROCKWELL, FRED GORE, Min. Engr. Box 55, Ishpeming, Mich.
 ROGERS, GEORGE RICHARD, Min. Engr., Gen. Mgr., Mann Mines, Ltd.,
 Box 609, Gowganda, Ont., Canada.
 RUFF, RICHARD A., Min. Engr. Crozer Land Association, Elkhorn, W. Va.
 RUST, HENRY BEDINGER, Special Representative, Babcock & Wilcox Co.,
 1177 Murray Hill Ave., Pittsburgh, Pa.
 RYPINSKI, J. E., Min. Engr., Aguacate Mines. San Mateo, Costa Rica, C. A.
 SALLIER DU PIN, RENE DE, Min. Engr. 25 Broad Street, New York, N. Y.
 SCHABER, CARL F., Min. Engr. Tyrone, N. M.
 SCHRAPS, PAUL CURTIS, Foreman, Cyanide Plant, So. Amer. Dev. Co.,
 Box 655, Guayaquil, Ecuador, S. A.
 SOMERS, RANSOM E., Instr. in Econ. Geol. Cornell Univ., Ithaca, N. Y.
 STAMPHER, JOSEPH P., Min. Engr., Underground Supt.,
 Empire Steel & Iron Co., Wharton, Mt. Hope, N. J.
 STEVEN, HUGH MACGREGOR, Min. Engr., Canadian Min. & Finance Co.,
 Timmins, Ont., Canada.
 STOTESBURY, HAROLD W., Min. Engr. Tonopah Mining Co., Tonopah, Nev.
 SUYDAM, ALBERT GODFREY, Min. Engr., Met. Ohio Copper Co., Lark, Utah.
 TANNER, WALLACE N., Chief Engr. Anaconda Copper Mining Co., Anaconda, Mont.
 THOMSON, HERBERT G., Cyaniding. Care Globe Con. Min. Co., Dedrick, Cal.
 UYENO, KAGEAKI. 39 Sakuradamachi, Azabu, Tokyo, Japan.
 VALERIUS, M. M., Geol. Clinton Bldg., Tulsa, Okla.
 WALLACE, WILLIAM JOHN, Supt., Cusi Mining Co., Cusiuhuirachi, Chih., Mexico.
 WHITE, CHARLES KENNETH, Min. Engr. Treadwell, Alaska.
 WHITTINGHAM, HAROLD, Min. Engr., Supt., Miniera Gennamari, Ingurtosu,
 Arbus, Sardinia, Italy.
 WICKSTROM, CLARENCE L, Cons. Engr. 314 Hyde Blk., Spokane, Wash.
 WILSON, JAMES W., Supt., Lower Dept., National Tube Co., McKeesport, Pa.
 WRAITH, CHARLES R., Chem. Anaconda Copper Mining Co., Anaconda, Mont.
 WRIGHT, PERCY E., Mech. Engr., Coal Mining Plants, 424 New York Block,
 Seattle, Wash.

Associate Members

- HO, CHIEH. Sec. B., Taylor Hall, Lehigh Univ., So. Bethlehem, Pa.
 IMBRIE, JAMES, Banker. William Norris Imbrie & Co., 61 Broadway, New York, N. Y.

Junior Members

- CORNELISSEN, JOHANNES, Student. Hartley Hall, Columbia Univ., New York, N. Y.
 MOSES, PERCIVAL SNEED, Student, Kappa Sigma House,
 Colo. School of Mines. Golden, Colo.

CANDIDATES FOR MEMBERSHIP

The following persons have been proposed during the period Apr. 10 to May 10, 1914, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communi-

cations, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Charles Edward Addams, Ray, Ariz.

Proposed by Seeley W. Mudd, Louis S. Cates, W. S. Boyd.

Born 1881, Plymouth, Pa. 1901-11, Chief Min. Engr. and Mgr. Wesselson and Dutoetspan Mines, DeBeers Con. Diamond Mines, Ltd., Kimberley, S. Africa. 1911-12, Genl Mgr., Voorspoed Diamond Mining Co., Ltd.

Present position: 1913 to date, Asst. Supt., Ray Cons. Copper Co., Ray, Ariz.

Harry H. Alexander, Maurer, N. J.

Proposed by F. C. Newton, Clarence P. Linville, L. V. Emanuel.

Born 1866, Lexington, Va. Grad. Va. Niel Institute. Special student in Chemistry, University of Va., and School of Mines, Columbia. 1899-1908, Supt. and Genl. Mgr., Colorado Dept., American Smelting & Refining Co.

Present position: 1908 to date, Asst. Mgr. and Mgr., Perth Amboy Plant, American Smelting & Refining Co.

Arthur Watts Allen, London, E. C., England.

Proposed by F. Lynwood Garrison, R. H. Sanders, Eli T. Conner.

Born 1879, Devon, England. Technical Schools, Plymouth, Eng. 1896-1902, St. John's College, Cambridge, Eng.; B. A. 1910, M. A. 1904-05, Cyanide Mgr., British Exploration Co., Australia. 1905-07, Met., Westralia Mount Morgans Co., Australia. 1907-09, Met., Palmareijo de Mexican Co., Mexico. 1909, Cons. Met., Rio Plata Co., Mexico. 1909-10, Cons. Met., Torres Mines Co., Mexico. 1910-11, Met., Uruguay Mines Co., S. A.

Present position: 1912 to date, Met., Rhodesia Gold Mining & Investment Co., and Reduction Works Mgr., Lonely Reef Mining Co., S. Africa.

Mark N. Alling, Downieville, Sierra Co., Cal.

Proposed by F. Lynwood Garrison, W. S. Morley, E. A. Hersam.

Born 1879, Stockton, Cal. 1899, Grad. Stockton High School. 1903, Univ. of Cal., College of Mines. 1901-02, Miner, Hidden Treasure Drift Mine, Cal. 1903, Assayer, Gold Bluff Mg. Co., Cal. 1904, Gravel examination work, Nome, Alaska. 1907, Engineered the McKay 17-mile ditch, Nome, Alaska. 1905, Engineering work, Solomon River Mg. Co., Alaska. 1906, Engineering work, Northwestern Dev. Co., Alaska. 1907-08, charge of Gravel mine, Nome, Alaska. 1908, tracing, locating, and developing ancient river beds of Cal.

Present position: Mining and Civil Engineering office, Downieville, Cal.; County Surveyor of Sierra County.

Leslie Douglass Anderson, Midvale, Utah.

Proposed by W. G. Sharp, C. F. Moore, Ernest Gayford.

Born 1879, Ishpeming, Mich. 1896, High School. 1901, Univ. of Michigan; B. S. in M. E. 1901-02, Erecting machinist and draftsman, Nordberg Mfg. Co. 1902-03, Draftsman, Bucyrus Co. 1903-04, Mech. Engr., Champion Copper Co. 1904-05, Checker, Bucyrus Co. 1905-06, Chief Engr., Arizona Smelting Co. 1906-12, Draftsman and Asst. Chief Engr., U. S. Smelting, Refining & Mining Co.

Present position: 1912 to date, Supt., U. S. Smelting Co.

John Carter Anderson, Coalinga, Cal.

Proposed by Franklin W. Smith, Ernest D. Hersam, T. T. Read.

Born 1886, Woonsocket, S. D. 1904, Sioux City, Iowa, High School. 1909, School of Mines, Univ. of Cal., Berkeley, Cal. 1906, Greene Cons. Copper Co., Cananea, Mexico, surveying. 1907, Arizbe Mining Co., Cananea, Mexico. Underground and surface surveying. 1908-1909, Director and Treas., Anderson Packing Co., Sioux City, Iowa. 1909, Transitman, City Engr. Sioux City, Iowa, School of Mines, Univ. of Cal. 1910, Kern Trading & Oil Co., Coalinga and Casmalia, Cal. 1911-13, Director and Sec'y, Taft Petroleum Co., Casmalia, Cal.

Present position: Standard Oil Co., Coalinga, Cal.

Charles Seelye Arms, Toledo, Ohio.

Proposed by J. Burns Read, A. W. Smith, Mark A. Ammon.

Born 1890, Cleveland, Ohio. 1904-08, Central High School, Cleveland, Ohio. 1908-12, Grad. in Min., B. S., Case School of Applied Science, Cleveland, O. 1912-13, Student in Metallurgy and Metallography, Royal Technical School, Charlottenburg, Germany. Summer, 1911, Underground work, Silver King Mine, Park City, Utah.

Present position: 1913 to date, Asst. Met., Willys-Overland Co., Toledo, Ohio.

Arthur Austin, Tooele, Utah.

Proposed by Leonard S. Austin, Alonzo F. Bardwell, William Wraith.

Born 1884, Murray, Utah. 1901, Grad. Manual Training High School, Denver, Colo. 1905, Grad. Colorado School of Mines; E. M. 1906, Special work, Michigan College of Mines. 1906-11, Chemist, Washoe Reduction Wks., Anaconda, Mont. 1911-14, Chemist, Tooele Smelter, International Smelting & Refining Co., Tooele, Utah.

Present position: Chief Testing Engineer, Tooele Smelter, Tooele, Utah.

John Baragwanath, Morococha, Peru.

Proposed by J. F. Kemp, Charles P. Berkey, Alfred J. Moses.

Born 1884. Grad. Columbia College about 1907 and specially devoted himself to geology, mining and metallurgy. Remained as a graduate student for a year, taking a Master's degree in these branches. Was a member of an expedition of the American Museum in Ecuador for a year; became engineer of a placer property in Ecuador; later of a deep mine whose complicated vein system he mapped and solved; and is now captain of the San Francisco Mine of the Morococha Co., Peru.

Present position: Captain, San Francisco Mine, Morococha, Peru.

Alexander Conn Beeson, Washington, Pa.

Proposed by J. M. Clark, C. E. Krebs, J. S. Cunningham.

Born 1876, Mt. Braddock, Pa. 1897, Washington and Jefferson College, Washington, Pa.; B. S. 1897-99, Taught French, German and Math., Chambersburg Academy, Chambersburg, Pa. 1899-1903, H. C. Frick Coke Co., afterward U. S. Steel Corp'n., as Chairman, Levelman, Transitman and Asst. Engr., on mining, construction and railroad work, including the building of several coke plants complete. 1903-04, Division Engr., Pittsburg-Buffalo Co. of Pittsburg, Pa. Designed and built coal plants for this company at Burgettstown, Pa., and Johnetta, Pa. 1904-09, Construction Engr., Pittsburg-Buffalo Co., designed and built coal plants at Canonsburg, Pa., and Marianna, Pa.

Present position: 1909 to date, Chief Engr., Pittsburg-Buffalo and Four States Coal & Coke Companies.

Thomas Parry Billings, Salt Lake City, Utah.

Proposed by R. C. Gemmell, Alfred Frank, W. A. Wilson.

Born 1883, Salt Lake City, Utah. 1889-97, Salt Lake Public Schools. 1897-1901, Salt Lake High School. 1901-03, Univ. of Utah. 1904-06, Univ. of Utah; B. S., E. M. Nine years' practical experience. 1903-04, Hydraulic Survey, State of Utah, helper. 1906, Transitman on railroad and power plant locations. 1907, Field Engineer, Ohio Copper. 1908, Millman, Boston Con. Co. 1908, Transitman, Utah Copper Co. 1909-10, Min. Engr., Nevada Utah Min. Co. 1910, Assayer, Bingham Mines Co.

Present position: 1910 to date, Min. Engr., Bingham Mines Co. and Eagle Bluebell Co.

Reginald G. Bowman, Salt Lake City, Utah.

Proposed by William Wraith, E. P. Mathewson, F. F. Frick.

Born 1886, Pueblo, Colo. 1891-1903, Public Schools, Peru, Ind. 1904-06, Colo. School of Mines, Golden, Colo. 1907-08, Special work, Organic Chemistry. George Washington Univ., Washington, D. C. 1909-11, Colo. School of Mines, Golden, Colo.; E. M. 1903-04, Asst. Chemist, Peru Steel Casting Co., Peru, Ind. 1906-07, Asst. Assayer, Indicator Cons. Gold Mining Co., Independence, Colo. 1907-08, Asst. Chemist, U. S. Bureau Int. Rev., Washington, D. C. 1908-09, Asst. Examiner, U. S. Patent Office, Washington, D. C. 1910, Field Geol., Colo. State Geological Survey. 1911-12, Asst. Engr., Butte, Anaconda & Pacific Ry., Anaconda Mont. 1912-13, Chemist and Foreman, Copper Extraction Co., Anaconda, Mont.

Present position: Statistician, International Smelting Co.

Jesse Taylor Boyd, Arden, Nev.

Proposed by Russell B. Paul, Cony T. Brown, Robert Hursh.

Born 1883, Cornwall, Pa. Common School, Lebanon, Pa. 1903-08, Colorado School of Mines; E. M. 1896-99, General experience in machine shops and mine, Cornwall Ore Banks Co., Cornwall, Pa. 1899-1901, Lackawanna Iron & Steel Co., engineer corps, coke oven construction. 1901-03, Railroad Contracting for Penn. Nickle Plate Road. 1903-1911, General experience with Camp Bird Mines; vacations at mine school, 1904-08; 1909-1911, with company as resident engineer. 1912, Genl Supt. North American Smelting Co., Perth Road, Ont. 1913 to present, with Empire Zinc Co. as engineer.

Present position: Supt., Potosi Mine, Arden, Nev.

Harry Cyril Boydell, Brokpan, Transvaal, S. Africa.

Proposed by James E. Thomas, Walford R. Dowling, Thomas T. Read.

Born 1873, Berkshire, England. 1890-95, New Zealand Univ.; B. S. Otago School of Mines (N. Z.) 1896, First-Class Mine Manager's Certificate, N. Z. 1913, First-Class Mine Manager's Certificate, Transvaal. 1895-97, Mine Manager, Preece's Point Co., Coromandel, N. Z. 1898, Met., New South Wales Goldfields, Ltd., Sydney. 1899, Mgr., Sunlight G. M. Co., Hillgrove, N. S. W. 1900-07, Lecturer, Bendigo School of Mines, Victoria. 1908-11, Practising as Min. Engr. in Bendigo and Melbourne.

Present position: Government Gold Mining Areas, Ltd., Transvaal, S. Africa.

Eric Vaughan Burnett, Collahuasi, Chile, S. A.

Proposed by Walter T. Holberton, L. C. Stuckey, Lee Fraser.

Born 1878, East Acton, Eng. 1893-1896, Apprenticed to Henderson & Son, Member, Inst. of C. E., Truro, Cornwall, carrying on profession of Engineers and Surveyors to nearly all mines in Cornwall and also Civil Engineers to Townships, etc. 1896-1900, Asst. Mgr., Pile Works of the Pile and Blaina Manganese works, also surveyor to various Collieries in the district of Glamorganshire, Wales. 1901-02, Assayer and Surveyor to Clintons Gold Concessions, Axim, West Africa, also of Lawrence Pountney Hill, E. C. 1902-03, Head Surveyor and Assayer to the New Gold Coast Agency of Old Jewry, London, E. C., and of Tarkwa Gold Coast. 1903-05, Chief Assayer and Surveyor to the Abosso Gold Mining Co., Ltd., Tarkwa, West Africa. 1905-07, Chief Engr., Argentine Sierra del Fuego Exploration Co., Buenos Aires. 1907-08, Chief Engr., Mining Exploration Co., Ltd., Santiago, Chile. 1908-14, Chief Engr., Société des Mines de Cuivre de Catemes (de Paris) Chile. 1911, Elected Associate, Institute of Mining and Metallurgy, London. 1914, Member of the above Institute.

Present position: Chief Engr., Sections North and East of the Société des Mines de Cuivre de Collahuasi, la Grande, Collahuasi, via Antofagasta, Chile, S. A.

James Watts Caples, Salmon, Idaho.

Proposed by R. B. Caples, Jr., G. H. Cox, C. R. Forbes.

Born 1881, Glasgow, Mo. 1891-1901, Pritchett College, Glasgow, Mo., starting with Intermediate Dept., then Preparatory Dept., followed by four years of College work; B. A. 1902-05, Missouri School of Mines; B. S. 1900, Vacation, Chicago & Alton R. R. Co., in Engineering Corps. 1901-02, Rodman and computer, Terminal R. R. Association of St. Louis, Mo. 1902, Levelman and resident engineer, St. Louis, Memphis & Southeastern R. R. 1903, Levelman, Denver, Northwestern Pacific R. R. 1904, Vacation, Russell & Sharp, Cripple Creek, Colo. 1905-06, Special work at Missouri School of Mines. 1906-14, Member of Herndon & Caples and Lumby Engineering Co., Salmon, Ida.

Present position: Mining and Civil Engr., U. S. Deputy Mineral Surveyor.

William Howes Coghill, Evanston, Ill.

Proposed by U. S. Grant, H. H. Stock, F. W. DeWolf.

Born 1876, Roseville, Ill. 1894-96, Academy and College work at Shurtleff College, Upper Alton, Ill. 1899-1903, Colo. School of Mines; E. M. Some post-graduate work at Univ. of Chicago and Northwestern Univ. 1902-03, Surveyor and Assayer, Colo.-Cal. M. & M. Co., Idaho Springs, Colo. 1903, Surveying mining claims, Bisbee, Ariz. 1904, Asst. Engr., Calumet & Ariz. M. Co., Bisbee, Ariz. 1905-06, Asst. Engr., Tamarack M. Co., Calumet, Mich. 1906-07, Chief Engr., United Zinc Co., and Supt., Highland Mine, Joplin, Mo. 1907-13, Instructor Min. and Met., Northwestern Univ., Evanston, Ill. 1913, Asst. Prof. Min. and Met., Northwestern Univ.

Present position: Asst. Prof. Min. and Met., Northwestern Univ., and operating Ore Testing Laboratory.

Edgar Maurice Cortright, Red Lodge, Mont.

Proposed by Harry Boyle, A. F. Blair, Amos Slater.

Born 1884, Mauch Chunk, Pa. 1900, Mauch Chunk High School. 1900-01, Mt. Pleasant Military Acad. 1901-02, Bethlehem Prep. School. 1902-05, Univ. of Penn. 1905, Penn. R. R. 1906, Lehigh Valley Coal Co., Lehigh Coal & Navigation Co. 1907, Northwestern Improvement Co. 1909, City Engr., Red Lodge, Mont. 1910-12, Res. Engr., Mont. Coal & Iron Co.

Present position: Res. Engr., Northwestern Improvement Co., Red Lodge, Mont.

Carlos Garcia Cross, Vina del Mar, Chile, So. Amer.

Proposed by Mark R. Lamb, T. H. Barclay, Ralph Wilcox.

Born 1872, Santiago, Chile. Grad. Univ. of Chile.

Present position: Part owner and Mgr., Compania Minera Las Vocas.

Harry L. Day, Wallace, Idaho.

Proposed by Horace V. Winchell, E. Percy Smith, Rush J. White.

Born 1865, Dayton, Nev. Public Schools of Nevada and Sacramento Counties, Cal. 1885, Grad. St. Mary's College, San Francisco, Cal. 1901-12, Genl. Mgr., Hercules Min. Co., Pres., Tamarack & Custer Cons. Co. 1912, President and Genl. Mgr., Federal M. & S. Co. Twenty-five years' practical experience as prospector, miner, millman and operator.

Present position: Pres. and Genl. Mgr., Federal Min. & Smelt. Co.

William Harrison Dufur, Oil City, Pa.

Proposed by Anthony F. Lucas, Philip W. Henry, David T. Day.

Born 1859, Chelsea, Mass. Common School in Penn. Have been connected with every branch of the oil business extensively, with the exception of transportation. Also have been employed by Philip W. Henry, Southern Pacific Co., Standard Oil and others, as consulting expert.

Present position: Producer of Oil.

William Nicol Ellis, Wallace, Idaho.

Proposed by J. T. Wright, W. H. Teel, Rush J. White.

Born 1889, Leesburg, Fla. 1908-09, McGill Univ. College; B. C. 1909-12, Univ. of Idaho; Min. Engr. Underground work one year, Last Chance Mine, Wardner, Ida. 1912, Assayer, Hercules Min. Co., Wallace, Idaho. 1910-12, one year underground at Last Chance Mine, Wardner, Ida.

Present position: 1912 to date, Assayer, Hercules Mining Co., Wallace, Idaho.

Keith Colt Fraser, Bartlesville, Okla.

Proposed by G. H. Cox, C. R. Forbes, H. A. Buehler.

Born 1886, Lyndonville, N. Y. 1904, Grades and High School, Lyndonville, N. Y. 1906-10, Missouri School of Mines; B. S. 1909, Summer, Tending vanners at mill of Utah Copper Co. 1910, Chainman on Wm. Perkin's engineering corps in Mont. 1910-11, Mucker and chainman, Bunker Hill & Sullivan Min. & Mill Co., Kellogg, Ida. 1911-12, Sample foreman and chemist, Bartlesville Zinc Co. 1912, Miner, Copper Queen Cons. Min. Co., Bisbee, Ariz. 1912-13, Chainman, Bunker Hill & Sullivan Min. & Mill Co. 1913, Asst. Supt., National Zinc Co., Bartlesville, Okla. 1913-14, Yard Foreman and Chemist, Bartlesville Zinc Co.

Present position: Asst. Supt., Bartlesville Zinc Co., Bartlesville, Okla.

Walter Edward Gaby, Salt Lake City, Utah.

Proposed by Robert H. Bradford, Dorsey A. Lyon, Robert S. Lewis.

Born 1890, Salt Lake City, Utah. 1905-09, Scientific Course, Salt Lake High School. 1909-14, Univ. of Utah; B. S. in Min. Engrg. Two years of varied experience in mill and smelter work, mining and railroad surveying during vacations and one year spent out of school. 1911, Smelter Supply Acct., Steptoe Valley Milling & Smelting Co., McGill, Nev. 1912, Topographer, Utah Railway Co., Salt Lake City, Utah. 1912, Sampler in mill, Steptoe Valley Mill. & Smelt. Co. Timekeeper, Nevada Cons. Min. Co. 1913, Asst. Surveyor, Giroux Cons. Min. Co. 1913, Assayer, Veteran Mine, Nevada Cons. Min. Co. Machineman, Daly-West Min. Co., Park City, Utah. Computer, Oregon Short Line R. R. Co. 1914, Student, Univ. of Utah.

Present position: Student Stanford Univ., summer geological survey. *En route* to Stanford field-geology trip.

Arthur John Garrett, Lansford, Pa.

Proposed by J. B. Warriner, Edwin Ludlow, W. G. Whildin.

Born 1887, Ashland, Pa. Previous to 1900, Orwigsburg, Pa., Public School. 1900-04, Orwigsburg, Pa., High School. 1904-08, Penn. State College; B. S. 1908-10, Transitman, Philadelphia & Reading Coal & Iron Co., Pottsville, Pa. 1910, Transitman, Northwestern Improvement Co., Red Lodge, Mont. 1910-11, Chief Draftsman, Northwestern Improvement Co., Tacoma, Wash. 1911-12, Min. Engr., Denny-Renton Clay & Coal Co., Taylor, Wash.

Present position: 1912 to date, District Engr., Lehigh Coal & Navigation Co., Lansford, Pa.

Walter Scott Grether, Kamloops, B. C., Canada.

Proposed by G. H. Cox, C. R. Forbes, H. A. Buehler.

Born 1882, Bunker Hill, Ill. 1899, Grad., Greenfield, Mo., High School. 1901, Grad., St. Louis Manual Training. 1906, Grad., Missouri School of Mines. 1901-02, Mach.-Apprent., Mo. Pac. Ry., St. Louis, Mo. 1906-08, Engr., Federal Lead Co., Flat River, Mo. 1908-09, Engr., Cons. Pittsburgh Plate Glass Co., Crystal City, Mo. Locating Engineer, Oregon Short Line Ry., Salt Lake City, Utah. 1909-10, Asst. Masonry Supt., International Smelting & Refining Co., Tooele, Utah. 1910-14, Engr., Supt., Cons. and Mill Supt., Rosiclare Lead & Fluorspar Mines, Rosiclare, Ill.

Present position: Supt., Kamloops Copper Co., Kamloops, B. C., Canada.

William George Haldane, Golden, Colo.

Proposed by D. W. Brunton, Frank M. Taylor, Edward W. Keith.

Born 1878, Cleveland, Ohio. 1885-92, Public Schools, Cleveland, Ohio. 1892-95, Central High School, Cleveland, Ohio. 1896-1900, Case School of Applied Science; B. S. Summer session at Columbia Univ., studying under Dr. Wm. Campbell. 1899-1900, Asst. Engr., Amer. Steel & Wire Co. 1900, Colorado School of Mines, Instructor in Mathematics and Descriptive Geometry, Instructor in Metallurgy, Asst. Prof. Metallurgy, Associate Prof. Metallurgy, Mgr., Experimental Plant.

Present position: Acting President, Colorado School of Mines.

John Bradstreet Harper, Salem, Mass.

Proposed by H. P. Henderson, H. A. J. Wilkens, W. B. Devereux, Jr.

Born 1882, Flushing, N. Y. 1896-1901, St. Paul's School, Concord, N. H. Passed preliminary examinations for Harvard but did not enter. Since that time have privately studied mathematics, surveying, assaying, ore dressing, mechanics and construction, geology and mineralogy, and other subjects relative to mining and milling. Have written articles on cyaniding, surveying, etc., for *Mining Science* and *The Mining World*. 1903, Acted as agent, timekeeper, under A. F. Wuensch, on copper prospect, Milagros, N. M. 1904, worked at various jobs in mills and underground, Yavapai County, Cal., and Clear Creek County, Colo. 1905, Miner and Contractor, Kokomo Mine, Central City; in charge of small prospect, Caribou. 1906, worked as lessee on several tungsten prospects at Nederland and on lead-silver, Georgetown. 1907, Miscellaneous mill and underground work in Boulder County, prospecting in Routt and Larimer Counties. 1908, Supt., Development, Rosebud Syndicate, Rosebud, Nev. 1909, Examination of prospects with P. Geo. Gow, Cal.; mill supt., Chafey Reduction Co., Chafey, Nev. 1910, Miscellaneous examination of prospects and surveying. 1911, Milling, sampling and miscellaneous work, Four Hills Min. Co., Plumes County, Cal. 1912, Prospecting in Nevada and Idaho; Cyaniding, assaying, etc., Dun Glen, Nevada. Examination of prospects, Eagle County, Colo., and Yavapai, Ariz. 1912-14, Contractor on underground development, also assaying, surveying, etc. Vermont Copper Co., So. Strafford, Vt.

Present position: Supt., Copper Queen Gold Min. Co., Stoddard, Yavapai County, Ariz.

Justin Hiram Haynes, Denver, Colo.

Proposed by John V. N. Dorr, B. P. Morse, George E. Collins.

Born 1882, Telluride, Colo. 1903, Completed four-year course in metallurgy at Univ. of Denver. 1903-05, Millman, Tucker mill, Central City, Colo. 1905-08, Cyanide shiftman, Liberty Bell Gold Min. Co. 1908-10, Met., Hudson Reduction Co., Idaho Springs, Colo. 1910-12, Gen. Supt., Ophir Gold Mines Mill. & Power Co., Ophir, Colo.

Present position: 1912 to date, Mgr., Junta Cons. Gold Min. Co., Telluride, Colo.

Frederick Bradley Henderson, San Francisco, Cal.

Proposed by Anthony F. Lucas, E. W. Parker, David T. Day.

Born, 1863, Peoria, Ill. Grammar school only. 1879-1889, clerical, Wabash R. R. 1890-1902, Chief clerk to Genl. Mgr., A. T. & S. F. Ry., Los Angeles, Cal. 1902-05, Genl. Supt., Associated Oil Co., Bakersfield, Cal., in charge of operations. 1905-11, Genl. Mgr., Amalgamated Oil Co., Los Angeles, Cal.

Present position: 1911 to date, Asst. Genl. Mgr., Associated Oil Co., San Francisco, Cal.

Charles Nelson Hickok, Cleveland, Ohio.

Proposed by Lewis E. Dunham, Max McMurray, F. B. Richards.

Born 1879, Harrisburg, Pa. 1890-94, Harrisburg Academy. 1894-97, St. Paul's School, Concord, N. H. 1897-1900, Sheffield Scientific School, Yale Univ.; Ph. B. 1900-03, Latrobe Steel & Coupler Co., Chicago, Ill. 1903-05, Dayton Malleable Iron Co., Dayton, Ohio. 1905-06, Coal Dept., M. A. Hanna & Co. 1906-14, Mgr., Coal Dept., M. A. Hanna & Co.

Present position: 1913 to date, Manager, Coke Dept. and Ore Salesman, M. A. Hanna & Co.

Rukard Hurd, St. Paul, Minn.

Proposed by John Birkinbine, J. L. W. Birkinbine, George C. Davis.

Born 1858, Cincinnati, Ohio. 1878, Penn. Military College; C. E. 1912, Penn. Military College, Honorary Ph. D. Previously Quaker School, Chickering Insti. and Public Schools, all of Cincinnati, Ohio. I am continuing my study of mining engineering, geology, metallurgy and chemistry to assist me in an original research to result in a formulation of a standardization of iron-ore values. 1880-88, General Produce and River Transportation. 1889-1905, Insurance. 1901-02, Member of the Legislature, Minn. 1907, Sec'y. Minn. Tax Commission. 1900, Author of Hurd's Insurance Rate Book. 1911, Author of Hurd's Iron-ore Manual. New edition of same now in preparation for printing during the year.

Present position: Secretary, Minn. State Tax Commission.

Frederick Jaeger, Rahway, N. J.

Proposed by Clarence P. Linville, W. R. Crane, H. D. Pallister.

Born 1878, Baltic Provinces, Europe. 1903-05, Grad. Weymouth High School. 1905-09, Grad. Massachusetts Institute of Technology; S. B. in Mining and Metallurgy. 1909-10, Asst. Instructor in Metallurgy at Mass. Inst. of Tech. 1910, Chemist, American Smelting & Refining Co., Maurer, N. J. Experience in tank room, slimes department, nickel plant and in platinum, palladium recovery.

Present position: Chemist, American Smelting & Refining Co., Maurer, N. J.

Joseph Jeusen, San Francisco, Cal.

Proposed by F. Oskar Martin, Charles P. Berkey, J. F. Kemp.

Born 1886, Salt Lake City, Utah. 1900-04, Salt Lake High School. 1904-08, Utah State School of Mines; B. S. 1911-12, Royal School of Mines, Freiberg, Saxony. 1912-13, Post-graduate work, Columbia Univ., N. Y.; M. A. 1906, Asst. Engr., and Geol., Bamberger-DeLamar Gold Mines Co., DeLamar, Nev. 1908, Mining Editor, Salt Lake *Tribune*. 1908-10, Mineral Inspector, G. L. O. work in Utah, Idaho, and Wyo. 1912, visiting metal, coal and salt mines in Europe.

Present position: Mineral Inspector, U. S. General Land Office.

Estey A. Julian, Fairview, Nev.

Proposed by A. R. Parsons, E. E. Carpenter, H. P. Henderson.

Born 1883, Licking, Mo. 1900, Finished High School, Licking, Mo. 1904, William Jewel College, Liberty, Mo.; S. B. 1904, Mill, Dorcas Milling Co., Florence, Colo. 1904, Mill, U. S. Reduction & Refining Co., Florence, Colo. 1905, Mill, Woods Investment Co., Cripple Creek, Colo. Mine and Mill, B. B. B. Mine Co., Howbert, Colo. 1906, Miner, Jo Dandy Min. Co., Cripple Creek, Colo. 1906-07, Miner, Stratton's Independence, Cripple Creek, Colo. Miner, Prince Albert Mines Co., Cripple Creek, Colo. 1907, Mill, Tonopah-Belmont Dev. Co., Millers, Nev. 1908, Mine, Goldfield Cons. Mine Co., Goldfield, Nev. Mill, Nevada Goldfield Reduction Co., Goldfield, Nev. 1909, Mine and Mill, Homestake King Min. Co., Rhyolite, Nev. 1910, Mine and Mill, Picacho Basin Min. Co., Picacho Basin, Cal. 1911, Mill, Goldfield Con. Mines Co., Goldfield, Nev. 1912-13, Mine and Mill, Nevada Hills Min. Co., Fairview, Nev.

Present position: General Mgr., Nevada Hills Min. Co., Fairview, Nev.

Oscar Alfred Knerr, Layland, W. Va.

Proposed by Howard N. Eavenson, James S. Cunningham, G. S. Borden.

Born 1884, Lebanon, Pa. 1900-04, Albright College; B. S. 1904-05, Chainman, Phila. & Reading Coal & Iron Co., Pottsville, Pa. 1905-08, Asst. Engr. for G. S. Borden. 1908-12, Div. Engr., N. R. & P. C. Coal Co.

Present position: 1913 to date, Supt. of Mines, New River & Pocahontas Cons. Coal Co.

Oswald Mark Lewyn, Rancagua, Chile, S. A.

Proposed by J. F. Kemp, W. J. Turner, B. T. Colley.

Born 1886, New York City. 1904-08, Columbia School of Mines; E. M. 1908-09, Repairman in Mill of Steptoe Valley Mining, Milling & Smelting Co. 1909-10, Sampler, Chemist and Asst. Engr. Mines, Cumberland-Ely Copper Co. 1910-14, Asst. to Robert Marshur in examination mines, Braden Copper Co. Mine Foreman, Min. Engr., Asst. Supt., and Supt. Mines, Braden Copper Co.

Present position: Supt. of Mines, Braden Copper Co.

Elmer List, Neodisha, Kan.

Proposed by G. H. Cox, C. R. Forbes, H. A. Buehler.

Born 1884, Cape Girardeau, Mo. 1903, Elementary Teacher's Certificate, State Normal School, Cape Girardeau, Mo. 1910, Missouri School of Mines; B. S. 1909, Assayer, St. Louis Smelt. & Ref. Co., St. Francois, Mo. 1910, Chemical Inspector, Union Pacific R. R., Omaha, Neb. 1911, Chemist, St. Louis Smelt. & Ref. Co., Collinsville, Ill. 1913, Chemist, Buffalo Mines, Ltd., Cobalt, Ont., Canada.

Present position: Engr., Granby Min. & Smelt. Co., Neodisha, Kan.

Archibald R. Livingston, Canon City, Colo.

Proposed by George C. Stone, J. A. Van Mater, Lewis G. Rowand.

Born 1868, New York. 1891, School of Mines, Columbia; C. E. Lackawanna Steel Co. Various engineering, construction power plants, etc.

Present position: 1901 to date, Supt., in charge of Mill, of Empire Zinc Co., Canon City, Colo.

Richard O. Meents, Boulder, Colo.

Proposed by C. C. O'Harra, Welton J. Crook, Albert Smith Halley.

Born 1887, Germany. 1893-97, Common Schools, Germany. 1898-1904, Common Schools, Battle Creek, Ia. 1904-08, Finished High School, Battle Creek, Iowa. 1908-09, Iowa State College, Ames, Iowa. 1909-11, Colo. School of Mines, Golden, Colo. 1911-12, Mich. College of Mines, Houghton, Mich. 1913-14, South Dakota State School of Mines, Rapid City, S. D.; B. S. in Min. Engrg. Member, Mining and Geology Society of American Universities. 1911, summer, general underground experience in St. Lawrence Mine, Butte, Mont. 1912, Sept. and part of October, general underground experience in St. Lawrence Mine, Butte, Mont. 1912-13, Five months mine sampler at Berkeley mine, Butte, Mont. Amalgamated Copper Mining Co. Six months, Queen Min. Co. of Mexico.

Present position: Mine Operator, Sonora, Mexico.

John P. K. Miller, Scottdale, Pa.

Proposed by Samuel A. Taylor, William G. Wilkins, H. S. Harrop.

Born 1857, Martinsburg, W. Va. 1874, Finished High School. 1874-77, St. Lonoah College, Martinsburg, W. Va. 1878-80, Special instruction in R. R. Engineering, Masonry, Structural Design, General Geology, and Hydraulic Design. 1893-96, Sprague Correspondence School of Law, Detroit, Mich. 1880-82, Pittsburg & Western R. R. Co. Level and Transit on Prelim. and Locational Construction, Asst. to Prin. Asst. Ch. Engr., on Construction of tunnels, viaduct, etc. 1882-83, On construction P. McK. & Y. N. R. R. (P. & L. E. R. R.), Pgh. to Connellsville, Pa. Part 1883, Ran Gen. Engrg. Office. Had charge of Mining Engineering work of Powers Brown & Co. and Hamilton Co.'s Mines, Reynoldsville, Pa. 1884-95, Ch. Engr., McClure Coke Co.

Present position: 1895 to date, Ch. Engr., H. C. Frick Coke Co.

Lou Fuller Mitten, Wilkes-Barre, Pa.

Proposed by Frank A. Hill, Paul Sterling, Charles Enzian.

Born 1879, Herrick, Pa. 1886-96, Public Schools at West Pittston, Pa. 1896-99, Susquehanna Collegiate Institute, Towanda, Pa. 1900 to date, Vulcan Iron Works, Mach., Clerk, Asst. Supt., Sales Dept., Mining Dept., special work on steam and electric winders for coal and metal mines.

Present position: Sales Engineer.

William Franklin Newman, Sheffield, England.

Proposed by F. L. Cole, Thomas T. Read, George D. Barron.

Born 1866, Launceston, Tasmania. 1878-82, Anglo French College, London. 1883-84, Ecole Professionnelle, Vicennes, Paris. 1893, Chevalier Class, Order "Crown of Roumania." 1900, Gold Medal, Paris Exhibition, Coal and Coke from Asia Minor. 1901, Third Class Imperial Ottoman Order of "Medjidie." Languages, English, French, German, Italian, Russian, Roumanian. 1884-94, Coal in South Russia and Roumania. 1895-96, Organized, as Genl. Mgr., the Roumanian State Maritime Service. 1897-1900, Commercial Mgr., Coal Mines, Société d'Heraclee, Heraclee, Asia Minor and Paris. 1901-13, Representative for Armamento, Thos. Firth & Sons, Ltd., Sheffield, England. 1914, Genl. Mgr., British Metallurgical Syndicate for China, etc.

Present position: Genl. Mgr., Representation for British Manufacturers, Ltd., Peking, China.

Wadsworth W. Norton, Salt Lake City, Utah.

Proposed by R. C. Gemmell, W. H. Howard, C. W. Whitley.

Born 1867, Berlin, Conn. General education, Syracuse High School, Syracuse, N. Y. 1886-89, Technical education, Cornell Univ., Ithaca, N. Y., course in Chemistry and Metallurgy. 1889-91, Asst. in plant of the Chicago & Aurora Smelt. & Ref. Co., Aurora, Ill. 1891-98, Foreman, Chemist, Met., United Smelt. & Ref. Co., East Helena, Mont. 1898, Supt., U. S. & R. Co., Great Falls, Mont. 1898-1905, Asst. Supt. and Supt., American S. & R. Co., East Helena, Mont.

Present position: 1905 to date, Supt., Murray Plant, American Smelting & Refining Co., Murray, Utah.

Thomas S. O'Brien, Amador City, Cal.

Proposed by C. R. Downs, Webb Smith, William A. Thacher.

Born 1871, Hanibal, Mo. 1890, Common School. 1896, Metal-Mining Course, I. C. S. 1904, Surveying and Mapping Course, I. C. S. 1895, General mining, Leadville, Colo. 1895-97, Foreman, Sedalia Mine, Leadville, Colo. 1897-99, Foreman, Iron Silver Mining Co., Leadville, Colo. 1899-1903, Mgr., New Leadville G. M. & M. Co., Leadville, Colo. 1903-07, General field work, Nevada and Arizona. 1907-10, Prospecting, Arizona. 1910-12, Foreman and Supt., Frisco Gold Mines Co., Kingman, Ariz.

Present position: Original Amador Con. Mines Co., Amador, Cal.

Calvin N. Payne, Titusville, Pa.

Proposed by Anthony F. Lucas, E. W. Parker, David T. Day.

Born 1844, Irvington, Pa. General education. 1885-1910, Standard Oil Co. of N. J. Constructing, Managing and Vice Pres. of Buffalo Natural Gas Co., United Natural Gas Co.; Oil City Fuel Supply Co.; Hope Natural Gas Co.; Penn. Gas Co., and eight other distributing companies. 1899-1910, Vice Pres. and Mgr., Natural Transit Co., Buckeye Pipe Line Co., Eureka Pipe Line, Penn. Pipe Line Co., Indiana Pipe Line Co.

Present position: Operating oil and gas properties on own account.

Morton Edgar Pratt, Guayaquil, Ecuador, S. A.

Proposed by T. W. Mather, Rush M. Hess, J. W. Mercer.

Born 1878, Winnemucca, Nev. 1895-99, Nevada State Univ., Special in Mining Engineering. 1899 to date, Have followed metal mining in its different branches, exclusively. Four years Yukon Territory and Alaska. Five years Nevada, California, and Arizona. Three years Mexico (Northern States). Three years, Ecuador, S. A.

Present position: Supt., South American Development Co., of Guayaquil, Ecuador, and 15 Broad St., New York.

John Callum Reid, Lethbridge, Alberta, Canada.

Proposed by G. H. Cox, C. R. Forbes, H. A. Buehler.

Born 1871, Sharon, Pa. 1877-89, Public Schools of Pa., Missouri, and Indian Territory. 1889-93, Missouri School of Mines; M. E. 1895-1900, Min. Engr., Coal Dept., M. K. & T. Ry. Co., Parsons, Kan. 1900-02, Supt., Oklahoma Mines. M. K. & T. Ry. Co., Coalgate, Okla. 1902-06, Genl. Supt., Osage Coal & Min. Co. and Great Western Coal & Coke Co., McAlester, Okla. 1906-11, Genl. Mgr., Osage Coal & Min. Co. and Great Western Coal & Coke Co., McAlester, Okla.

Present position: 1911 to date, Genl. Mgr., Chinook Coal Co., Ltd., Sherlock Bldg., Lethbridge, Alberta, Canada.

William Birzey Richards, Lansford, Pa.

Proposed by Edwin Ludlow, W. G. Whildin, J. B. Warriner.

Born 1868, New Philadelphia, Pa. Lost Creek High School, Correspondence Schools. 1886-1891, Lehigh Valley Min. Corp., Lost Creek, Pa. 1891-1903, Min. Engr., Mill Creek Coal Co. 1903-05, Min. Engr., A. B. Cochran & Son, Pottsville. 1905-07, Supt., Pine Hill Colliery, Mineville, Pa.

Present position: 1907 to date, Min. Engr., Lehigh Coal & Nav. Co., Lansford, Pa.

Soren Ringlund, Silver City, N. M.

Proposed by Russell B. Paul, Robert Hursh, Edward W. Keith.

Born 1875, Denmark. 1912, School of Mines, Socorro, N. M.; B. S. and E. M. 1912, Mining, U. S. Mines, Bingham, Utah. 1912-13, Operating department at concentrator, Chino Copper Co., Hurley, N. M. 1913, Laboratory, El Paso Smelter, El Paso, Texas.

Present position: 1913 to date, Engr., Empire Zinc Co.

Beauregard Ross, Denver, Colo.

Proposed by G. H. Cox, C. R. Forbes, H. A. Buehler.

Born 1861, Marshfield, Mo. 1867-79, Public School and Academy, Houston, Mo. 1879-82, Mo. School of Mines, Rolla, Mo. 1887-98, Editor daily newspaper. 1898-1914, Engaged in developing mines in Colo., Nevada, New Mexico, except one year operated assay office in Cripple Creek, Colo. My system has been to own and operate my own properties under corporations which I controlled myself. Operated successful lease in Cripple Creek, Colo. Three years producing \$385,000. Built first cyanide plant in Cripple Creek.

Present position: Genl. Mgr., Colo. Idaho Mines and Development Co.

George W. Salisbury, Denver, Colo.

Proposed by J. C. Robert, John A. Davis, W. G. Swart.

Born 1888, Whitewater, Wis. 1894-1902, Grade Schools, Whitewater, Wis. 1902-06, Whitewater High School, Wis. 1906-11, Colo. School of Mines. 1910-11, North American Smelting Co. 1911-14, U. S. Bureau of Mines.

Present position: Safety Engr., United Verde Copper Co., Jerome, Ariz.

Richard Warren Senger, Port Jervis, N. Y.

Proposed by A. S. Dwight, H. J. Stehli, P. A. Mosman.

Born 1879, Port Jervis, N. Y. 1892-95, Port Jervis High School. 1901-05, Mass. Inst. of Tech., Mining and Metallurgical course; B. S. 1895-97, Chemist, Greene Cananea Copper Co., Cananea, Son., Mexico. 1897-1911, Chemist, American Smelting & Refining Co., El Paso and Aguascalientes.

Present position: 1911 to date, Asst. Supt., A. S. & R. Co., Agnascalientes.

Homer Kent Sherry, Mascot, Tenn.

Proposed by G. H. Cox, C. R. Forbes, H. A. Buehler.

Born 1888, Brownsville, Ind. 1895-1904, Common Schools of Osage County, Kan. 1904-08, High School, Eaton, Ind. 1908-12, Missouri School of Mines and Metallurgy; B. S. 1912-13, Machine helper, machine man and power man, American Lead, Zinc & Smelting Co., in mines at Cartersville, Mo. 1911, Summer, mines of Colo., most of time at Victor.

Present position: 1913 to date, American Zinc Co. of Tenn. at Mascot, as powder foreman and shift boss.

Irving T. Snyder, Denver, Colo.

Proposed by D. W. Brunton, Frank M. Taylor, Edward W. Keith.

Born 1884, Denver, Colo. 1905, Sheffield Scientific School, Yale Univ.; Ph. B. 1908, Took charge of the Western Alkali Mfg. Co. of Green River, Wyo., in the manufacture of caustic and sal soda; have continued with the company as Genl. Mgr. and later as Pres. 1912, became actively associated with Vindicator Cons. Gold Min. Co., with mines at Independence, Colo.

Present position: 1913 to date, Vice-Pres., Vindicator Min. Co.

Eugene L. Steindler, Cobalt, Ont., Canada.

Proposed by J. F. Kemp, Charles P. Berkey, Alfred J. Moses.

Born 1887, San Francisco, Cal. 1901, Grad. Public School, New York. 1901-03, DeWitt Clinton High School, transferred to High School of Commerce. 1908-12, School of Mines, Columbia University. 1903-08, In business, Block Light Co., 17 Park Place, New York City, as clerk, Advertising Mgr., and Asst. to Pres.

Present position: 1912 to date, Secretary-Treas., Dominion Reduction Co., Ltd.

Allen Pinckney Stramler, Mogollon, N. M.

Proposed by William B. Phillips, Anthony F. Lucas, David T. Day.

Born 1875, Stephenville, Texas. 1900-01, John Tarleton College, Stephenville, Texas. 1901-04, Student in Univ. of Texas, Austin, Texas; B. S. 1904-05, Post-graduate student in Chemistry and Mining Engineering. 1906, Six months miner and laborer, Victor, Colo. 1906, Assayer and Surveyor, Chisos Min. Co., Terlingua, Texas. 1907, Supt., Ario Copper Co., Ario de Rosales, Michoacan, Mexico. 1908, Supt., Tamaulipeca Min. Co., Victoria, Tamaulipas, Mexico. 1909-13, Employed in Cyanide depts. and Bullion rooms of various mills, Guanajuato, Pachuca, El Oro, Mexico.

Present position: Miner, Mogollon, N. M.

Karl Aksel Strand, Hanover, N. M.

Proposed by Robert Hursh, Russell B. Paul, Cony T. Brown.

Born 1885, Christiania, Norway. 1912, School of Mines, Socorro, N. M.; B. S. and E. M. 1912, Utah Copper Co.'s Mill, Garfield, Utah. 1913, Copper Queen Co., Bisbee, Ariz.

Present position: Engineer, the Empire Zinc Co., Hanover, N. M.

David Taylor, Salt Lake City, Utah.

Proposed by Ernest Gayford, E. R. Lidvall, L. S. Austin.

Born 1881, Lancaster, Pa. 1894-98, St. Paul's School. 1898-1902, Williams College; B. A. 1902-03, New York Law School. 1903-05, Clerk, Taylor & Brunton Sampling Co., Victor, Colo. 1905-10, Asst. Mgr., Taylor & Brunton Ore Sampling Co., Salt Lake City, Utah. 1910-11, Mgr., Taylor & Brunton Ore Sampling Co., Silver City, Utah.

Present position: 1912 to date, Chief Stockholder and Mgr., David Taylor & Co.

Curtis Christopher Webb, New York, N. Y.

Proposed by T. W. Mather, Rush M. Hess, J. W. Mercer.

Born 1888, Near Yreka, Cal. 1892-96, Country District School, Cal. 1896-98, Gymnasium, Dresden, Germany. 1898-1900, About fourteen different schools in So. Africa. 1900-01, Belmont School, Cal. 1901-02, Browne & Nichols School, Camb., Mass. 1902-06, Rindge Manual Training School, Cambridge, Mass. Obtained certificate of graduation. 1905-06, Summer, Harvard Engineering Camp, Squam Lake, N. H.; course in plane surveying. 1906-10, Mass. Inst. of Tech., Boston, Mass.; B. S. in Mining and Met. During my college course I had additional employment as follows: four months surveying and night mill man, Queen of Sheba mine, Siskiyou County, Cal. Two months examination of mine at Atlanta, Idaho. 1910-11, Nevada Cons., McGill, Nev. 1911-12, Anglo-Columbian Development Co. 1912, Two months, Examination on Trinity River near Lewiston, Cal. 1912-14, Canadian Klondyke Min. Co., Dawson, Yukon Territory.

Daniel B. Wentz, Philadelphia, Pa.

Proposed by Samuel D. Warriner, H. M. Chance, Eli T. Conner.

Born 1872, Jeddo, Pa. 1892, Phillips Academy, Andover, Mass. 1896, Warren Scientific School, Harvard Univ., Cambridge, Mass. 1896 to date, Virginia Coal & Iron Co., Big Stone Gap, Va. 1900 to date, Stonega Coke & Coal Co., Stonega, Va. 1909 to date, Midvalley Coal Co., Philadelphia, Pa. 1908 to date, Maryd Coal Co., Philadelphia, Pa. 1908 to date, J. S. Wentz & Co., Philadelphia, Pa. 1905 to date, Royal Colliery Co., Chicago, Ill.

Present position: Executive officer of above corporations.

George Harrison Wigton, Great Falls, Mont.

Proposed by John H. Klepinger, Milo W. Krejci, C. R. Kuzell.

Born 1888, Hastings, Neb. 1903-07, East Denver High School. 1913, Colo. School of Mines; E. M. 1910-11, Asst. Engr., Camp Bird Mine, Ouray, Colo.

Present position: Investigator, B. & M. Red. Dept., Anaconda Copper Mining Co.

Fred Willard Young, Winchester, Mass.

Proposed by William Kingman Page, E. A. Cappelen Smith, Charles H. White.

Born 1887, St. Stephen, N. B., Canada. Public Schools, Brookline. 1907-11, Harvard College; B. S. 1911-12, Grad., School of Applied Science, Harvard Univ.; M. E. 1913, Worked on machine drill, American Zinc & Lead Co., Mo. 1913, Worked in Butte on a machine; in the assay office, and in the mill of the Butte & Superior Copper Co. Engineer, Davis Daly Copper Co.

Present position: Unemployed.

Associate Members

James Barnett Ambler, Salt Lake City, Utah.

Proposed by W. A. Wilson, George W. Riter, Frank Anderson.

Born 1878, Philadelphia, Pa. 1901, Univ. of Penn.; B. S. 1901-14, Engineer with General Electric Co.

Present position: Sales Agent with General Electric Co.

Willis Whittier Case, Jr., Denver, Colo.

Proposed by Fred H. Bostwick, C. L. Colburn, Cony T. Brown.

Born 1881, Washington, D. C. Public Schools, Washington, D. C. Columbian Univ., Washington, D. C. Ohio State Univ., Ceramic Dept. New York State School of Ceramics, Alfred, N. Y. Minor under instruction, U. S. Naval Gun Factory, Washington, D. C., four years. Twelve years, all departments of Denver Fire Clay Co. Mfg. of full line of assayers' materials, and inventor of Case Met. and Assay Furnaces.

Present position: Pres. and Genl. Mgr., Denver Fire Clay Co.

Spencer Disston Wright, Philadelphia, Pa.

Proposed by F. Lynwood Garrison, R. H. Sanders, H. M. Chance.

Born 1860, Philadelphia, Pa. Public School education. 1905-14, Pres., Warrior Copper Co., Globe, Ariz.

Present position: 1912 to date, Vice-Pres., Penn.-Canadian, Cobalt, Canada.

Junior Members

Theodore H. Barrett, Forest Hills, N. Y.

Proposed by J. F. Kemp, Charles P. Berkey, E. L. Kurtz.

Born 1892, Brooklyn, N. Y. 1914, E. M., Columbia School of Mines.

Present position: Student at Columbia.

William Wallace Colquhoun, Tunbridge Wells, England.

Proposed by Arthur L. Walker, Robert Peele, William Campbell.

Born 1890, San Francisco, Cal. Until 1904, early education in California. Six years at Glenalmond School. Until 1910, Perthshire, Scotland. Four years in the School of Mines, Columbia Univ. 1914, Degree M. E. I have not filled a technical position, but have been with my father, James Colquhoun, former President of the Arizona Copper Co., on many of his technical trips and tours of inspection.

Present position: Just graduated.

Walter Victor Fox, Flushing, N. Y.

Proposed by J. F. Kemp, Robert Peele, E. L. Kurtz.

Born 1893, New York City. 1898-1906, Public School, Flushing, L. I. 1906-10, Flushing High School. 1910-14, School of Mines, Columbia Univ.

Present position: Student at School of Mines, Columbia Univ.

Walter Franklin Graham, Graham, Tazewell Co., Va.

Proposed by C. E. McQuigg, W. R. Crane, H. D. Pallister.

Born 1891, Philadelphia, Pa. 1900-08, Haverford School, Haverford, Pa. 1908-09, Haverford College, Haverford, Pa. 1910, Penn. State College, State College, Pa. 1909-10, Metallurgical Dept., Standard Roller Bearing Co., Philadelphia, Pa. Summers of 1912, 1913, same company.

Present position: Student of Metallurgy at the Penn. State College, State College, Pa.

Richard Palmer Grieve, Brooklyn, N. Y.

Proposed by Robert Peele, Charles P. Berkey, J. F. Kemp.

Born 1886, Haverhill, Mass. 1903-06, DeWitt Clinton High School, New York City. 1906-08, Holyoke High School, Holyoke, Mass. 1910-14, Columbia School of Mines, Columbia Univ.

Present position: Student, School of Mines, Columbia University.

John Falcomer Haddow, Lebanon, Pa.

Proposed by C. E. McQuigg, W. R. Crane, H. D. Pallister.

Born 1890, Covington, Ky. 1909, Lebanon, Pa., High School. 1907-08, Worked for Penn. Steel Co., during summer vacation.

Present position: Student, School of Mines, Penn. State College, State College, Pa.

Francis B. Johnston, Bessemer, Mich.

Proposed by C. E. McQuigg, W. R. Crane, H. D. Pallister.

Born 1890, Watersmeet, Mich. 1904-08, Bessemer High School. 1908-11, Virginia Military Institute, Lexington, Pa. 1912, Penn. State College, State College, Pa. 1911-12, Worked at the Homestead Plant of the Carnegie Steel Co.

Present position: Student of Metallurgy at the Penn. State College, State College, Pa.

Mack C. Lake, Madison, Wis.

Proposed by C. K. Leith, Edwin C. Holden, Richard S. McCaffery.

Born 1890, Brodhead, Wis. 1895-1908, Common and High School, Brodhead, Wis. 1908-09, Teacher in common school. 1909-14, U. W., Mining Engr. 1911, Summer, Geological work on Vermillion Iron Range, Madison, Wis. 1912, Summer, U. S. Geological Survey in Wyo. 1913, Exploration work for E. J. Longyear Co., Minneapolis, Minn., at Bauxite, Ark. Have been assisting C. K. Leith, Madison, Wis., for two years when at school on mine reports, estimates, etc.

Present position: Student at Univ. of Wis., also assisting C. K. Leith.

Frank L. Pittman, Golden, Colo.

Proposed by F. W. Traphagen, Russell B. Caples, Jr., F. F. Frick.

Born 1889, Argentine, Kan. Finishing course at Colorado School of Mines.

Present position: Student.

William Cristy Pryer, Rapid City, S. D.

Proposed by C. C. O'Harra, Welton J. Crook, A. S. Halley.

Born 1890, Falmouth, Mass. 1896-99, Common Schools, Chicago, Ill. 1899-04, Common Schools, Falmouth, Mass. 1904-08, High School, Aberdeen, S. D. 1908-09, Northern Normal and Industrial School, Aberdeen, S. D. 1909, South Dakota School of Mines. 1910-13, Summers, drafting and surveying, Aberdeen, and Rapid City, S. D.

Present position: Candidate for B. S. degree, S. D. School of Mines, and First Asst. in Chemistry. Member of the Geological and Mining Society of American Universities.

Paul W. Rishell, Montoursville, Pa.

Proposed by C. E. McQuigg, W. R. Crane, H. D. Pallister.

Born 1893, Ansonville, Pa. 1904-09, Williamsport Dickinson Seminary. 1910, Grad. Pennington Seminary secondary school.

Present position: Student, School of Mines, Penn. State College.

Janshi Tsu En Sen, New York, N. Y.

Proposed by Robert Peele, Arthur L. Walker, J. F. Kemp.

Born 1882, Ping Yuen, Kwei Chow, China. 1900-09, National Univ., Peking, China; B. S. 1910-14, School of Mines, Columbia Univ.; E. M. 1909, Instructor in Chemistry, Teachers' College, Kwei Chow, China.

Present position: Student in School of Mines, Columbia Univ., New York City.

Everett W. Smith, Washington, D. C.

Proposed by C. E. McQuigg, W. R. Crane, H. D. Pallister.

Born 1888, Steelton, Pa. 1904-08, Steelton, Pa., High School. 1908-09, Bethlehem Preparatory School. 1911, Penn. State College. Worked during summer vacations the last nine years for the Penn. Steel Co. in various departments, including Electrical, Coke Oven, Open Hearth, etc.

Present position: Student of Metallurgy at the Penn. State College.

Ralph Evans Wells, Jr., Los Angeles, Cal.

Proposed by Robert H. Richards, Charles E. Locke, Carle R. Hayward.

Born 1891, Topeka, Kan. 1898, Public School, Prescott, Ariz. 1901, Public School, Los Angeles, Cal. 1903-05, Public School, Salt Lake City, Utah. 1906-10, Polytechnic High School, Los Angeles, Cal. 1910-14, Mass. Inst. Technology, Boston, Mass. 1910-11, Summer, Surveying Party on Pacific Electric R. R., Los Angeles, Cal. 1912, Summer, Laborer, Arthur Plant, Utah Copper Co., Garfield, Utah.

Present position: Student, Mass. Inst. of Technology.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period May 10 to June 10, 1914. This list, together with the lists published in *Bulletin* Nos. 88 to 90, April to June, 1914, and the foregoing list of members, therefore, supplements the annual list of members corrected to Mar. 1, 1914, and brings it up to the date of June 10, 1914.

AUSTIN, E. A.	Care Yukon Gold Co., Iditarod, Alaska.
BARBOUR, PERCY E.	887 Middle Street, Bath, Me.
BATCHELLOR, STILLMAN	Littleton, N. H.
BECKWITH, H. T.	2415 Ninth St., Wichita Falls, Texas.
BETTS, ANSON G.	Stackhouse, N. C.
BIRKINBINE, J. L. W.	Parkway Bldg., Broad and Cherry Sts., Philadelphia, Pa.
BLOOD, GEORGE D.	1021 Kearns Bldg., Salt Lake City, Utah.
BOUTWELL, JOHN M.	Howard-Canfield Bldg., Santa Barbara, Cal.
BRADT, HARLAN H.	5732 Kentucky Ave., Pittsburg, Pa.
BURNS, E. Z.	25 Madison Ave., New York, N. Y.
BUSH, M. W.	American Trust Bldg., Birmingham, Ala.
BUTLER, M. C.	Care Yukon Gold Co., Dawson, Y. T., Canada.
CHASE, F. D.	Dedham Ave., Dedham, Mass.
CLEMENT, HARRISON E.	University Club, Salt Lake City, Utah.
CLERC, F. L.	Estes Park, Colo.
COLE, ROBERT J.	Care Queen of the West Mines Co., Cornucopia, Ore.
COOPER, J. H.	Skidoo Mines Co., Skidoo, Cal.
COUSIN, R.	Rukuba Tin Mines, Ltd., Care Naraquta Post Office, Northern Nigeria, W. Africa.
COXE, EDWARD H.	P. O. Box 444, La Follette, Tenn.
CREMER, FELIX	Needles, Cal.
DICKMAN, R. N.	P. O. Box B, Wequetonsing, Mich.
DRAPER, CARL H.	112 Bloomfield St., Dorchester, Mass.
EMERY, A. B.	Care The Engineers' Club, 32 W. 40th St., New York, N. Y.
EPFLEY, MARION	Princeton, N. J.
FAHRENWALD, F. A.	2170 Stearns Road, Cleveland, Ohio.
FOSS, THEODORE	Manejni pereoulck, 16, log. 38, St. Petersburg, Russia.
FYFE, ALEXANDER	Care Institution of Mining and Metallurgy, 1 Finsbury Circus, London, E. C., England.
GIBBS, GEORGE H.	83 Linwood Road, Handsworth, Birmingham, England.
GLOVER, G. H.	Short Hills, N. J.
GRAY, E. F.	Kimberly, Nev.
GREENE, LEWIS H.	Hotel Carlton, Berkeley, Cal.
GREENEFELDER, NELSON S.	Care Tacoma Smelting Co., Tacoma, Wash.
GROSS, JOHN	Mines Alotepeque, Concepcion, Depto. Chiquimula, Guatemala, C. A.
HAGAR, EDWARD H.	1514 New Continental and Commercial Bank Bldg., 208 South La Salle St., Chicago, Ill.
HAMILTON, HARRY T.	Moctezuma Copper Co., Nacozari, Son., Mexico.
HICKS, CHARLES M.	Marengo, Ill.
HILL, WALTER HOVEY	Cambridge, Idaho.
HIRSHBERG, FRANK J.	Lock Box P., Choteau, Mont.
HORCASTAB, A. S.	66 Madison Ave., New York, N. Y.
HULICK, WILLIAM H.	11 Broadway, New York, N. Y.
HUTCHINS, J. P.	441 Salisbury House, London, E. C., England.
JACKSON, F. H.	Care Mines Co. of America, 714 Mills Bldg., El Paso, Texas.
JAQUES, W. H.	Little Boar's Head, N. H.
JARVIS, ROYAL P.	Instructed to hold all mail.
JAYNE, A. R.	127½-5-N., Seattle, Wash.
JENNINGS, HENNER	Care Mariposa Commercial and Mining Co., 609 Insurance Exchange Bldg., San Francisco, Cal.
KAFFER, STEPHEN L.	Care Burro Mountain Copper Co., Tyrone, N. M.
KEMP, JAMES T.	211 West 139th St., New York, N. Y.
KIRKCALDY, N. M.	Shire of Walpeup-Ouyen, Victoria, Australia.
KLESATTEL, RICHARD	808 American Bank Bldg., Seattle, Wash.
KNUTZEN, THEODOR	112 N. Plum Street, Richmond, Va.

KOELLE, CARL	Instructed to hold all mail
KRAUSE, WALTER B.	421 Chestnut St., Lebanon, Pa.
LADOO, RAYMOND B.	7 Oak Avenue, Leominster, Mass.
LAURIE, FRANK C.	Care T. J. Ryder, Cia. Mex. de Petroles "El Aguilá" S. A. Ave. Independencia 15, Vera Cruz, Mex.
LEE, GEORGE B.	Brentwood, Long Island, N. Y.
LEWIS, ROBERT S.	270 South 13th East St., Salt Lake City, Utah.
LINCOLN, FRANCIS CHURCH	264 Boylston St., Boston, Mass.
LUCKE, P. K.	Greenwood, Kirn, Scotland.
MCCLINTOCK, A.	Hermon, St. Lawrence Co., N. Y.
MACFARLANE, RIENZI W.	Care J. E. Robertson & Co., 509 Mills Bldg., El Paso, Texas.
MADSON, FRANK H.	General Delivery, Bessemer, Mich.
MAGEE, JAMES F.	Detroit Copper Mining Co., Morenci, Ariz.
VON MALITZ, EDMUND	Hess Steel Castings, Ltd., Bridgeton, N. J.
MANN, WILLIAM SEWARD	Cisco, Cal.
MARSHALL, GEORGE B.	442 E. 136th St., New York, N. Y.
MARSHALL, S. M.	408 Tioga St., Westmont, Johnstown, Pa.
MOHRMAN, EDWIN M.	1293 West 111th St., Cleveland, Ohio.
MORRIS, H. G.	R. F. D. 1, Fillmore, Cal.
MORRISON, W. L.	Welland, Ont., Canada.
MOSER, GEORGE F.	Martell, Amador Co., Cal.
MURPHY, E. M.	5 Concord Ave., Belmont, Mass.
MYERS, J. W.	P. O. Box 345, Golden, Colo.
NICHOLAS, ASKIN M.	Block 14, Torrington, Bismuth, via Deepwater, N. S. W. Australia.
PAGE, WILLIAM KINGMAN	Room 1711, 165 Broadway, New York, N. Y.
PAINE, F. W.	82 Devonshire St., Boston, Mass.
PALMER, C. H., JR.	Pacific Mines Corporation, Ludlow, Cal. (Stagg P. O.)
PARKER, R. A.	802 Equitable Bldg., Denver, Colo.
PERRY, W. A.	2nd Ave. and 71st St., Brooklyn, N. Y.
PORTER, JESSE C.	Care R. A. Macfie, Calle Tetuan 7, San Juan, Porto Rico.
POWELL, F.	5025 12th Ave., N. E., Seattle, Wash.
PROCHAZKA, GEORGE A., JR.	Banff, Alberta, Canada.
PYNE, FRANCIS R.	29 Scotland Road, Elizabeth, N. J.
REMICK, WALTER L.	Eagle River Mine, Eagle River, Alaska.
RICKARD, EDGAR	724 Salisbury House, London, E. C., England.
RICKARD, T. A.	724 Salisbury House, London, E. C., England.
RIES, H.	19 Whitehall Place, E. Orange, N. J.
RODGERS, GEORGE B.	St. Louis Smelting & Refining Co., St. Francois, Mo.
SEAMAN, HENRY B.	233 Broadway, New York, N. Y.
SMITH, H. DEWITT	Kennecott, Alaska.
SPROAT, A. D.	Care American Consul, Vera Cruz, Mexico.
STANTON, ROBERT B.	700 West End Avenue, New York, N. Y.
STARR, CHARLES C.	1204 Ridge St., Reno, Nev.
STUCKEY, L. C.	2 Elm Bank Gardens, Barnes, London, S. W., England.
TACHMANN, HENRY	Bunker Hill & Sullivan M. & C. Co., Kellogg, Idaho.
TIBBY, BENJAMIN F.	178 "E" Street, Salt Lake City, Utah.
TRISCHKA, CARL	Care F. M. Lay, Lavernia, Texas.
VALLAT, BENJAMIN W.	347 E. Grand Boulevard, Detroit, Mich.
VIGEON, E. C., Met.	27 Caithness Road, Mitcham, Surrey, England.
WARRINER, R. C.	Box 102, Crown Mines, Johannesburg, Transvaal, S. Africa.
WHERRY, H. P.	323 Broadway, Cincinnati, Ohio.
WILLIAMS, FRED T.	1361 Stratford Ave., Salt Lake City, Utah.
WILSON, GORDON	Instructed to hold all mail
WORTH, JOHN G.	Box 295, Santa Fe, N. M.
YOUNG, JAMES S.	Care Milne & Co., Lima, Peru, S. A.
YOUNG, WILLIAM A.	Kendall, Mont.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name.	Last Address of Record, from which Mail has been Returned.
BOLLES, J. H.	114 W. 79th St., New York, N. Y.
BRYCE, R. A.	Ross M. M. Co., Silverton, Colo.
CHAMBERLAIN, JOHN R.	El Aguilá Oil Co., Apartado 150, Tampico, Tamps., Mexico.

DISBROW, W. F.....	1108 Foster Bldg., Denver, Colo.
ELLIOT, JOHN L.....	54 New Bond St., London, E. C., England.
HELDT, HANS L.....	Sombrerete, Zac., Mexico.
HOFFMANN, A. O.....	Handtverkargatan 15, Stockholm, Sweden.
JEFFREY, ROBERT H.....	Gabriel, Dur., Mexico.
LEAVELL, JOHN H.....	Newhouse Bldg., Salt Lake City, Utah.
PEARSE, EDWIN A.....	Raymond, Madera Co., Cal.
POLKINHORN, E. J.....	Apartado Postal No. 12, Terreon, Coahuila, Mexico.
RAWLINGS, STUART L.....	Durango, Mexico.
SCOBAY, J. C.....	3424 Colfax Ave. B., Denver, Colo.
SIMINGTON, FRANCIS J.....	6444 Stoney Island, Jackson Park Sta., Chicago, Ill.
TEEL, WILLIAM H.....	205 19th Ave. N., Seattle, Wash.
WHEELWRIGHT, O. W.....	Madison, Wis.
YOUNG, C. D.....	Tower City, N. D.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period May 10 to June 10, 1914.

Date of Election.	Name.	Date of Decease
1912	*Barlow, A. E.....	May 28, 1914.
1902	*Harris, W. F.....	Nov. 4, 1913.
1904	*Weniger, Rudolf.....	

*Member.

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SAMUEL W. BEYER,	R. R. GOODRICH,
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NOAH F. DRAKE,	WALTER S. MORLEY,
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EDWARD H. BENJAMIN,
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WALTER DOUGLAS,PHILIP N. MOORE,
ROBERT H. RICHARDS,
MILNOR ROBERTS,
CARL SCHOLZ,
WILLIAM WRAITH,

THOMAS T. READ, <i>Secretary</i> , Woolworth Bldg., New York, N. Y.	
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RICHARD M. ATWATER, Jr.,	EDWIN C. HOLDEN,
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FREDERICK BRADSHAW,	ROYAL P. JARVIS,
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BRADLEY STOUGHTON.¹ Until Feb., 1915.² Until Feb., 1916³ Until Feb., 1917.⁴ Until Feb., 1918.

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CHARLES KIRCHHOFF,²

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Biographical Notice of Louis Janin

BY R. W. RAYMOND, NEW YORK, N. Y.

(Salt Lake Meeting, August, 1914)

THE death of Louis Janin, which took place Mar. 6, 1914, at Santa Barbara, terminated the earthly career of almost the last survivor, and



LOUIS JANIN

one of the most illustrious members, of that group of American mining engineers and metallurgists who, in the last quarter of the nineteenth century, directed the development of the mineral resources of the Pacific slope.

Louis Janin was born in 1836 at New Orleans. His grandfather, an officer in the French army, had emigrated to this country, with his family, at or before the outbreak of the French revolution. His father became an

eminent member of the New Orleans bar. He was the eldest of six sons, two of whom (Henry and Alexis) became, like him, mining engineers and metallurgists.¹

Of the other three, one, as I am informed, entered the Confederate army during the War for the Union, and died in battle; another, following his father's profession, achieved distinction as a lawyer in Washington, D. C. The third, Albert, the only survivor, is said to be now the manager of the Mammoth Cave property in Kentucky.

As I have explained more fully in my Biographical Notice of his brother, Henry Janin, their father was professionally engaged in the litigation (preceding the enactment of any U. S. laws governing mining titles on the Pacific slope) over the New Almaden quicksilver mines of California, and had thus become acquainted, on better authority than that of general rumor, with the immense immediate and prospective mineral wealth of the American West. No doubt he foresaw the opportunities of profit and honor which the development of this wealth would offer to trained experts; and this explains, in my judgment, the sudden change in his plans for the eldest two of his sons, Louis and Henry (only two years apart in age), who, in 1856, after completing their Sophomore year at Yale University, sailed for Europe, and entered in October, 1857, the Mining Academy of Freiberg, Saxony, where they studied for three years. That was, perhaps, the period of Freiberg's greatest glory—not because the Academy had not had before, or has not had since, good reason to boast of great leaders and teachers; but because the fame of that group of men, as scientific investigators, technical authors, skillful and thorough instructors, and sympathetic, influential critics and guides of practice, was world-wide and well-nigh unrivaled. The mere enumeration of the names of Weisbach, Breithaupt, Cotta, Gaetschmann, Scheerer and Fritsche—not to specify others of that splendid corps, who worthily took their places afterwards—is enough to explain the pre-eminence which Freiberg then enjoyed.

The Janins utilized thoroughly the benefits of such instruction, becoming in particular intimate associates of the genial and brilliant Cotta, with whom, after finishing their course at Freiberg, they took a long professional trip through Bohemia and Hungary, of which Cotta subsequently published a highly interesting account. It was when they returned from this journey for a farewell visit at Freiberg, in October, 1860, that I formed with them the acquaintance which became a life-long friendship.

After a brief supplementary course at the Paris École des Mines, the two brothers sailed for America in 1861. As I have said already, one of their younger brothers entered the Confederate army. But, so far as I can now recall, neither of them passionately took part in the conflict

¹ See my obituary notice of Henry Janin, *Bulletin* No. 53, May, 1911, p. xxviii.

of opinion which preceded the conflict of arms. No doubt they sympathized more or less with the South; but they had been abroad during the years of exciting controversy which had developed State pride in the South from a traditional sentiment into a fanatical, belligerent religion. Moreover, they were not particularly interested in the State of Mississippi, or the cultivation of cotton and sugar, or the institution of slavery, intimately connected therewith. Their thoughts had long been turned to the new, wide, free region further West, for the service of which they had been preparing themselves by years of study and travel. In fact, they were already enlisted for that war which went on *pari passu* with the War for the Union, and was waged by an army of prospectors and miners, for the physical conquest of a new Empire, while other hosts were battling to determine its political destiny. It is not surprising, therefore, that the two brothers soon found their way to the Pacific slope, and into active practice as mining engineers. One of Louis Janin's earliest experiences was an encounter with Apache Indians, who attacked in Arizona the exploring party of Col. Butterworth, of which Janin was a member. The story is told by J. Ross Browne in *Harpers' Magazine*, vol. xxx (1864-65), p. 287 ff., and in a book by the same author, entitled *Adventures in the Apache Country*. It shows young Janin to have behaved with cool and intelligent courage.

After a brief engagement as Superintendent of the Enriquita quick-silver mine in the Coast Range of California—a position in which his brother Henry was first his assistant and afterward his successor—Louis seems to have turned his attention to the mining and metallurgy of silver ores, for which his studies abroad had specially prepared him. The pressing problems which called at that time for the aid of trained experts were centered around the Comstock lode, the rich bonanzas of which were extracted with difficulty and reduced at enormous loss. Such superintendents as Deidesheimer invented methods of mining suited to local conditions; but the treatment of the peculiar auriferous silver ores of the Comstock mines required long and patient experiment, the history of which has been well told by A. D. Hodges, Jr., in his paper, "Amalgamation at the Comstock Lode, Nevada," etc. (*Trans.*, xix, 1890, p. 195), and in an article in the *Mining and Scientific Press* of San Francisco, May 21, 1910.

I shall do no more than briefly indicate the nature and value of the services of the Janins, and particularly of Louis Janin, to the metallurgy of the Comstock and other Pacific districts, not only by their laborious investigations and ingenious inventions, but also by their unremitting attention to details and discipline in practice, which diminished the leakages and losses, even of imperfect processes.

Already in 1863, according to Mr. Hodges, Louis Janin was metallurgist of the "Mexican" mill, at Empire City, Nev., where the auriferous

silver sulphides of the Mexican mine on the Comstock lode were treated. Some three years before, the Washoe process of pan-amalgamation had been introduced by Almarin B. Paul, a mill manager of genius and experience, who combined in this method the principles of the Mexican patio process with the use of pans, such as he had used in the gold mills of California. Operated with regard chiefly to mechanical capacity for speed, endurance, and large production, the Washoe process showed magnificent mills and machinery, and yielded large dividends of profit from the simple sulphides of the richest ores. But a large proportion, even of this material, was lost as "slimes," being ground to crystalline powder, the form and fineness of which caused it to float away, escaping the pan-amalgamation. To some extent, this loss of slimes was due to a chemical cause, namely, the presence, even in the rich ores, of multiple sulphides, which were not easily and quickly decomposed under the rude and rapid pan-grinding so as to yield amalgamable metal. An additional, and perhaps, in the aggregate, a larger loss was incurred in the sands or tailings from leaner ores, the valuable portion of which, enveloped in or adherent to particles of siliceous gangue, did not float, like the slimes, but, escaping the pan-amalgamation, whether for mechanical or for chemical reasons, flowed away into the tail-race. And finally, the tail-race received also many particles of amalgam already formed, but not aggregated so as to be successfully collected in the ordinary settling apparatus. These various sources of loss were not unknown, or totally disregarded; but the abundance of profits and the pressure for dividends required the exhaustive efforts of mine and mill managers, and little was done to determine or remedy current leakages, beyond the impounding of tailings and (so far as possible) of slimes, in yards and reservoirs. One further cause of this neglect was the lack of complete frankness and co-operation between the mines and the mills. Even when the same company owned both mine and mill, the latter might receive also "custom" ore for treatment; and it was not regarded as good policy to let customers know exactly what the mill extracted. The delivery of a certain percentage of the sample-assay value (and a pretty low percentage, too, it used to be!) settled the transaction. Of course, when the mills afterwards became the property of a separate set of owners, the betrayal of their economic efficiency was still less desirable. And what is not to be told to certain other persons had better not be known to anybody. At all events, Mr. Hodges says that in his active experience in Nevada, he knew of only two silver mills in that State, treating silver ores without roasting, which checked their own work so as to know exactly what they were doing.

This was the situation in 1863, when Janin was at the Mexican mill, where (upon his initiative, as I naturally presume) the "Freiberg" system of amalgamation in revolving barrels, instead of pans, was tried, first

upon raw tailings, with only partial success, and afterward upon dried and roasted tailings, with better results. The latter practice became known afterward as the Freiberg process, and was widely practised, for a while, on the Pacific slope in the treatment of "refractory" ores (*i.e.*, multiple sulphides, antimonides, arsenides, etc.); the trouble of drying the pulp being avoided by dry crushing in the mill. But the expense of roasting, added to the extra cost of dry crushing, proved fatal in the end to the Freiberg process; and, though it died hard, it died. Janin realized its economical weakness; and when, in 1864, the technical ability he had shown at the Mexican mill led to his appointment as Superintendent of the Gould & Curry Co., he gave his attention at once to the devising of a practicable humid process which would secure the chemical effects of roasting without involving that fatally expensive feature.

With what indomitable courage, patience and skill he prosecuted this inquiry, first for the Gould & Curry Co., and afterward, with greater freedom and daring, in a mill of his own, with no Board of Directors to prohibit experiments and demand dividends; how he attained technical success, only to see his commercial reward swept away by a flood which carried off the richer half of the 100,000 tons of tailings and slimes in which he had invested his capital; how he bravely overcame this disaster, and established, with his brothers, Henry and Alexis, a large and profitable business in the treatment of tailings and slimes by a modified Washoe process, without roasting, and by the intelligent use of well-known chemical reagents in the pan; how this process, not being patentable (since mere genius and industry and patience, however unusual, are not legally new), was copied by others, at least so far as to make mining companies refuse to sell their tailings and slimes, so that the Janins were at last driven out of business—all this is told in detail by Mr. Hodges, in the Institute paper already cited. And I might add to his story an additional chapter, telling how the establishment of railroad connections and the establishment of great smelting plants, where varied materials from many regions could be advantageously collected and mixed for economical reduction, has revolutionized many metallurgical situations of 50 years ago, and, in many cases, solved by simply erasing the problems over which the pioneers of American metallurgy spent their lives and broke their hearts.² But no such subsequent obliteration of immediate results should be permitted to obscure our grateful recognition of the work of those men who were the leaders of their generation.

I believe that, after leaving Nevada, Louis Janin was for some years manager of mines in Mexico; and I know that, somewhere in the '70's, he was employed for a year by the Japanese government, to make plans concerning the operation of certain gold and silver mines in Japan.

² See, in this connection, my Biographical Notice of Henry Janin, *Bulletin* No. 53, May, 1911, p. xxviii.

That was the period of Japanese progress when the two parties were so evenly balanced that the party of progress was strong enough to inaugurate new enterprises, and its opponents were strong enough to hinder their prosecution. I have often thought that this circumstance explains the erroneous notion, entertained in those days by many of us, that the Japanese were shallow and fickle. I remember Janin's humorous account of his experience among them—with what overwhelming courtesy, amounting to reverence, he was treated; how promptly his handsome salary was paid in gold; how freely all facilities for his work were provided; with what effusive gratitude and compliment his report and plans were received—and how absolutely nothing happened afterward, until, ashamed of drawing pay while doing nothing, he resigned and came home. Other American engineers had, in those days, a similar experience.

After his return from Japan, Mr. Janin enjoyed for many years an extensive practice as consulting engineer and expert, especially in connection with mining lawsuits involving questions of economic geology. Being not only a practitioner of long and varied training, but also a constant student and critic of technical literature, and characterized by imperturbable good nature and self-control, he was one of those expert witnesses whom wise cross-examiners handle with courtesy and caution. As an adviser, his clients found him upright, loyal and vigilant. His versatility, wit and comprehensive culture made him a fascinating companion in the old Union (later the Pacific-Union) Club of San Francisco, which was for many years his business address; and his generous helpfulness to younger mining engineers made a host of them his grateful debtors.

A little more than 20 years ago, Mr. Janin established his residence on a ranch at Gaviota, near the Mission of Santa Ynez, in southern California, where he lived with his family, except when temporarily called away for professional work. I never visited him there; but I had been repeatedly a guest in his Nevada home; and I could appreciate the desolation which fell upon him with the death of his wife, some years ago. Other afflictions he was called, in later days, to bear—including partial blindness and physical disability of other kinds, which must have made life a burden to him. Yet, if I may judge from his occasional letters to me, he carried that burden bravely and patiently to the last, crowning an honorable and useful life with a serene old age and a victorious death.

Louis Janin joined the Institute in 1872. He died at Santa Barbara, Cal., Mar. 6, 1914, of heart disease, after an illness of about ten days.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Losses of Zinc in Mining, Milling and Smelting

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(Salt Lake Meeting, August, 1914)

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INTRODUCTION

A GREAT DEAL of attention has recently been given to the metallurgy of zinc, and much of that which has been said and written on this subject has been in the nature of a criticism. For instance, we are quite accustomed to hearing it said that the metallurgy of zinc has stood still, that it is antiquated, out of date, etc. We are, however, inclined to agree with George C. Stone¹ when he says that, although there have been no such spectacular and radical changes as were worked in the metallurgy of copper and iron by the introduction of converters, there has nevertheless been a decided and marked progress in the metallurgy of zinc, and especially during recent years. However, it must be admitted that, due to the fact that the chemistry of zinc is so different from that of the other metals, the retort process is the only process by which zinc is produced commercially; and, moreover, that losses which occur in the zinc industry—that is, in the mining, milling, and smelting of zinc ore—are very great, for it is stated² that probably less than 50 per cent. of the zinc mined, even

¹ The Metallurgy of Zinc. A Discussion. *Bulletin* No. 85, Jan., 1914, p. 163.

² Parsons, C. L.: Notes on Mineral Wastes. *Bulletin*. No. 47, U. S. Bureau of Mines (1912).

TABLE I.—*Production of Spelter, 1912*

Source	Short Tons Spelter	Zinc "Ore"			Zinc-Lead Ore			All Other Ores		Total	
		Crude Ore	Recover- able Zinc Content	Per cent.	Crude Ore	Recover- able Zinc Content	Per cent.	Crude Ore	Recover- able Zinc Content	Crude Ore	Recover- able Zinc Content
Arizona.....	4,092	22,321	3,828	17.2	4,190	551	13.2			26,511	4,379
Arkansas.....	604	32,755	748	2.3						32,755	748
California.....	1,672	6,472	2,160	33.4						6,472	2,160
Colorado.....	60,841	177,946	42,495	23.9	229,107	21,339	9.3	2,169,573	2,277	2,576,626	66,111
Idaho.....	6,800	1,236	309	25.0	480,126	6,644	1.4			481,362	6,953
Illinois.....	3,952	147,859	4,065	2.6						147,859	4,065
Kansas.....	5,668	744,254	10,633	1.4						744,254	10,633
Kentucky.....	394	12,000	491	4.1						12,000	491
Missouri.....	149,557	9,321,145	136,551	1.5						9,321,145	136,551
Montana.....	14,196	136,552	13,431	9.8	91	28	30.8			136,643	13,459
Nevada.....	6,132	7,394	2,108	28.5	20,054	4,553	22.7			27,448	22,162
New Jersey.....	16,941	459,585	69,755	15.2						459,585	69,755
New Mexico.....	6,882	16,394	4,012	24.5	15,792	2,771	17.5			32,186	6,783
Oklahoma.....	2,041	293,500	5,769	2.0						293,500	5,769
Tennessee.....	1,935	32,347	2,191	6.8						32,347	2,191
Texas.....	245	351	119	33.9						351	119
Utah.....	7,756	9,422	2,800	29.7	216,764	5,729	2.2			226,186	8,529
Virginia.....	62	2,132	236	11.1	2,803	13	0.5			4,935	249
Wisconsin.....	34,137	1,363,636	33,050	2.4						1,363,636	33,050
	323,907										

in those mines where zinc is the only product, reaches the form of spelter. Such being the case, the question naturally arises, why do we have this loss, and is there any one branch of the industry which is responsible for more than its share of this loss? It is for this reason that we have brought up this subject at this time, and in a brief manner we have endeavored to show where and why these losses occur and why it is that so much attention, and properly so, is at present being given to the matter of developing new processes which will be suited to the treatment of zinc ores.

THE PRESENT SOURCES OF ZINC

The zinc-reduction plants of the United States produced, from domestic ores, a total of 323,907 short tons of spelter in 1912. The ores originated in 19 States, and in tonnage ranged from 62 tons in Virginia to 149,557 tons in Missouri.

Table I shows the production of primary spelter in short tons, also the source of mine production and kinds of ore, in short tons, in 1912:

All these 19 States produced zinc ore and 8 States both zinc ore and lead-zinc ore.

The variety of these ores is almost limitless. They vary in zinc content from 33.9 per cent. in Texas to 1.5 per cent. in Missouri; from a clean-cut sphalerite in a limestone-chert gangue free from any appreciable quantity of iron-lead sulphides to an admixture of zinc carbonate and silicate, lead, copper, and iron minerals.

Table II, compiled from statistics for 1911, shows, for each Western zinc-producing State, wherever possible, the number of zinc-producing mines, the tons of crude ore produced, and the output of gold, silver, copper, lead, and zinc.

The zinc-producing ores of the United States are of many kinds. Of the total mine production of zinc in 1912, a quantity corresponding to 88.4 per cent. was derived from zinc ore, 11 per cent. was derived from lead-zinc ore, and 0.6 per cent. was recovered from all other ores.

Zinc Ores

Excepting Montana and Arizona zinc ores, nearly all Western zinc ores produced are shipped direct to the zinc smelters. During 1912 some ore was shipped from Leadville carrying as low as 20 per cent. zinc. The zinc ore shipped from other sections of the West carried well above 25 per cent. zinc. As a general proposition there is but a small profit to the Western shipper of zinc ores which contain less than 30 per cent. zinc. Sulphide and carbonate zinc ores are shipped direct to the zinc smelters, the carbonate ores forming the larger portion. The zinc ores of Butte average approximately 21.7 per cent. zinc^a with a saving of 80 per cent.

^a Annual Report, Butte & Superior Co., for the year ending Dec. 31, 1912.

TABLE II.—Zinc-Producing Mines in Western States, 1911

Producers			Tons of Ore Sold or Treated	Mine Production of Metals				
Zinc Ore	Lead- Zinc Ore	Siliceous Ores		Gold, Ounces	Silver, Ounces	Copper, Pounds	Lead, Pounds	Zinc, Pounds
Arizona.....	4		16,106	1,119.01	40,061	1,676	211,046	3,593,974
Arizona.....	4		10,756	1,633.33	44,783	19,329	2,223,351	960,067
Arizona.....		170	243,900	104,011.72	414,981	730,140	8,913	8,913
California.....	2		4,625					2,807,035
Colorado.....	no record		110,845	164.54	55,969	11,883	407,007	51,388,636
Colorado.....			214,385	10,215.35	476,049	160,835	23,687,597	38,224,372
Colorado.....			1,874,103	866,252.76	5,096,026	5,614,076	18,724,095	4,994,219
Idaho.....	2		51,050	7.74	33,669	6,192	1,027,140	6,622,185
Idaho.....	7		*397,354	446.01	510,667	111,408	31,098,194	1,718,064
Montana.....	2		225,586	817.1 ^a	491,161	1,340,427	846,680	43,734,150
Montana.....	1		128				17,446	75,995
Nevada.....	8		3,013	0.29	9,216	764	180,910	1,999,898
Nevada.....	3		4,654	14.89	32,870		1,151,750	1,598,134
New Mexico.....			8,402					2,204,029
New Mexico.....			16,333	14.00	35,870	66,898	1,557,093	8,033,147
Utah.....	6		5,753	1.60	491	478	9,601	1,165,064
Utah.....	14		280,817	4,959.31	1,358,311	1,115,770	35,100,763	16,675,197
Washington.....	1		51	9.00	1,602	411	11,982	20,590

^a 0.2151 per cent. zinc.

made in concentration. The Franklin Furnace region of New Jersey is the principal zinc-producing region of the Eastern part of United States. This ore is an intimate mixture of three zinc minerals, franklinite, zincite and willemite. Virginia and Tennessee produce zinc carbonate and sulphide. The zinc recovered from the ores of the Eastern States ranges from 108.4 lb. in Virginia to 303.5 lb. in New Jersey.

For the most part the zinc ore treated in the Central States consists of sulphide ore. The metal recovered from each ton of ore in the concentrates ranges from 38 lb. in Kansas to 72 lb. in northern Illinois.

Lead-Zinc Ores

A variety of ores is included under this heading. They range from "dry" ores high in silver with low lead and zinc values and ores with gradually increasing lead and zinc values to the complex micro-crystalline ores found in a number of camps throughout the West, a good example being those of Rico, Colo. There are some ores in the West of this nature that are shipped either to lead or zinc smelters, depending on the gold and silver values. If the percentage of zinc is sufficiently high, the shipper sacrifices the gold and silver values, but on the other hand, if the gold and silver values are high the ore is sent to a lead or copper smelter, and the shipper is not only penalized for the high zinc content of the ore, but the zinc itself is lost.

About 11 per cent. of the zinc production of the United States is recovered from lead-zinc ore. Except for small amounts produced in the Eastern States all of this 11 per cent. comes from the Western States, Colorado and Utah producing the major portion.

The only zinc-bearing siliceous or dry ore produced which is worthy of note is in Colorado. The zinc from these ores is saved in concentrating mills as a by-product.

Fig. 1 shows graphically the present source of zinc and the ores which are not being utilized.

LOSS OF ZINC AT THE MINE

The losses of zinc at the mine, in the aggregate, amount to a large tonnage. Ultimately some of these losses can be recovered. In the average Western lead mine zinc generally occurs as a shell in the stopes from which oxidized ores have been extracted. In most mines the lateral extent of these shells is unknown. To fill these stopes with waste will, in most cases, make the shells an absolute loss. As mining progresses the walls of all stopes should be sampled at regular intervals and a record kept of the results. What is too lean to treat to-day may net a profit in the near future.

In the indiscriminate dumping of low-grade ore and waste on the surface, much zincky material is lost. In the development of many lead-silver mines, considerable tonnages of zinc-iron or zinc-iron-lead

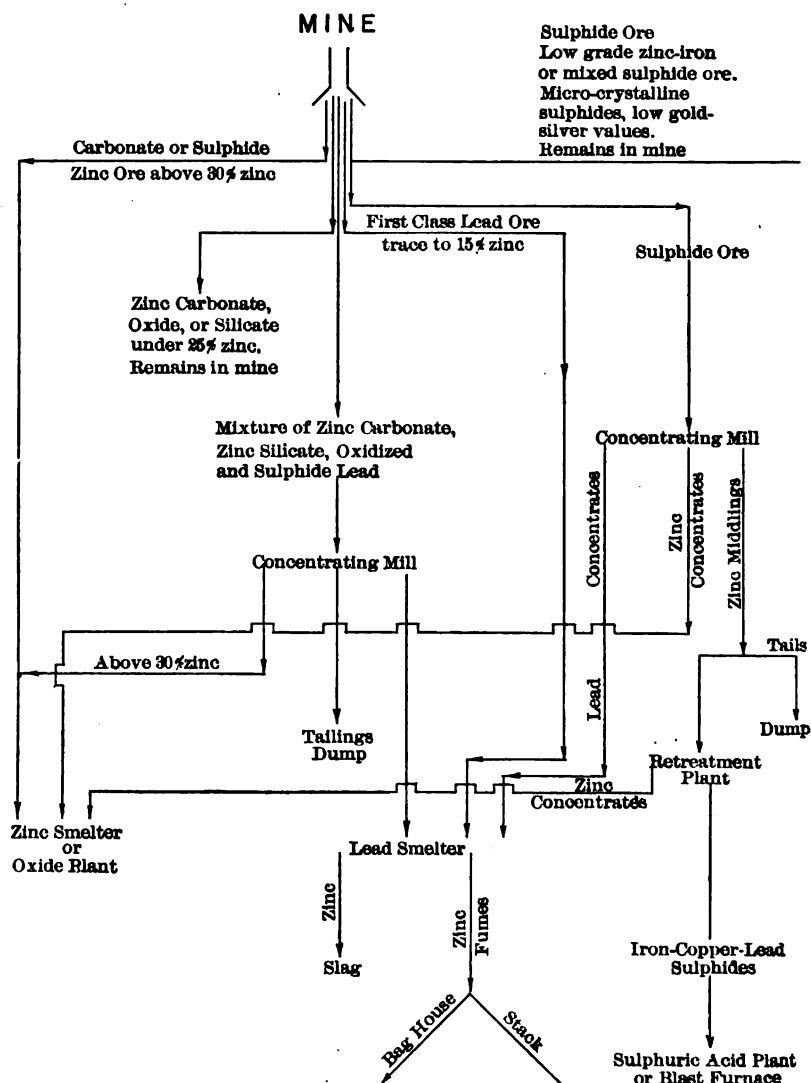


FIG. 1.—PRESENT SOURCE OF ZINC, AND ORES NOT UTILIZED.

material, which are micro-crystalline in their nature, are often encountered in the mineralized zone adjoining the ore or mixed throughout same. Much oxidized zinc ore will be mined above the water level in many silver-lead or lead-zinc properties and in mines producing siliceous ores.

There is a vast tonnage of zinc tied up in the low-grade zinc ores showing values in excess of \$5. Of this value the major portion is in oxidized zinc.

THE PRESENT METHOD OF EXTRACTING ZINC FROM ITS ORES

As previously stated, the retort process is the only process in commercial use at the present time which seems to meet the requirements of the chemistry of zinc; namely, the requirements imposed by the fact that the metal is reduced from the oxide at a higher temperature than is required to volatilize the metal, and for that reason it is reduced as a vapor, which must then be condensed to liquid form. Then, too, the condensation temperature must be regulated within very narrow limits, or a large part of the zinc vapor will condense as very finely divided zinc mixed with oxide (blue powder). Due to these requirements, that is, the proper regulation of the temperature of the reduction and the condensation, the retorts in which the process is conducted must of necessity be small, and this fact in turn requires the use of small charges, and for this reason, as is well known, the zinc smelters can only treat at a profit ores which contain a comparatively high percentage of zinc.

CLASS OF ORES SUITED TO ZINC RETORT SMELTING

At the present time the lower limit of zinc in ores or mill products suited to this process is about 25 per cent. Therefore the ores or products suited to the retort process of zinc smelting may be obtained from:

- (a) Straight zinc ores;
- (b) The concentration and separation of zinc-lead ores;
- (c) The concentration and separation of mixed and complex ores, containing gold, silver, lead, copper, zinc, etc.

CONCENTRATION PROCESSES USED FOR TREATING ZINC ORES

As the ores smelted by zinc smelters are for the most part products of concentration processes, we will first consider a few of these which are in use at the present time, in order that we may have a better understanding of the problem which presents itself. For the sake of convenience, we will consider these processes in the following order:

1. Hydro-mechanical processes;
2. Electro-mechanical processes;
 - (a) electromagnetic,
 - (b) electrostatic.
3. Flotation processes.

Hydro-Mechanical Processes

As these processes are well known, and as the various types of concentrating machines in use at the present time have been fully described in

the literature on the subject, it is not necessary to more than state that such machines are generally used, so far as zinc ores are concerned, for obtaining a concentrated product which may afterward be separated into a zinc product and a lead product, or a zinc product and a product which may contain lead, copper, gold and silver, and which is sent either to a lead or a copper blast furnace.

From *Technical Paper No. 41, U. S. Bureau of Mines*, by Clarence A. Wright, we have the following interesting data showing the efficiency of hydro-mechanical concentration processes in Joplin, Mo. These ores require the simplest form of hydro-mechanical concentration.

Quantity of ore through mill, tons.....	309.70
Quantity of zinc concentrates produced, tons.....	9.235
Assay of zinc concentrates, per cent. Zn.....	59
Quantity of blende (ZnS, 67 per cent. Zn). in concentrate, tons.....	8.132
Quantity of tailings, tons.....	300.465
Assay of tailings from mill (0.96 per cent. Zn.), per cent. of ZnS.....	1.43
Loss of blende (ZnS) in tailings, tons.....	4.296
Total quantity of blende (ZnS) in ore through mill, tons.....	12.428
Total recovery of blende through mill, per cent.....	65.43
The average loss, therefore, in the concentration of zinc ores in the Joplin district is 35 to 40 per cent. There are heavy losses in smelting. Total loss reaches nearly.....	
	50 per cent.

Electro-Mechanical Processes

(a) *Magnetic Separators*.—Although there are many types of separators of this class, a magnetic separator in general consists of a set of magnets so arranged that when paramagnetic or weakly magnetic minerals are brought into the magnetic field they are influenced in such a manner as to effect the desired separation. For example, if a mixed sulphide be crushed to the required degree of fineness and then passed over a wet-concentrating table which will separate it into a galena-pyrite product and a pyrite-blende product and the latter product then be passed over a magnetic concentrator, such as the Wetherill, two products are obtained. The first, a blende product, assaying about 50 per cent. zinc, 10 to 12 per cent. iron, and 1 per cent. lead, is sold to the zinc smelters; the second, a pyrite product, containing some lead and about 5 to 7 per cent. zinc, which is mixed with the galena-pyrite heads from the Wilfley tables, is sold to the lead smelters, for whom it is a desirable ore. In smelting this ore in the lead blast furnace, the zinc is lost in the slag and fume.

(b) *Electrostatic Separators*.—Separators of this class are based upon the fact that while most gangue minerals and certain of the sulphides, such as blende, are relatively poor conductors of electricity, many metallic sulphides and other minerals are good conductors, and likewise upon the fact that the bodies, charged alike electrically, repel one another, while if charged oppositely they attract each other. Thus, as pointed out by Richards:⁴

⁴Richards, Robert H.: *Ore Dressing*, vol. iii, p. 1549 (1909).

"If a mixture of good and poor conductors, in a neutral state, be dropped upon a highly charged conducting surface, the good conductors immediately receive a charge similar to that of the surface and are repelled, while the poor conductors are much more loath to receive the charge and therefore not so readily repelled. If, however, a material charged to a high potential of the opposite sign be fed to the above-mentioned surface, the good conductors, as before, assume immediately the condition of the charged surface and are repelled, while the very poor conductors, carrying a charge opposite to that which the surface carries, by the above law, tend to cling to that surface, thus making a sharper division of the separation. Theoretically the separation of two minerals does not require that one of them be a very good conductor and the other a very poor conductor, but merely that there be a difference in the degree to which they will conduct electricity. Commercially, of course, there are limits to this."

An example of the working of such a separator is the work done at Midvale, Utah, near Salt Lake City, at the plant of the United States Smelting, Refining & Mining Co. This plant has been in operation since 1909, and it is stated⁴ that from 5,000,000 to 7,000,000 lb. of zinc per year is saved which heretofore had gone to waste. At this plant zinc-iron middlings, partly furnished by the company's concentrator, partly from custom ore from the district, are passed over a Huff separator. MacGregor states that the "feed" to the mill assays approximately 0.04 oz. gold, 2.5 oz. silver, 2.5 per cent. lead, 1 per cent. copper, 23 per cent. iron, 28 per cent. zinc, and the separated zinc product assays 48 to 50 per cent. zinc with 3 to 5 per cent. iron, and the iron product 10 per cent. zinc. Most of the gold, silver, copper and lead is found in the iron product, and this product is smelted in the company's lead furnaces.

In this connection it is to be noted that the iron product contained about 10 per cent. zinc. The assays on other products given by MacGregor show the same thing; that is, that in the "iron product of the copper product," or whatever name is given to the product which goes to the blast furnace and from which the greater part of the zinc has been separated, there is still anywhere from 8 to 12 per cent. zinc, and, as previously stated, the zinc in this product is not recovered.

Flotation Processes

The flotation process has been briefly described as follows: A mixture of ore, oil, and water is agitated by a rotary stir; the valuable metalliferous part of the ore, through an affinity of the oil for the sulphides, will float when assisted by the air included by agitation, while the worthless gangue will sink.

Based on this principle, a selective flotation process known as Horwood's process has been devised. It is stated by the *Australian Mining*

⁴ MacGregor, Frank S.: Progress in Electrostatic Ore Dressing, *Transactions of the American Electrochemical Society*, vol. xxiv, p. 352 (1913).

Standard of Jan. 15, 1914, that the successful establishment of this process at the works of the Zinc Corporation was one of the metallurgical achievements of 1913.⁶

"The principle of this process is the roasting of mixed lead-zinc concentrates obtained by flotation, whereby the lead particles are slightly sulphatized while the zinc is unaffected. On floating the roasted product the zinc alone floats and the lead remains submerged.

"A plant having a capacity of 500 tons weekly has been in operation for over six months, treating current and accumulated slime concentrates. The slimes are first dewatered and washed to remove soluble salts, then lightly roasted in an Edwards furnace, and finally treated again by flotation. Typical actual results from this plant are as follows:

	Zinc, Per cent.	Lead, Per cent.	Silver, Oz. per Ton
Feed to roaster.....	40.4	14.6	21.4
Zinc concentrate.....	48.7	5.2	11.6
Lead residue.....	10.2	44.2	54.6"

Thus we see that, as in the electrostatic and electromagnetic processes, the lead residue which goes to the blast furnace contains about 10 per cent. of zinc, and this zinc is lost.

In studying the processes above mentioned, we find that in every case the product which goes to the lead or copper blast furnace does not contain much less than 10 per cent. of zinc, as shown in Table III.

LOSS OF ZINC IN LEAD AND COPPER BLAST-FURNACE SMELTING

In presenting these facts as regards the methods which are used for concentrating zinc ores, we do not wish to be understood as criticising any of the concentration processes mentioned, as they do the work for which they are installed; namely, to separate gold-silver-lead-copper-zinc bearing ores into two products, one of which shall contain enough zinc to enable it to be profitably treated by the zinc smelter, and the other, no more than will permit of its being satisfactorily treated in a copper or lead blast furnace. As is well known, in such furnaces, if the zinc content of the ore constituting the charge is not greater than 10 per cent., it can be gotten rid of in the slag and fumes. The loss of this amount of zinc is, of course, a distinct waste. As to the amount of zinc lost in this manner, the following example will serve to illustrate the same.

If we treat ore containing, say, 28 per cent. of zinc, we separate this into two products, one of which will contain, say, 50 per cent. zinc, while the other will contain most of the gold, silver, copper, and lead, and about 10 per cent. of zinc. In other words, of our original product of 50 tons we will have 28 tons (more or less, dependent upon the per cent. of iron pres-

⁶*Metallurgical and Chemical Engineering*, vol. xii, No. 5, p. 350 (May, 1914).

Table III—Recovery of Zinc by Various Concentration Processes

Process	Per cent. Zinc in Con- centrate	Per cent. Zinc in Product going to Blast Furnace	Per cent. Recovered			
			Zn	Pb	Ag	Rem.
Ordinary Wet Concentration:						
Daly West, Park City, Utah.			50.0	95.0	75-80	
Daly Judge, Park City, Utah.			66.0	92.0	78.0	
Buffalo Hump Mining Co., Burke, Idaho.			25.0	90.0	85.0	
Joplin, Mo.			65.43			
*Wetherill-Magnetic.	50.0	5-7				
^a Huff-Electrostatic	48-50	10.0				
	41.3	12.6				
	49.3	8.6				
*Potter-Delprat.			66.2	38.5	50.9	
^a De Bavay	1910	48.1	11.7			
	1911	48.4	14.2			
	1912	49.2	16.6			
*Elmore Vacuum.	46.0	15.84				
Mineral Sep. Process.			85.4	74.8	80.1	
Horwood Process.	48.7	10.2				
Hyde Process at Butte and Superior			88.1		83.4	For
			86.4	62.5	80.0	1913

* Magnetic Separation of Zinc Blende at Denver, Colo. *Engineering and Mining Journal*, vol. lxxiv, No. 7, p. 217 (Aug. 16, 1902).

^a MacGregor, Frank S.: Progress in Electrostatic Ore Dressing. *Transactions of the American Electrochemical Society*, vol. xxiv, p. 352 (1913).

^a Hoover, T. J.: *Concentrating Ores by Flotation*, p. 91 (London, 1912).

^a *Idem*, p. 97.

^a *Idem*, p. 105.

ent) of product containing 50 per cent. zinc, and 22 tons containing 10 per cent., and in addition most of the gold, silver, copper, and lead contained in the original product. This latter product goes to the blast furnace and ordinarily the zinc content is not saved, but is lost in the fume and slag. This amount of zinc therefore represents a loss of almost 1,500,000 lb. a year. Moreover, this zinc is lost for all time, as it cannot be recovered, and when we consider that this is lost from the treatment of only 50 tons of concentrates per day, we can readily understand, as has been shown by those who have investigated this subject, that the amount of zinc lost in this way amounts in the aggregate to thousands of tons annually, and, as has been pointed out by Clevenger⁷ and others, in the

⁷ *Metallurgical and Chemical Engineering*, vol. xii, No. 5, p. 299 (May, 1914).

inability to recover this zinc lies one of the greatest weaknesses of our present-day metallurgical practice, in that the zinc so lost represents a distinct economic waste, much of which will never be recovered, and that we must look to the prevention of this waste as one of the sources from which we shall in the future obtain a large portion of our zinc supply.

The same thing is of course true to some extent of lead, for in copper smelting, if lead is present, it is volatilized and no attempt is made to recover it, and so the lead is completely lost. As before stated, these facts are not presented with the idea of bringing an indictment against electro-mechanical methods of concentration, or against modern methods of copper and lead smelting, but rather to emphasize the fact that although mechanical methods of concentration and modern methods of smelting have reached a very high stage of development, their use nevertheless entails a very great loss of zinc, which is not recoverable, and that therefore it is quite likely that in the future it will be necessary to stop these wastes in order to add to the available supply of zinc.

POSSIBLE FUTURE SOURCES OF ZINC SUPPLY

A study of the data as given in the preceding tables, etc., brings out two points very clearly; namely, that only straight and mixed low-grade sulphide ores are at the present time being concentrated with any degree of success, at least so far as effecting a recovery of the zinc content of the ore is concerned, and, second, that low-grade carbonate ores—that is, ores containing less than 20 per cent. zinc—are not treated at all. Therefore, we may say that at present zinc ores are not being treated by any process except the retort process, and that this process cannot profitably treat ores containing less than 20 per cent. zinc. Thus we see that, so far as zinc ores themselves are concerned, we have as a possible future source of zinc:

1. The values which are now wasted as a result of imperfect concentration methods;
2. The zinc which is lost in the slag or as fume when ores containing an appreciable amount of zinc are smelted in copper and lead blast furnaces;
3. The low-grade carbonates which cannot be treated at a profit by present-day metallurgical processes;
4. The micro-crystalline sulphides of zinc, iron, and lead; *e.g.*, such ores as are found in the Commonwealth mine, near Reno, Nev.; in some of the Pioche, Nev., mines; and in Rico, Colo.

It is true that ores of the latter class containing sufficient values in lead and silver may be mined and shipped as lead-silver ore, but if so, no attempt is made to recover the zinc; in fact, in such cases it constitutes an interfering element and is lost in the slag or as fume when such ores are smelted in copper and lead blast furnaces. Moreover, the shipper is

penalized at so much per unit for the zinc content of his ore. For this reason, as above stated, we may consider the zinc that is lost in this way (that is, as slag and fume in the smelting of ore where zinc is considered a harmful impurity) as one of the possible future sources of zinc.

THE PROBLEM OF THE METALLURGIST AND THE PRESENT STATUS OF NEW PROCESSES

If these are the possible sources of zinc, the problem that presents itself is how to utilize them in a commercially feasible manner, and it is one to which is being given a great deal of time, attention, and money.

Perhaps no one process can be devised which will solve it, but that on the other hand local conditions, and the nature of the ore to be treated, will determine in each particular instance whether an igneous concentration, a chemical, an electro-chemical, an electro-thermic, or some other process shall be used which does not belong to any of the types of processes just mentioned. As above stated, a great deal of work has been done in an experimental way on all of the types of processes above mentioned, and inasmuch as they have been quite fully discussed from time to time in the technical journals and before this and other societies, it is not necessary to do so in this paper.

As to the present status of these various processes, we can perhaps do no better than to quote from an editorial on "Developments in the Metallurgy of Zinc in 1913," which appeared in the January number (1914) of *Metallurgical and Chemical Engineering*:

"From a metallurgical point of view the status of spelter production was not materially changed during 1913. This is not to imply a dearth of experiment and research in the winning of zinc; on the contrary, there was marked activity in almost all branches of zinc-ore treatment. Some hopes have been realized, and progress has been recorded; but a reduction process has not been brought forth that can be considered a serious competitor of retort smelting.

"On the whole a commendable lot of work has been done toward improving present processes and in attempts to develop new ones. A great deal of time and money have been spent in investigation, and there is apparent willingness to test new ideas of merit. Apparently no line of development has been finally dropped, and the coming year should witness a nearer approach to the realization of the hopes and ambitions of different investigators."

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 29th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Precipitation of Copper from Solution at Anaconda

BY FREDERICK LAIST AND F. F. FRICK, ANACONDA, MONT.

(Salt Lake Meeting, August, 1914)

Introduction

IN a leaching process, having obtained the copper in solution, the choice of the precipitation method is influenced by the following factors:

1. Availability of precipitant.
2. Adaptability to the leaching process.
3. Final product desired.
4. Regeneration of leaching solutions.
5. Fouling of solutions.

There are three general classes of precipitation methods:

1. The use of iron, scrap or sponge.
2. Electrolytic deposition.
3. The use of some gas or reagent by which the copper is obtained, usually in the form of an intermediate precipitate which requires further treatment. For example, the precipitation of the copper as Cu_2Cl_2 from chloride solutions, with SO_2 gas.

The use of iron is adapted to almost any process. It has the great advantage of extreme simplicity, and the recovery of the copper and any values present in a highly concentrated, easily treatable form. The disadvantages are the cost in isolated places, the uncertainty of the scrap-iron market, and the fouling of the leaching solutions. The latter is a serious factor, when a reagent such as salt is used in them, which makes necessary their re-use.

Where applicable, electrolytic deposition is very attractive. The copper is obtained in a form directly marketable, there is a regeneration of acid and no fouling of solutions. Unfortunately, chloride solutions do not lend themselves to electrolysis. An entirely satisfactory anode material has not yet been discovered, although magnetite, apart from being rather brittle, seems to answer the requirements fairly well.

The use of a method of the third class is involved in many special

leaching processes, and is highly attractive theoretically. Such methods usually involve regeneration of leaching solutions, no fouling of solutions, and the use of some cheap reagent, as lime, or a by-product of the process itself, or of some process such as smelting. Some method of this kind is undoubtedly that of the future where electrolytic deposition is not applicable.

The copper solution from the 80-ton leaching plant, in which the sand-treatment method now being installed on a large scale was developed, averaged Cu, 1.91; $\text{Fe}_2\text{O}_3 + \text{Al}_2\text{O}_3$, 3.88; NaCl, 8.3 per cent.; Ag, 0.634 oz. The copper content of this solution cannot be increased much economically. The application of this solution to successive lots of roasted tailing, from which about 10 lb. of copper per ton can be recovered, would necessitate excessive washing of the tailing. The copper can be precipitated from this solution by means of scrap iron, the cement copper going directly to the blast furnaces or converters. The use of sponge iron is more attractive than the use of scrap iron, however. It can be made from calcines which contain about 50 per cent. iron. This makes the process independent of the scrap-iron market. But the use of iron necessitates the discard of about one-fourth of the solution each time to stop the accumulation of iron sulphate, and this leads to the loss of much valuable salt.

Owing to its high chlorine content the solution is not adapted to electrolysis. With a view to getting something better than scrap iron, precipitation experiments have been and are being conducted on precipitation by sulphuretted hydrogen made from matte (also from calcium sulphate), sponge iron from calcined concentrates, and sulphur dioxide from roasting-furnace gases.

In the following discussion we shall attempt to explain the operation, difficulties, advantages and disadvantages, and the results obtained with each of these methods.

Hydrogen Sulphide

Hydrogen sulphide was used as the precipitant in the work done on the experimental plant operated during the summer of 1912.

The precipitation tank, 10 by 10 by 7 ft., was equipped with an agitator and a system of lead pipes in the bottom for gas distribution. The generator was a lead-lined iron drum 2 ft. in diameter and 4 ft. high. It was built to stand 90 lb. pressure and fitted with air and steam connections. The iron sulphide was put in through a hand hole in the top. The acid was introduced through an iron pipe by gravity. The gas passed through a lead main to the distributing pipes. When action ceased a plug was removed from the bottom of the tank, 90 lb. air applied and the residue blown out.

The copper sulphide was filtered in a small hand-made wooden filter press, the inside dimensions of the frames of which were 12 by 12 by 1½ in. Ordinary filter paper backed up with heavy canvas was used. The copper sulphide was allowed to settle over night, the clear solution drawn off and the thickened residue drawn into a pressure cylinder, from which it was forced with 90-lb. air through a ½-in. lead pipe to the press.

The absorption of the H_2S gas was very inefficient. The gas entered under considerable pressure and escaped in big bubbles through a depth of 3½ ft. of solution. Some of the iron sulphide, which was the commercial kind, was of very poor quality. It was found that when ground to 4 mesh the rate of evolution of the gas was about right.

A charge was 400 lb. of water, 100 lb. of 4-mesh iron sulphide, and 130 lb. of 66° Baumé acid which was slowly added. The acid strength on starting was about 24.5 per cent. and averaged 7.5 per cent. at the end; 2.4 lb. acid per pound of copper were required for the generation of the H_2S . The efficiency of the acid used in the generation was about 62 per cent. The efficiency of the FeS in the laboratory was 80.2 per cent., but in the plant only 54 per cent. was realized, due to poor absorption of the gas.

About 100 lb. of matte were made by fusing in proper proportions a heavy pyrite cre, an oxidized iron ore, and lime rock, in graphite crucibles. This matte decomposed readily and was 97.2 per cent. as efficient as the commercial FeS .

Analysis of Matte and FeS

	Matte, Per Cent.	FeS , Per Cent.
FeO.....	78.8	81.2
S.....	31.1	28.6
Cu.....	2.23	
Ins.....	0.8	0.4

The filtration gave some trouble, due to defects in the press. When the press was working well the cakes were of about the consistency of cheese and assayed: Cu, 58; S, 29.0 per cent.; Ag, 69.5; Au, 0.04 oz. The cakes carried 48 per cent. moisture.

With a generator having an agitating device, and a properly designed absorber, there is no doubt but that a high efficiency could be obtained by the use of this process. The reaction between $CuSO_4$ and the H_2S liberates 1.55 lb. acid per pound of copper precipitated, and the reduction of the ferric salts also liberates some acid.

The advantages of this process are,

1. Regeneration of acid.
2. No fouling of solutions.
3. A product which is high grade and can be readily worked into the smelter process.

4. The matte used in generating the H_2S is enriched by removal of its iron and sulphur.

Another method of generating H_2S which was considered, but not experimented with in connection with the present problem, consists of an adaptation of the Chance process, which is most successfully used for working up the so-called "soda waste," resulting from the manufacture of soda by the Le Blanc process. "Soda waste" is largely composed of calcium sulphide (CaS). This is finely ground and made into a thin mud with water and is then decomposed with carbon dioxide, the reaction being as follows: $\text{CaS} + \text{CO}_2 + \text{H}_2\text{O} = \text{CaCO}_3 + \text{H}_2\text{S}$.

Experiments were made on this line by one of the writers several years ago and gave quite satisfactory results. The calcium sulphide (CaS) was made by reducing gypsum (CaSO_4) with coal in a shaft furnace similar to a lime kiln. The gypsum and coal were formed into bricks with water. It was found that a very fair quality of calcium sulphide could be made in this manner, a yield of about 75 per cent. of the theoretical being obtained.

The calcium sulphide was decomposed by means of gases taken from the reduction furnace. These averaged about 14 to 16 per cent. CO_2 and answered the purpose very well. The calcium sulphide absorbs the CO_2 with avidity. No heat is required. A train of five cylinders, 1 ft. in diameter by 3 ft. tall, constructed of galvanized iron and connected by pipes and valves in such a way that any cylinder could be made the first (or last) of the series, was used for the decomposition of the CaS . The CO_2 gas was forced through the cylinders by means of a small compressor. The gas on leaving the last CaS cylinder was conducted to a similar series of wooden cylinders containing copper solution, no intermediate pumping of the gas being required. The absorption of the H_2S was perfect. No odor whatever was perceptible at the outlet of the last cylinder. The conversion of the calcium sulphide into calcium carbonate by the carbon dioxide gas was also perfect. No odor of H_2S could be detected when the residue from the tanks was dissolved in dilute hydrochloric acid.

Sponge Iron

In June, 1913, a sample of a concentrate high in iron sulphide was roasted to a calcine assaying: SiO_2 , 4.3; FeO , 72.2; S, 4.2; Cu, 6.2; Fe, 56.2 per cent.

Two parts of this calcine were mixed with one part of fine Diamondville coal, sealed in a graphite crucible and heated to about $1,700^\circ \text{F}$. for a couple of hours. The residue had a copper equivalent of 1.77. (The "copper equivalent" is the grams of sponge iron required to precipitate 1 g. of copper from a 1 per cent. copper solution prepared from crystallized copper sulphate.) A series of crucible tests followed, which gave suffi-

ciently good results to warrant the collection and roasting of about 125 tons of the special high iron sulphide for further experiment.

The roasting was done in a 16-ft. six-hearth MacDougall furnace at an average rate of 16.3 tons per day. Oil burners were used on the lower hearths to maintain the temperature so that the roast could be carried below 4 per cent. in sulphur.

Assays of Concentrates and Calcine

	Concentrates Per Cent.	Calcine Per Cent.
Cu.....	5.7	
FeO.....	49.0	
Ins.....	7.0	14.0
S.....		3.4

A retort furnace was built near the leaching plant and operated during October and November, 1913. The equipment consisted of a three-retort coal-fired furnace, an elevation of which is shown in Fig. 1, and three air-tight cars for cooling the sponge. The retorts, similar to those used in gas making, were 8½ ft. long, 15 in. deep, and 2 ft. wide inside, had 3 in. thick walls, and cast-iron fronts with gas connections. The fine reduced iron reoxidized so easily that air-tight cars were necessary for cooling.

The charge consisted of a mixture of 30 parts ground Diamondville coal and 70 parts calcine. The first lot was prepared by grinding the coal, mixing with the calcine and grinding the mixture so that 96 per cent. passed 40 mesh. Later it was found that it was unnecessary to grind the calcine. The coal was ground in a Hardinge mill so that 90 per cent. passed 60 mesh.

The first charge, only one muffle used, was left in for 24 hr. and gave a copper equivalent of 2.3. This was continued for a couple of weeks. The temperature was raised as fast as possible and the pyrometer reading was at 1,600° for at least 4 hr. before discharging. The pyrometer rod was not stuck down in the mixture, but was above it. The results from these charges were uniformly good, the copper equivalent varying from 2.2 to 3.0 g. iron per gram of copper. No special precautions were taken at any time in charging or discharging. The samples of material were taken with a pipe sampler from a car of cooled material. The charge was always dampened to prevent too much dusting. The charges were about 450 to 500 lb. of original mixture. These gave on an average, 275 to 300 lb. of "iron." One charge during this time was rabbled every half hour for 9 hr. and the temperature worked up as fast as possible. It was discharged at a temperature of 1,500° F. and gave a copper equivalent of 2.7 g. iron per gram of copper.

At the end of the first two weeks it was deemed advisable to try to

get more material through the furnace, and a scheme was worked out to give each charge 8 hr. and work them off faster. This proved unsatisfactory. It was not possible to obtain more than $1,300^{\circ}$ temperature and the resulting iron was very poor. About this time the muffles began to crack and let air in. This naturally resulted in the production of a very poor grade of iron.

The best iron which was made under the foregoing conditions was a black, dusty material. It contained considerable coke and was quite light. The coarser calcine gave as good results as the ground material. None of the pyrometer readings was over $1,620^{\circ}$. In all cases the pyrome-

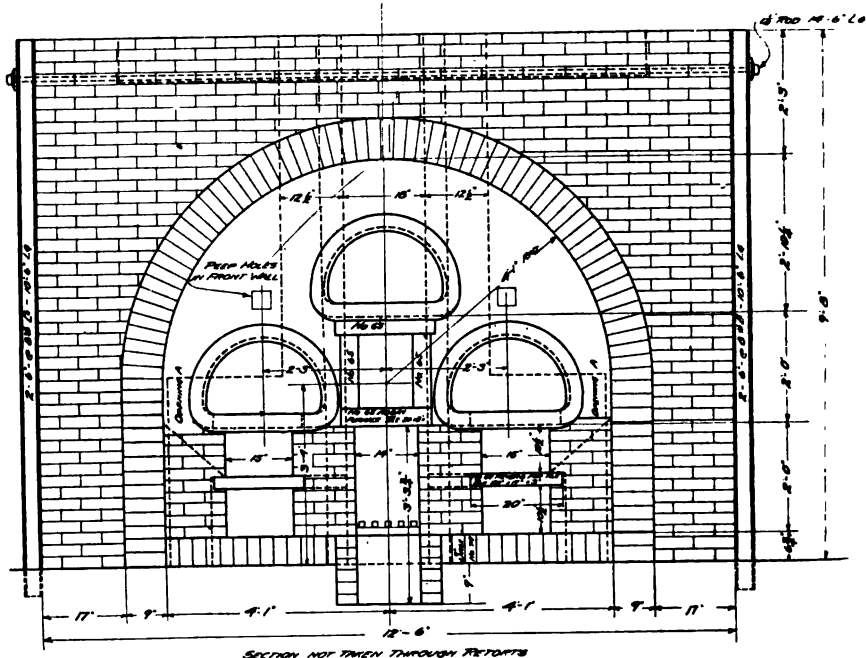


FIG. 1.—ELEVATION OF SPONGE IRON RETORT FURNACE.

ter tube was over and not buried in the mixture. Toward the end of the experiments, the muffles had quite a cake in the lower corners and on the bottom, which seemed to be more or less fused to them, and was impossible to clean out.

It was early recognized that while a good sponge iron could be made in retorts, the fuel efficiency was low and the life of the retorts limited. So it was decided to try reduction in a specially rebuilt MacDougall furnace. The furnace was so equipped that hot partly roasted calcine from another furnace was fed to the first hearth by a steel conveyor. The roasting was completed and the calcine preheated to $1,700^{\circ}$ by oil

burners on the third hearth. The three lower hearths were equipped with oil-fired muffle floors. Fine coal was fed in on the fourth hearth. The plan was to obtain a mixture of the preheated calcine and fine coal on the lower hearths under strongly reducing conditions such that the reduction would be accomplished. The oil burners directed into the lower hearths worked alright but those into the muffles, due to the necessarily restricted size of the muffle, localized the flame too much, melting the bricks in the direct path of the flame and depositing carbon on the other portions.

The oil burners were removed and coal firing was tried. Two fire boxes were installed on the third floor, the fine coal being fed on the fourth floor. The addition of this cold coal lowered the temperature to such an extent that the calcine did not reduce, and on increasing the temperature the coal burned off. It seemed impossible to get a high temperature and a reducing atmosphere at the same time on the lower hearths. It was then decided to add another fire box. At first it was connected to the fifth hearth muffle only, but finally to both the fourth and fifth hearth muffles. While increasing the heat in these hearths, the muffles were poor conductors of heat and the piers and floors above burned out. Gas analyses made on the fifth and sixth hearths showed CO_2 , 15; O_2 , 2.7; CO , 0.7 per cent., the temperature in the fifth hearth being $1,240^\circ \text{F}$. The best gas sample showed only 2.0 per cent. CO on the fifth hearth and this for only a short time.

Fine coke was next tried and had the same effect of cooling the calcine. Though added on the third hearth, it practically all dropped down to the fourth hearth before mixing with the calcine. By laboratory experiments it was shown that coke did not have the same reducing action on calcines as coal or charcoal under the same conditions and its use was discontinued. No appreciable reduction was obtained during this period. The product made by the furnace was magnetic, but showed no signs of metallic iron and would not precipitate any copper.

While using two fire boxes the coal burned was 4,000 lb. per 24 hr. When the three were in operation it required 7,500 lb. per 24 hr. The special iron concentrates fed to MacDougall No. 62 at a rate of 26 tons per 24 hr. averaged: Insol., 7.1; FeO , 47.6; and S, 46 per cent. They were roasted to 5 per cent. S.

Owing to the cooling effect of adding the coal dust to the calcine in the furnace, feeding a mixture of coal dust and calcine thoroughly mixed was tried. A mixture of two-thirds calcine and one-third coal dust by weight was fed into the top of the furnace. No difficulty was experienced in obtaining a high heat on the three upper hearths, but the lower hearths were cold. No reduction was obtained, due, probably, to unavoidable leakage of air.

In January, 1914, it was decided to start a new series of experiments. A series of laboratory experiments had been made in December using

Diamondville coal, Sunnyside coke, and charcoal as reducing agents. Equally good results were obtained from the coal and charcoal, but practically no reduction from the coke at the temperatures used. The significance of these results was not appreciated at the time, as will be explained later.

The retort furnace was remodeled, a scrap cast-iron hydraulic cylinder $8\frac{1}{2}$ ft. long and 18 in. in diameter, with a hinged door, being installed. This retort was operated for a couple of weeks with a mixture of 30 parts fine Diamondville coal and 70 parts calcine, with indifferent results.

In the meantime it happened that a crucible run was made, in the laboratory, using lumps of coal about thumb size instead of fine coal. To our surprise, excellent results were obtained. A new line of experiments was started in which calcine was heated and then mixed with coarse coal. A charge of calcine was heated to $1,500^{\circ}$ in the retort and coarse coal was charged and rabbled in. The results were poor because of the difficulty of rabbling the coarse coal down into the fine calcine. However, some reduction was obtained.

The great disadvantage of the retort method is the low fuel efficiency. This was especially true when Diamondville coal, 38 per cent. volatile combustible matter, was used. The gas formed carried off the heat as rapidly as applied and the temperature could not be raised above $1,000^{\circ}$ F. until it was driven off. It was known that efficient reduction did not take place under $1,500^{\circ}$ F., after all the gas was driven off; also that while blast-furnace coke was not a reducer at this temperature, charcoal was. So it was suggested that the coke resulting from the coal must be the reducer and that the hardness and character of the coke must be factors in its efficiency.

Diamondville coal assayed in 1912: volatile combustible matter, 38.38; fixed carbon, 45.28; ash, 10.01 per cent. It is not a coking coal, in the commercial sense, coke resulting from it being very soft and friable.

A charge of fist-size Diamondville coal was coked in the retort at $1,400^{\circ}$ F. A charge of coke was ground, mixed with calcine, and reduced in the retort with indifferent success. The next day the remaining coke, uncrushed, was charged with calcine into the retort. Conditions seemed to be just right this day and an excellent product resulted. The residue was a gray spongy mass mixed with coarse unconsumed coke and had a copper equivalent of 1.4. It was now decided to preheat coke and calcine under efficient conditions and mix them in an air-tight container.

The retort was shut down and a series of 40 crucible tests made in the laboratory to determine conditions of time, temperature, per cent. of reducer, hardness of coke, possibility of mixing hot, and other details. The results of these experiments showed that if 15 parts of Diamondville coke and 85 parts of calcine could be preheated to $1,800^{\circ}$ F., mixed, and maintained at this temperature for an hour, the resulting residue would have a copper equivalent of 1.5.

A double crucible furnace to hold two No. 50 graphite crucibles was built. Oil burners were used for heating. The mixer was a sheet-iron shell 2 ft. deep and 20 in. in diameter lined with $4\frac{1}{2}$ in. of fire brick. The mixer was provided with an air-tight cover and three tuyères, and placed on trunnions so it could be dumped. Coke was burned in the mixer to preheat it, air being supplied through the tuyères. When the charge was ready the coke was dumped from the mixer and the tuyères closed.

Charges of calcine and coke were heated in the crucibles to between $1,700^{\circ}$ and $1,800^{\circ}$ F. and dumped together into the mixer. There was too much cooling in this manipulation and poor results were obtained. Putting the coke in the mixer and heating it by burning a part of it, using air through the tuyères, was tried. This failed, due to the difficulty of getting the coke heated to a uniform temperature throughout. However, a copper equivalent of 3.2 was obtained from a picked sample.

It was then decided to build two small hand-rotated, oil-fired, brick-lined cylinders for preheating, from which the charges could be very quickly drawn into the mixer. The soft Diamondville coke was heated in one cylinder and the calcine in the other cylinder to $1,800^{\circ}$ F. The charges were so small, however, that when they were transferred to the mixer they lost about 300° F. in temperature. The resultant mixture, $1,500^{\circ}$ F. at the start, cooled rapidly and very poor results were obtained.

Heating a mixture of the coke and calcine in one cylinder to $1,800^{\circ}$ and then quickly sealing the ends, gave a product with a copper equivalent of 2.35. This was so promising that a larger furnace of the Brückner type was made by lining a section of a White Howell furnace, 5 ft. in diameter and 12 ft. long, with $4\frac{1}{2}$ in. of fire brick about the middle and a 30-in. brick wall in each end, with a 12-in. opening in one end for an oil burner and an 18-in. opening in the other for a stack connection. A 6-in. charging and discharging door was put in one side midway between the ends, and a power drive arranged which gave 1.28 rev. per minute. This gave a furnace 4 ft. 3 in. in diameter inside and 7 ft. long, with a capacity of 2,400 lb. of charge. Fig. 2 is a photograph of this furnace, including the oil burner.

The sheet-iron box around the middle of the furnace is stationary and makes a reasonably air-tight space in which the sponge iron can be discharged without excessive oxidation. The discharging door revolves inside the box and the iron falls into a rapid stream of water in a launder below, where it is quenched and washed into a collecting box. Complete data on this furnace have not been obtained. However, the results obtained so far are satisfactory and no difficulties are apparent which will prevent the development of the process to a commercial basis.

The furnace is operated as follows:

1,400 lb. calcine are charged and heated with a fuel-oil flame to about

1,300° F. This requires about $1\frac{1}{2}$ hr. In a commercial plant the calcine would be drawn hot directly from the MacDougall hoppers to the furnace. About 600 lb. of coal are then shoveled in through the front, in small lots. The furnace continues to revolve and in about $\frac{3}{4}$ hr. after starting to charge the coal the hydrocarbons are burned off. The oil flame is again started. In $1\frac{1}{4}$ to 2 hr. the charge is up to 1,680° to 1,700° and reduction is complete. The discharging door is removed and the charge quenched.

We are making sponge iron with a copper equivalent of 1.5 on samples screened through a 14-mesh screen. The coal used amounts to 40 per cent. of the weight of the calcine. It is probable that this percentage



FIG. 2.—SPONGE IRON FURNACE.

can be reduced. Coke to the extent of 25 per cent. of the weight of the coal is recovered by screening the product discharged from the furnace. This coke can undoubtedly be used again. The quenching works well, the quenched product averaging 1.7 copper equivalent. The best results are obtained at a temperature of 1,680° to 1,700° F. If the temperature is carried much above 1,700° F. there is a tendency for the charge to nodulize. The fuel consumption for heating, 20 gal. fuel oil to a ton charge, is very reasonable when it is considered that the process is intermittent, the scale is small, and the apparatus is still in the experimental stage. We expect to present a paper later giving a full account of this process.

Sulphur Dioxide Precipitation

When cupric copper in solution is reduced with SO_2 , in the presence of chlorides, there is a precipitation of cuprous chloride, the amount of

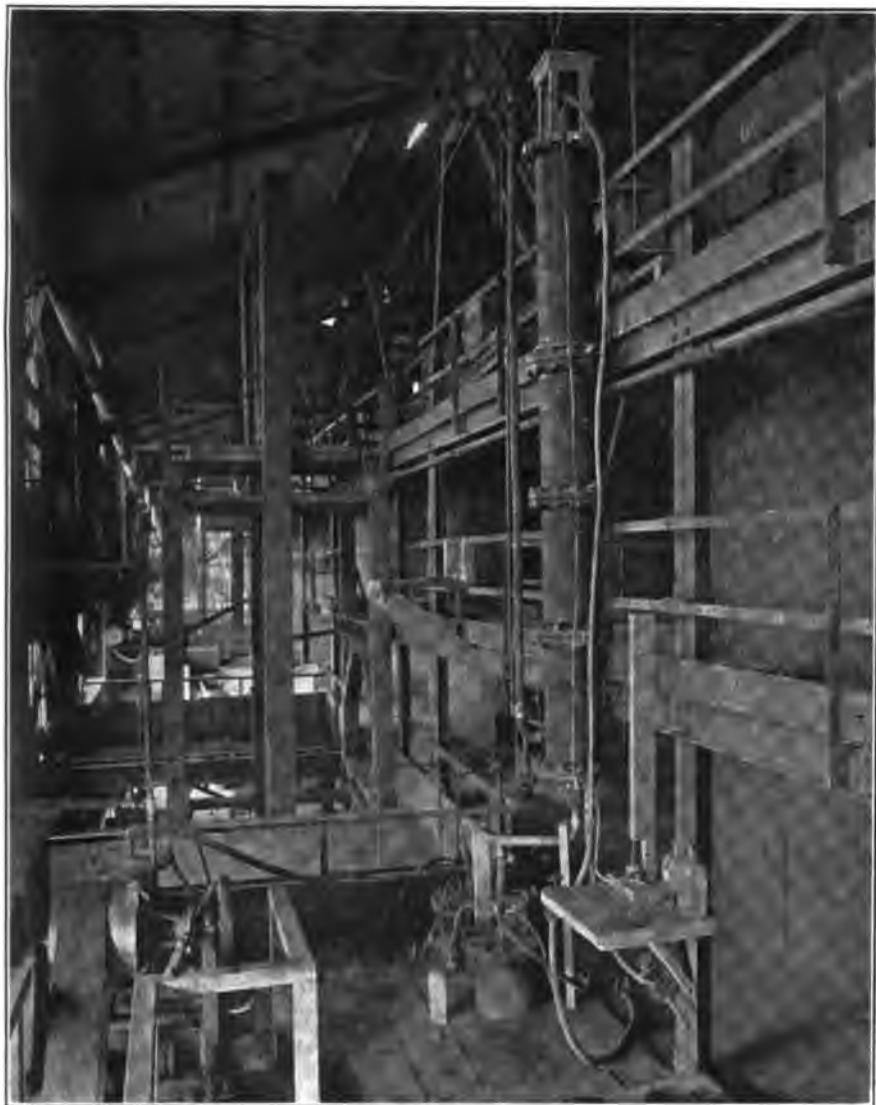


FIG. 3.—TWO-TON SULPHUR DIOXIDE PRECIPITATION PLANT.

precipitation depending on the solubility of the cuprous chloride in the solution. The reaction is, $2\text{CuCl}_2 + \text{SO}_2 + 2\text{H}_2\text{O} = \text{Cu}_2\text{Cl}_2 + \text{H}_2\text{SO}_4 + 2\text{HCl}$.

According to this reaction the equivalent of 1.54 lb. of H_2SO_4 is regenerated per pound of copper reduced. Actually, from our solutions, 2 to 2.5 lb. of acid per pound of copper reduced are regenerated, due to reduction of ferric salts and a catalytic action of the cuprous chloride. Either pure SO_2 , gaseous or liquid, or a gas containing 10 per cent. SO_2 by volume, can be used for the reduction. A gas containing less than 8 per cent. SO_2 by volume does not give satisfactory results. The most satisfactory conditions for the reduction and precipitation of the copper, so far as we have determined them, are saturation of the cold solutions by passing the SO_2 gas through at 15 lb. pressure per square inch, heating to boiling under 20 lb. pressure per square inch and then cooling to 60° to 70° F. The

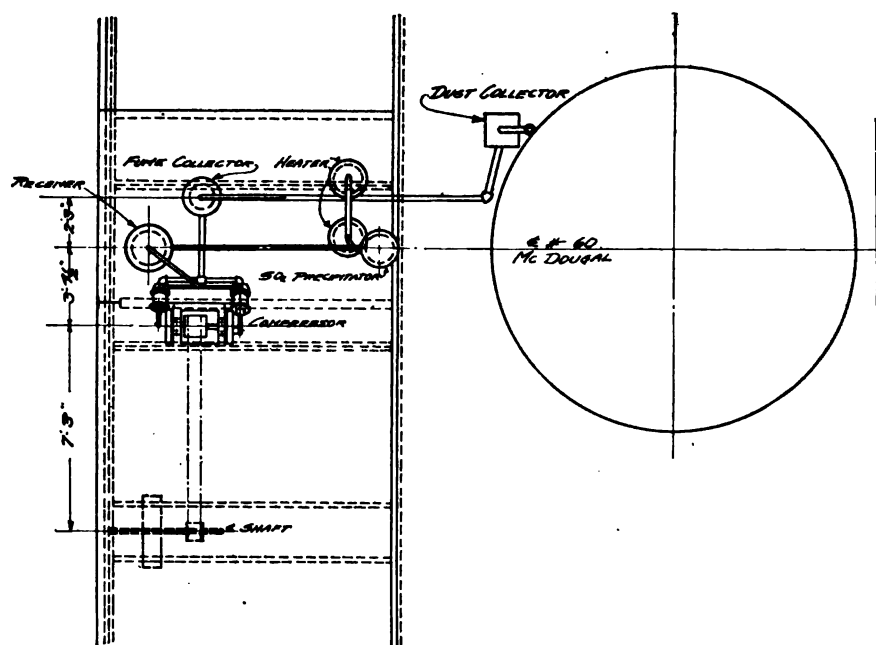


FIG. 4.—PLAN OF 2-TON EXPERIMENTAL SULPHUR DIOXIDE PRECIPITATION PLANT.

cuprous chloride separates as a heavy white crystalline precipitate which settles readily.

Three and one-half tons of the 2 per cent. Cu solution carrying 8.5 per cent. NaCl have been precipitated in 500-lb. lots. An average of 80 per cent. of the copper and 100 per cent. of the silver was precipitated, giving a tail solution carrying 0.4 per cent. Cu and 4.0 per cent. H_2SO_4 , the cuprous chloride being soluble in this 8.5 per cent. NaCl solution to the extent of 0.4 per cent. Cu.

The first experiments were made in a lead-lined iron auto-clave, holding about 700 cc. The solution was saturated under 10 lb. pressure

per square inch with pure SO_2 gas, obtained from cans of liquid SO_2 , transferred to the auto-clave, and heated to boiling under 10 lb. pressure per square inch. On a leaching-plant solution carrying 1.5 per cent. Cu a 71 per cent. precipitation was made.

The copper solution used in the following experimental work contained: Cu, 2.0; $\text{Fe}_2\text{O}_3 + \text{Al}_2\text{O}_3$, 4.6; NaCl, 4.8 to 8.5; H_2SO_4 , 0.5 per cent.; Ag, 0.45 oz.

A lead-lined pressure tank 1 ft. in diameter and 2 ft. deep, with a lead heating coil, was used. The copper solution from the 2,000-ton sands-leaching plant will have about this analysis and carry 8.5 per cent. NaCl. The solution was saturated under pressure, which was maintained during the heating stage. At first pure SO_2 was used. Later 10 per cent. SO_2 gas made from pure SO_2 and air was tried. After the details of manipulation were worked out a 90 per cent. precipitation was made with no trouble. Following are the data of one of these runs:

Volume solution, liters.....	26.5
Per cent. Cu in head solution.....	2.0
Per cent. NaCl in head solution.....	4.8
Per cent. Cu in tail solution.....	0.2
Per cent. Acid in head solution.....	0.5
Per cent. Acid in tail solution.....	4.0
Per cent. SO_2 in gas.....	10.0
Saturation time in hours.....	3.0
Heating time in hours.....	4.0
Final temperature in degrees F.....	195.0
Saturation pressure in pounds.....	15.0
Heating pressure in pounds.....	20.0
Per cent. precipitation.....	90.0

NOTE.—Later by a change of steam connections the heating was accomplished in $\frac{1}{2}$ hr.

Experiments were made on the operation of a MacDougall roaster to give a 10 per cent. gas. The furnace was run hot on a heavy feed of undried fine concentrates. The gas, obtained without difficulty at 10 per cent., was drawn from the second hearth by a small compressor, and compressed to 90 lb. in a 5-cu. ft. oxygen cylinder. This gas gave as good results as the mixture of pure gas and air.

The success of the experiments mentioned warranted a trial of the method on a larger scale, 500 lb. to the charge. The installation shown in Fig. 3 was put in at the MacDougall building. It consists of a water-jacketed cooling pipe from the furnace, a dust catcher, a bag box, a small compressor, a receiver, the absorber, and two heaters. The general plan is shown in Fig. 4. A section of the absorber and one of the heaters is shown in Fig. 5. The absorber is made of sections of lead-lined 1-ft. diameter pipe with perforated diaphragms between sections. It holds about 500 lb. of solution. The heaters are lead-lined cylinders 4 ft.

long and 1 tf. in diameter. A woolen bag is used to remove the fine dust and fume. The residue in the bag consists of fine dust, elemental sulphur,

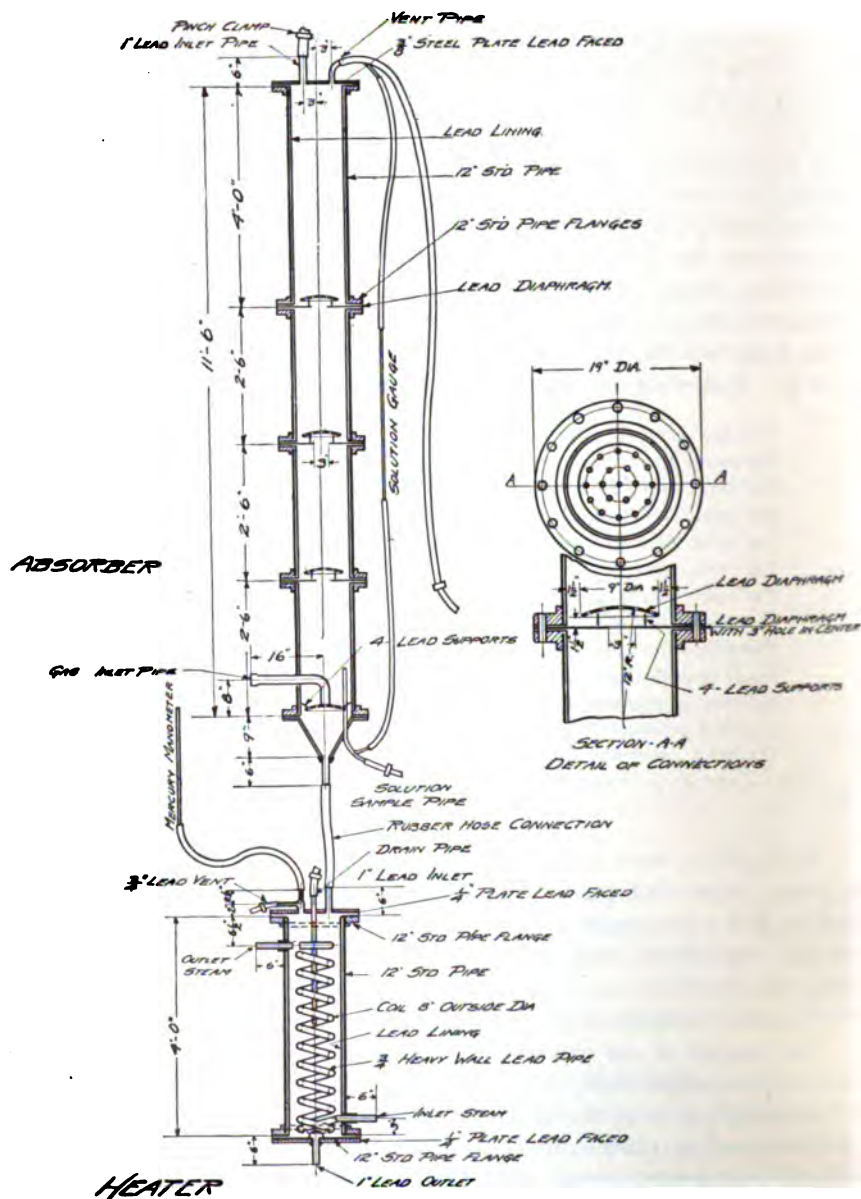


FIG. 5.—SECTION OF ABSORBER AND HEATER.

arsenic, and sulphuric acid, and is very corrosive, the bags lasting only about 6 hr.

The absorber is charged with about 500 lb. of 2 per cent. copper solution. The SO_2 gas is forced in under the lowest diaphragm. A pressure of 15 lb. per square inch is maintained by regulation of the vent. The gas has been passed through at rates varying from 0.8 to 6.1 cu. ft. per minute. The absorption of the SO_2 gas is almost complete for the first hour, and then gradually decreases to about 60 per cent. The solution changes from the blue cupric color to a greenish-brown color, and some Cu_2Cl_2 is precipitated. Samples are taken every hour, and after 2.5 to 3 hr. a small portion boiled in an Erlenmeyer flask and cooled has a characteristic appearance when the reduction is completed, which we have learned from experience indicates that the charge is ready for heating. The charge is drawn off in 125-lb. lots to the heaters. The heating time averages 35 to 40 min. The pressure is maintained at 20 lb. After 35 to 40 min. the pressure is gradually reduced and the charge allowed to boil several minutes. The four heats from one absorber charge are drawn into a barrel, and after cooling over night the supernatant solution is sampled for copper.

The first 18 runs were made on a solution running 2 per cent. Cu and 4.8 per cent. NaCl. An average precipitation of 90 per cent. of the copper was obtained. Then it was decided to increase the salt content of the solution to 8.5 per cent. the percentage which will be carried in the solutions from the 2,000-ton sands-leaching plant.

Ttable I shows the data of 14 runs. In all these runs, 500 lb. of copper solution assaying 2 per cent. Cu, 8.5 per cent. NaCl, and 0.5 per cent. H_2SO_4 was used. The saturation pressure was 15 lb. and the heating pressure 20 lb. per square inch. The absorption of the SO_2 gas varied from 60 to 75 per cent. The average copper content of the tail solutions was 0.36 and the average precipitation 81.7 per cent.

The operation of this plant was highly satisfactory. The only trouble experienced with the plant proper was the failure of the heater linings. The heaters were originally lined with old 6-lb. lead, and both failed. The first one which failed, after relining with new 8-lb. lead, served for 48 heats and then showed no appreciable deterioration.

Reduction of Cuprous Chloride

There are three methods of reduction of the cuprous chloride which have been worked out to some extent and which seem capable of development to a commercial basis.

1. Precipitation with sponge iron.
2. Reduction with coke with limestone present to flux and hold the chlorine.
3. Electrolytic reduction.

TABLE I.—*Precipitation of Copper by Sulphur Dioxide.*

Run No.	19	20	21	22	23	24	25	26	27	28	29	30	31	32	Avg.
Avg. per cent. SO ₂ in head gas	8.1		9.0	7.2	9.1	8.2	7.0	9.7		8.8	7.8	8.8	11.0	8.0	8.558
Cu. ft. gas per minute.....			6.0	6.0	6.0	6.0	6.0			6.0	6.0	4.5	6.0	6.0	5.85
Gas passed, hours.....	2.5		2.5	2.8	2.5	2.25	2.5	2.5		2.5	2.25	3.0	2.25	2.5	2.504
Vol. gas passed, cu. ft.....			900.0	1,008.0	900.0	810.0	900.0			900.0	810.0	810.0	810.0	900.0	874.8
Vol. SO ₂ passed.....			81.0	78.0	82.0	66.0	63.0			79.0	63.0	71.0	89.0	72.0	74.4
Avg. heating time, minutes..	45.0	45.0	45.0	45.0	45.0	45.0	45.0	35.0	35.0	30.0	30.0	30.0	30.0	30.0	38.21
Per cent. Cu in tail solution..	0.35	0.40	0.40	0.28	0.35	0.38	0.35	0.40	0.40	0.30	0.37	0.40	0.32	0.40	0.364
Per cent. H ₂ SO ₄ in tail solution.	4.25	3.1	3.9	4.2	4.1										3.91
Per cent. precipitation of Cu.	82.0	80.0	80.0	86.0	82.5	81.0	83.0	80.0	80.0	85.0	81.0	80.0	84.0	80.0	81.75

Precipitation with Sponge Iron

Sponge iron, when mixed with cuprous chloride 40 to 50 per cent. solids, gives a rapid and complete precipitation, with the evolution of considerable heat. The reaction is, $\text{Cu}_2\text{Cl}_2 + \text{Fe} = 2\text{Cu} + \text{FeCl}_2$. It is evident from this reaction that only one-half as much iron is required to precipitate the copper from cuprous chloride as from a cupric salt. A sponge iron with a copper equivalent of 2.30 on a copper sulphate solution gave a copper equivalent of 1.20 on cuprous chloride. The cement copper from the precipitation of copper from cuprous chloride with sponge iron is rather granular, settles readily and washes easily. A sample made from sponge iron with a 1.20 copper equivalent on cuprous copper assayed 56 per cent. Cu and 0.1 per cent. Cl.

The precipitation of the copper from solutions by the SO_2 method followed by the treatment of the cuprous chloride with sponge iron make an effective combination. Concentrates may be roasted in a suitable furnace to make the required SO_2 gas, and the hot calcine drawn directly from the hoppers beneath the roasting furnace into the revolving furnace for reduction to sponge iron. In a suitable furnace the concentrates can be roasted to under 4 per cent. S and 10 per cent. gas produced. It would figure about as follows: 100 tons concentrates assaying, Cu, 5.3; insoluble, 6.4; FeO, 48.6; Fe, 37.8; S, 44.8 per cent., will roast to a calcine assaying, Cu, 8.4; insoluble, 10.6; FeO, 62.6; S, 3.7 per cent. About 40 tons sulphur or 80 tons SO_2 are available. About 60 per cent. of this SO_2 , or say 50 tons, can be absorbed in precipitation of the copper. The reaction is, $2\text{CuCl}_2 + \text{SO}_2 + 2\text{H}_2\text{O} = \text{Cu}_2\text{Cl}_2 + \text{H}_2\text{SO}_4 + 2\text{HCl}$. It is evident from this reaction that 50 tons SO_2 will precipitate about 100 tons copper.

The original concentrates contain 37.8 tons iron, of which 80 per cent. is available, for precipitation of copper, in the form of a sponge iron with a copper equivalent of 1.5 to 1.6 on copper sulphate solution and 0.8 to 0.9 on cuprous copper. Cuprous copper requires 0.44 lb. pure iron per pound of copper for precipitation. $37.8 \times 0.80 \div 0.44 = 68.7$ tons copper precipitated. This gives a comfortable margin of SO_2 for precipitation and any excess can be easily wasted. The above calculation is rather rough, because, as before mentioned, the complete data of the sponge iron process are not yet worked out, but it is probable that it is not far off.

The combination of the methods has these advantages:

1. The conversion of a large part of the SO_2 from the roasting process directly into sulphuric acid in the leaching solutions, accompanied by the precipitation of the copper in a concentrated form, from which it can be recovered by precipitation with the sponge iron resulting from the reduction of the calcine.

2. The use of only one-half as much iron to precipitate the copper from

the cuprous form as from the cupric, and hence the return of only one-half as much iron into the leaching solutions to foul them.

3. The precipitation of the copper in a concentrated condition by a simple process, in a form easily washed and handled.

4. The possibility of the treatment of the ferrous chloride which results in such a way that the valuable chlorides can be returned to the leaching solutions unaccompanied by iron.

Reduction with Coke in the Presence of Limestone

By this method the cuprous chloride is directly reduced to metallic copper, by a furnace process, with the production of a calcium chloride slag from which the chlorides may be leached and returned to the leaching process. Cuprous chloride is very volatile at high temperatures. However, by having fine limestone about 10 to 15 per cent. in excess of the theoretical requirements, in intimate mixture with the coke and cuprous chloride, it is possible to accomplish reduction with a loss of not more than 5 or 6 per cent. of the copper.

A mixture of 100 parts of slightly moist cuprous chloride, 65 parts limestone, and 10 parts coke was melted in a No. 50 graphite crucible and the melt poured into a sand mold. The button assayed 97 per cent. and the slag 0.88 per cent. Cu; 93.1 per cent. of the copper was recovered in the button and 1.4 per cent. in the slag, making a total recovery of 94.5 per cent. The slag, which carried 86 per cent. of the original chlorine in the cuprous chloride, and assayed 35 per cent. Cl, was leached for 2 hr. with warm water; 56.6 per cent. was soluble and the residue assayed 1.6 per cent. Cl. This indicated a recovery of 98.7 per cent. of the chlorine in the slag and 84.9 per cent. of the original chlorine in the cuprous chloride. This recovery was probably low, due to loss of slag in cleaning up. The insoluble, consisting mostly of lime and carrying some copper, settled readily and could be worked up in the smelter process for its copper value without trouble.

Briquettes were made of a mixture of 100 parts moist cuprous chloride, 75 parts limestone, 10 parts coke, and 5 parts cement, by hand pressure, and with a testing machine at 2,500 to 5,000 lb. per square inch, in a die 3 in. in diameter. Excellent briquettes resulted in all cases, assaying about 26 per cent. Cu. A charge of briquettes was melted down in a small reverberatory about 12 in. wide and 20 in. long, using the flame of an oil burner for heating; 94.1 per cent. of the copper was recovered, 87.6 per cent. in a button and 6.5 per cent. in the slag. A charge of briquettes was also fused in a small cupola; 93.9 per cent. of the copper was accounted for, 90.8 per cent. in the button, which assayed 96.1 per cent. Cu, and 3.1 per cent. in the slag, which assayed 1.65 per cent. Cu.

In all cases there was considerable fume given off and the flame had a

characteristic bluish color. It is very probable that with a suitable installation 95 per cent. of the copper could be recovered in metallic form and in the slag, and that 75 per cent. of the volatile copper could be caught in a Cottrell treater or a milk of lime absorption tower.

Electrolytic Reduction

The reduction of the copper from cuprous chloride by electrolysis requires only one-half the current which would be required to precipitate

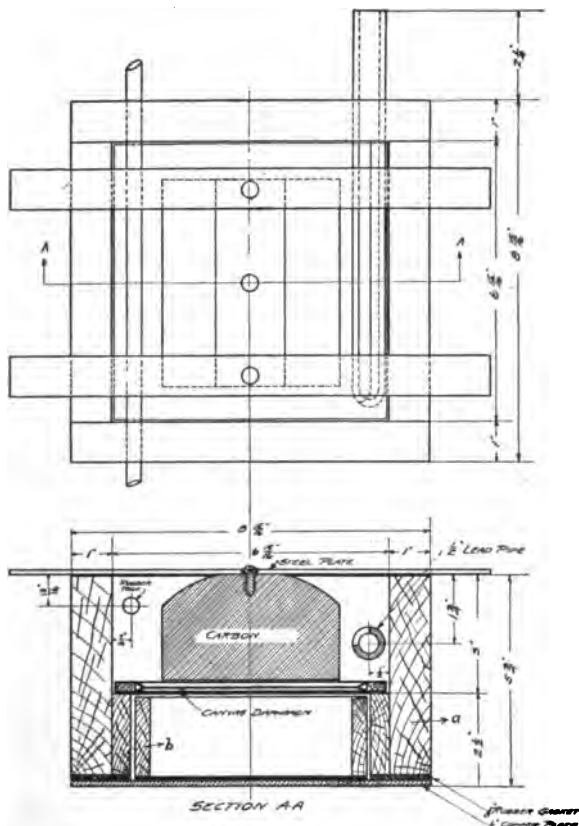


FIG. 6.—PLAN AND CROSS-SECTION OF 6-IN. REDUCTION CELL.

it from the cupric form. However, it is difficult mechanically, because the cuprous chloride is a solid and the strongly oxidizing gases given off at the anode must not come in contact with it. This makes necessary the use of a horizontal diaphragm cell and a circulating depolarizing solution. The solution resulting from the SO_2 precipitation after the cuprous chloride settles out is strongly reducing and makes an admirable depolariz-

ing solution, which on oxidation gains the equivalent of about $\frac{3}{4}$ lb. of sulphuric acid per pound of copper reduced. This makes the combination of this method with the SO_2 precipitation method very attractive. The early experiments were made in a cell 6 in. square, shown in plan and section in Fig. 6. It consisted of a wooden box (a) with a copper-plate bottom; a tray (b) with wooden sides and a sheet-copper bottom; a canvas diaphragm; a graphitized carbon anode; and intake and outlet pipes for the depolarizing solution. The cuprous chloride, filtered and washed, was mixed with about 10 per cent. NaCl , which made it thin enough so it could be readily poured into the tray. The reducing solution was circulated until there was a noticeable smell of chlorine from about the anode. As the run approached completion there was considerable evolution of gas from under the diaphragm. Part of the copper was deposited firmly adhering to the starting sheet and part in a loose form. The data of one of these runs were:

Depth of cuprous chloride, inches.....	2.0
Average amperage.....	3.32
Average voltage.....	2.67
Current density, amperes per square foot.....	19.14
Theoretical copper deposited, grams.....	537.4
Copper actually deposited:	
Adhering to starting sheet, grams.....	225.2
Loose, grams.....	229.0
Assay of copper:	
Adhering to starting sheet, per cent.....	99.56
Loose, per cent.....	97.03
Pure copper deposited, grams.....	446.4
Current efficiency, per cent.....	83.06
Power consumption per pound of copper, kw-hr.....	0.615
Volume depolarizing solution, liters.....	15.27
Average acidity of head solution, per cent. H_2SO_4	5.23
Average acidity of tail solution, per cent. H_2SO_4	6.97

A combination of three cells similar to those shown in Fig. 7 was then tried. Graphitized carbon immersed electrodes were used, the anode of one cell serving as the cathode of the one above it. Each cell had its own circulating arrangements.

The data of the run were:

Inside dimensions of trays, inches.....	1 by 4 by 4
Depth of cuprous chloride, inches.....	1.0
Total time of passing current, hours.....	36.16
Average amperage.....	2.41
Average voltage.....	7.46
Average voltage per cell.....	2.49
Current density, amperes per square foot.....	21.72
Theoretical amount of copper deposited, grams.....	620.1

Copper deposited:

Adhering to starting sheet, grams.....	256.0
Loose, grams.....	193.7

Assay of copper:

Adhering to starting sheet, per cent.....	95.18
Loose, per cent.....	94.04
Pure copper deposited, grams.....	425.8
Current efficiency, per cent.....	68.66
Kw-hr. per pound of copper.....	0.695
Volume depolarizing solution, liters.....	18.74
Avg. acidity head depolarizing solution, per cent. H_2SO_4	4.07
Avg. acidity tail depolarizing solution, per cent. H_2SO_4	5.64

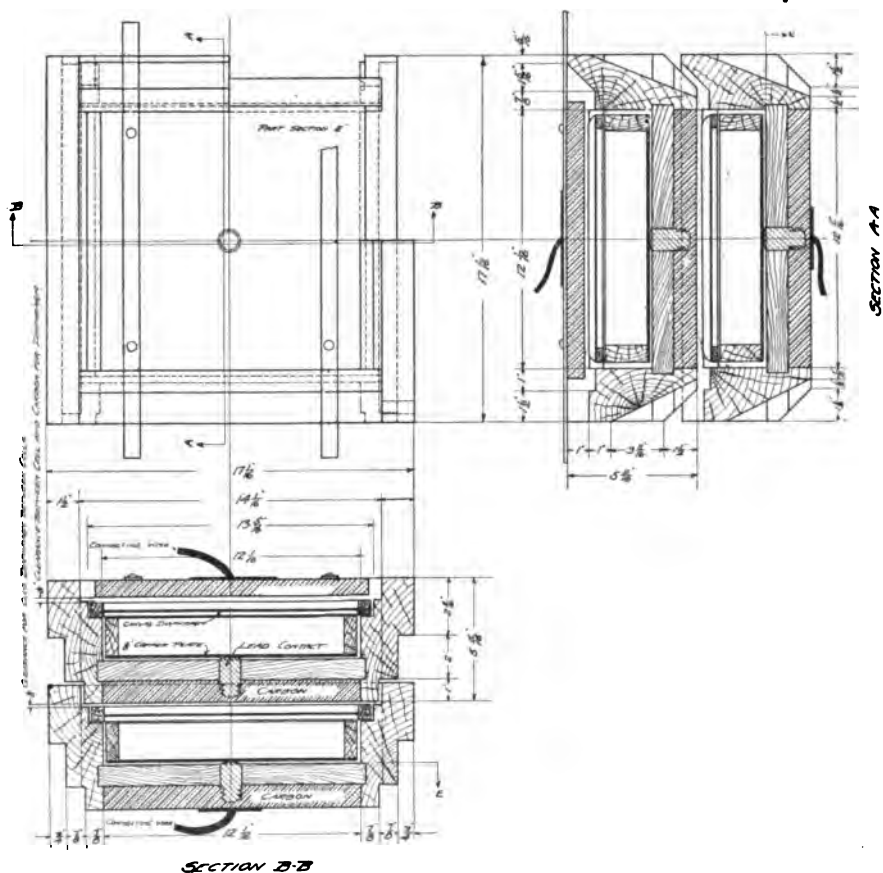


FIG. 7.—PLAN AND SECTION OF TWO ELECTROLYTIC REDUCTION CELLS OF THE 1-FT. SQ. 10-DECK INSTALLATION.

It was then decided to try the method on a larger scale. Ten cells as shown in Fig. 7 were built and set in a frame so that a cell would slide out from over the one below it, exposing the tray of the lower cell for

filling or removal. The cells were set on an incline of $\frac{3}{8}$ in. to the foot, the overflow side being the high side. This made unnecessary the use of a dam on the overflow side to bring the solution in contact with the anode surface.

As yet the cell has not been worked out to an entirely satisfactory basis, but it seems capable of development.

Conclusion

At present it seems advisable to precipitate the copper directly from the copper solutions of the 2,000-ton sands-leaching plant with sponge iron. The precipitation of the copper from such solutions with sponge iron is being worked out and promises to be simple. Until the leaching process is established on a definite operating basis it does not seem advisable to introduce a complicated precipitation method.

The acid plant now building will have ample capacity for the first 2,000-ton leaching unit. When the time comes to build further units, the installation of an SO_2 precipitation plant will depend on considerations of first cost of acid and precipitation plants, cost of operation and other factors which undoubtedly will develop.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Smelting Lead Ores in the Blast Furnace

BY IRVING A. PALMER, EASTON, PA.

(Salt Lake Meeting, August, 1914)

DURING the past 15 years in this country there have been few additions to the literature of lead smelting. After the consolidation of the principal smelting companies at the beginning of this period it became the policy of the larger company not to permit the publication of details in regard to many of its metallurgical processes. This led to similar action on the part of some of its competitors. The result, of course, was a decided decrease in the number of papers written upon the practical side of the subject. Even upon the theoretical side the contributions have been few. It would seem that the industry as a whole has suffered from this general unwillingness to exchange information. While it is true that a given company is entitled to the reward that may come from its own development of a new process, it is also true that a still greater reward might be the result of a free exchange of ideas with other plants. This principle has been recognized for a long time in other metallurgical industries, notably those engaged in the smelting of iron and the manufacture of steel. It is pleasing to note that the leading lead-smelting company has very recently decided to modify its policy of secrecy, and to permit the publication of papers covering some details of its metallurgical methods.

The statement has been made that there has been but little progress in lead smelting during the period just referred to. This impression is due in part to the scarcity of papers written on the subject. There has been, it is true, some discussion of the new sintering processes, but with little reference to detail, and to the effect that blast roasting has had upon the industry as a whole. In spite of the prevailing belief, it may be said that lead smelting, in common with other metallurgical industries, has made steady and substantial progress.

It is the purpose of the present paper to discuss briefly some features of the industry as it is now conducted, with special reference to the blast-furnace practice in the larger silver-lead smelting plants of the United States and Mexico.

It is generally conceded that the greatest advance made in lead smelting in recent years has been effected by the introduction of the various sinter-roasting processes. The past decade has witnessed the practical abandonment of hand roasting, and of mechanical roasting in which the product is not eventually sintered. As will be shown later, this improvement has greatly increased the efficiency of the blast furnace, both as regards tonnage and as to the character of the work done.

Second in importance has been the general adoption of mechanical methods in the handling of materials. The labor problem has always been a serious one in the industry. Owing to the comparatively small tonnages, the great variety of ores, the necessity of careful bedding and mixing, and the delicacy of the various metallurgical processes, it has never been possible to do away with hand labor to the extent that it has been accomplished in iron and copper smelting. But a great deal has been done in this direction, and further improvements are being considered.

The agitation in many localities against smelter smoke, as well as a more careful study of fume losses, has led to the construction of bag houses at a number of smelting and refining plants. These bag houses have not only effected a saving of values, but have thrown considerable light upon the whole subject of metal losses. They have added new problems as well, because of the large amount of volatile by-products collected by them.

There has been, also, an increase in the size of the blast furnaces and of other equipment employed in lead smelting. The increase has been moderate, however, and there have been no radical changes, either in size or in general arrangement. In this connection it may be said that the opportunities for experimenting along these lines by the average metallurgist are limited. Most companies are unwilling to stand the expense of conducting experiments on a large scale. Even when the money is forthcoming the difficulties in the way of testing properly a large blast furnace, for example, are great. It has been the general practice to increase the size of blast furnaces at new plants, or at plants that are being rebuilt, and in these cases the other equipment can be constructed to conform.

Let us now consider some of the details of present-day practice:

The blast furnace is the central feature of a smelting plant. The buildings and equipment should be constructed with the idea of serving the blast furnaces economically and efficiently. That this is not the case in some plants is a well-known fact, and is partly responsible for their reputation for conservatism.

As intimated above, there has not been much change in the construction of the lead blast furnace in recent years. At the present time the maximum length of the furnaces that are used in this country is 200 in. at the tuyère level, and the maximum width, 48 in. At most of the large

smelting plants the length varies from 136 to 160 in., and the width, from 42 to 46 in. So extreme a width has not been found in some foreign countries, as in the United States. A width of less than 48 in. is preferred by some because of their belief that hotter slags are produced.

The angle of the bosh has been the subject of considerable controversy, as well as of experimenting. With the furnace charge constituted as at present at most plants, the weight of experience is decidedly in favor of a moderate bosh. At a plant with which the writer is familiar the furnaces were built with a 20° bosh, so that the width at the top of the 6-ft. jackets was 84 in., or 36 in. greater than at the tuyère level. Whenever a furnace was blown in, and for some time thereafter, it was noticed that there was a tendency for the charge to hang at the bosh, and to descend irregularly. The assays of the slags and mattes were usually higher than those of the furnaces longer in blast. After a week or two the furnace work would begin to improve, and later would become normal. On blowing out a furnace, after a campaign, it was almost invariably observed that the space on each side of the furnace above the bosh had been filled in with partly-fused charge, changing the furnace lines so as to form a gently flaring shaft 60 in. wide at the top of the jacket, or with a flare of about 1 in. to the foot on each side. The furnace had thus accommodated itself to the descending charge, and, as stated, better work was done as a consequence. The extra width of the furnace above the bosh was not only useless, but detrimental. With the furnace charges as they are now prepared and until the fusion zone is reached there is but little contraction in volume as they descend in the shaft. There is also a much more rapid movement of the charge. In addition, the narrower shaft insures a better distribution of the ingredients of the charge, and as a rule produces hotter slags.

The tuyères are usually spaced 16 in. apart from center to center and have openings varying from $2\frac{1}{2}$ to 4 in. in diameter. On this point of tuyère area there is considerable difference of opinion. With coarse charges, wide furnaces, and the comparatively low blast pressures now used, there is a tendency to employ a smaller tuyère opening, in order to give a better penetration to the blast. With a tight charge and high blast pressure the question of tuyère area is not so important.

The proper height of the charge column is a matter that depends very largely upon the physical character of the materials of which it is composed. There has been a reaction from the low column formerly used, because of the flue-dust and fume losses, and of the excessive speed, which often results in fowl slags and mattes. On the other hand, a very high column has been found to decrease the furnace speed and to increase the tendency for the fire to creep up in the shaft. In the large furnaces now in use the charge is usually maintained at a height of from 15 to 18 ft.

The water jackets are now commonly made of soft steel. While

their cost is greater than that of cast-iron jackets, they last longer and can be easily repaired when damaged. To avoid burning, caused by the accumulation of mud and scale in the bottom of the jackets, they are washed out at regular intervals. In a late improvement the bottom of the jacket, instead of being horizontal, is inclined at an angle toward the outside of the furnace, thus throwing the scale and mud away from the fire sheet. Burned jackets are often the result of irregularities in the furnace work.

At the larger plants hand feeding of the furnaces has been generally superseded by mechanical feeding; that is, the charge is dropped from a car which is run over the furnace top. The car discharges either from the center or from each side. In either case the furnace is provided with what are known as "spreaders." These spreaders serve to throw the charge toward the sides of the furnace, thus distributing a large proportion of the fine ore along the sides, while the coarser parts of the charge roll toward the middle line of the furnace. The natural tendency of the blast being to follow the sides of the shaft, it is counteracted in this way. There are few things in lead smelting that have given the metallurgist more concern than this point of securing a proper distribution of the charge in the furnace. It is difficult to secure the right adjustment. Often it can be obtained only after repeated trials. To get the best results there must be taken into consideration the character of the charge, the size and shape of the furnace, the height of the charge column, and the construction of the car. It is one of the proofs of the sensitive nature of lead smelting that a slight modification of the spreading devices often affects profoundly the tonnage and efficiency of the furnace. Improper feeding often neutralizes the effect of good work in all the other departments of the plant.

The preparation of the furnace charge is an important feature in the operation of a lead-smelting plant. Every lead metallurgist knows that the efficiency of the blast furnace is very largely dependent upon the physical character of the charge. It is a difficult problem to prepare a charge that will smelt rapidly and also produce clean slags and mattes, and that is the reason why it is so difficult to introduce satisfactory methods for the mechanical handling of ores and by-products. It is this that makes lead smelting so widely different from iron and copper smelting.

The elimination of the greater portion of the fines from the furnace charge has long been considered one of the essentials of good work. In order to avoid the evil effects of a tight charge it was formerly the custom at some plants to screen the oxidized ores, and then briquette the fine with flue dust, slime, concentrate, and fine roasted ore from the mechanical roasters. The product of the hand roasters ordinarily was not briquetted. But briquetting is an expensive process, and the briquettes made were often so friable that they fell to powder in the upper part of the blast

furnace. However, something had to be done with the excessive amount of fine material that had to be smelted, and briquetting was considered a necessary evil. The practice undoubtedly increased the furnace speed and efficiency, but it was often carried to excess.

As noted at the beginning of this paper, probably the greatest advance made in lead smelting in recent years has been the general introduction of blast-roasting processes, such as the Huntington-Heberlein and the Dwight-Lloyd. In these processes roasting and sintering are accomplished in one operation, giving a product that greatly increases the capacity and efficiency of the blast furnace in which it is smelted. Sintering pots and machines have largely replaced the old-fashioned roasters and the briquetting presses. The common practice is to sinter a mixture consisting of oxidized fines, siliceous sulphides, concentrate, slime, and pre-roasted ores high in sulphur. Almost any fine ore can be used, provided the sulphur of the entire mixture does not exceed or fall below a certain figure, which usually varies from 12 to 14 per cent. Ordinarily only high-sulphur ores and lead mattes are subjected to a pre-roasting. In the latest practice this is done in Wedge mechanical roasters, in which a large tonnage can be roasted to almost any desired degree of oxidation, using practically no fuel. The sintering processes enable the lead metallurgist to handle cheaply and satisfactorily nearly all the classes of fine material with which he has to deal, except flue dust. This latter product usually interferes with the sintering process, and, in the opinion of the writer, should be briquetted. If a smelting plant be properly conducted the amount of flue dust made will be small. By using a mixture of roaster and blast-furnace flue dust usually a self-binding material can be obtained. It is also sometimes advisable to briquette high-grade concentrate, rather than to subject it to the sintering process. It is hardly necessary to condemn the practice of simply wetting flue dust and then charging it into the blast furnace. Yet this was done only a few years ago at a large plant in Mexico.

The advantages of the sintered ore as a material for the blast furnaces are to be found in its coarseness, its porosity, and in its similarity in composition to the general average of the charge. The latter is an important point. It is often held that the detrimental effects of fines on the charge are largely mechanical, such as decreasing the furnace speed, increasing the amount of flue dust made, and causing the formation of accretions and blow-holes. As a matter of fact, one of the most serious objections to fines is that they cause irregularities in the formation of the slag and matte. The fines of a charge naturally precede the coarse material in getting to the lower part of the furnace. This is due both to the natural fall through the openings in the charge and to the jiggling effect of the blast. This jiggling action is sometimes considerable. When the first furnace of one of the Mexican smelting

plants was blown in a few years ago the blast was supplied by a blower delivering 300 cu. ft. of air per revolution. Owing to the size of the blower it was run much below its normal speed, causing the blast to be delivered in slow pulsations. The furnace charge descended so irregularly that a satisfactory slag could not be made, and the metal losses were high. When more furnaces were blown in and the speed of the blower increased the furnace work rapidly improved.

The metallurgist computes the slag and matte composition from an analysis of the entire charge. In order that the slag and matte may be formed quickly and uniformly it is necessary that all of the constituents of the charge should be well mixed and arrive at the fusion zone of the furnace at approximately the same time. If they do not arrive at the same time it is evident that at any given moment some ingredient of the slag and matte to be produced will be either too much or too little. As nearly as possible all of the reactions involved in the slag and matte formation should take place simultaneously. The heat due to these reactions is thus evolved in the shortest possible time. Irregularities are almost sure to occur when the fines of the charge differ greatly in composition from the rest of the mixture, as when lead carbonates and sulphides, raw iron sulphides, and limonite ore are used as flux. Finely divided lead ore is particularly bad; much of it is imperfectly reduced and thus goes in part into the matte and slag. On the other hand, a portion of the lead carbonate fine is reduced too soon, and the upward rush of the blast carries small particles of metal to the upper part of the furnace, when they again combine with oxygen or sulphur, forming accretions, or going into the dust chamber. Shaft accretions always contain a large proportion of lead sulphide, even though the charge be almost free from galena. If the fines consist largely of lead and zinc sulphides the reduction is unsatisfactory. Zinc sulphide is considered to be about the worst mineral with which the lead metallurgist has to deal. It is the only sulphide, ordinarily met with, that does not alloy in all proportions with the other sulphides found in lead mattes. When present in considerable quantity it floats on top of the matte, forming a pasty scum and preventing a good separation of the matte and slag. Poor roasting is almost invariably followed by poor work in the blast furnace. Insufficiently roasted ore usually contains an undue proportion of lead and zinc sulphides, owing to the fact that, in any roasting process, these two sulphides are the last to be oxidized. The writer is familiar with a case where a pyrite concentrate containing a large percentage of zinc was subjected to a preliminary roasting in a Wedge furnace sufficient to oxidize nearly all of the iron. An analysis of the product showed that over 90 per cent. of the zinc contents remained as sulphide. Tests have shown that even in well-roasted Dwight-Lloyd sinter, low in zinc, nearly half of the zinc is present as sulphide. As the fine produced in the sintering processes contain most

of the unaltered sulphides, it can be seen how important it is to keep it as low as possible.

The reduction of lead and silver from sulphides is effected mainly by metallic iron and by the well-known reactions between oxides and sulphides, and between sulphides and sulphates. The direct reducing action of the coke is small, except in the case of carbonates. It is evident, therefore, that to obtain the best reducing action the lead and silver bearing portion of the charge must be in close proximity to all of the other principal ingredients. At one of the Mexican smelting plants some years ago the furnace charges were made up largely of coarse siliceous ores, coarse lime rock and limy ores, and iron-lead carbonates containing a large percentage of fines. Blast-furnace flue dust, and raw lead concentrates and pyrite ores were frequently added. The smelter had no sintering plant and no briquetting press. The charge as a whole was very open, as was indicated by the low blast pressure for a given volume of air. Yet the tonnage smelted was low and the lead losses high. The furnaces ran irregularly and there were continual crucible troubles. The fine of the lead and iron ore, the concentrate and the flue dust, sifting down through the charge, were the first to reach the fusion zone, thus destroying the adjustment of conditions necessary to good work. It was found that by crushing the lime ores and adding siliceous fines to the charge a material improvement in the results could be effected. In Pueblo, Colo., at one time it was a common occurrence for finely divided Cripple Creek ore to pass unfused into the slag. The fine was refractory and fed in large quantity. The furnace tonnage was low. It should be stated that careful assays showed that the gold tellurides had been thoroughly leached out of the unsmelted ore.

The best illustration of the disadvantage of using fines is to be found in the present practice in matte concentration. Formerly it was the custom at all plants to calcine the first matte in hand roasters. The product was rather fine, although pretty free from dust. The charge for concentrating this to a high-grade matte was prepared by mixing the roasted matte with coarse siliceous ore and coarse lime rock. Chemically considered, the charge could not have been better, yet the furnace work was poor. The tonnage was usually low, and the mattes and slags high in lead. There was a considerable reduction of metallic copper, with resulting crucible troubles. Often no attempt was made to keep the lead-well open, and the lead was run out with the slag and matte. Sometimes the anomaly would be presented of a considerable formation of speiss, as indicated by the assays of the slag and matte, in spite of poor reduction. The fine roasted matte descended irregularly through the charge and was partly reduced before it could react properly with the silica and lime rock. The present practice in matte concentration at the more advanced smelters is to pre-roast the matte and then sinter it with

siliceous fines. This gives a coarse, fairly neutral product which with the lime rock and siliceous ore descends uniformly in the furnace and causes no segregation at the fusion zone. The result is that the speed and reduction of the matte-concentration furnaces are often better than is the case with the furnaces smelting ore. Using sintered matte the writer has seen concentration furnaces run steadily month after month, producing clean matte and slag, making but little speiss and metallic copper, and having no crucible trouble.

There must be a certain amount of fine material on the charge, otherwise this is so open as to cause too rapid smelting and insufficient reduction. The fines should be of the same general composition as the rest of the charge. Segregation of coarse and fine material would not then result in irregularity. Sufficient fines for this purpose are usually obtained in connection with the various sintering processes.

The larger pieces of the different ingredients of the charge should be crushed to a fairly uniform size. Oxidized ores are now ordinarily sampled mechanically, and according with previous crushing. Lime rock, sintered ore, and return slag are often fed in larger sizes, but better results are secured by moderate crushing.

It is also now recognized that thorough mixing of the charge is conducive to the best furnace work. In the old days stress was sometimes placed upon the necessity of feeding the ore, flux, fine, and slag in layers. With the mechanical feeding now in general use such a practice would be impossible, even were it desirable. The better the mixing the more rapid and efficient will be the smelting. This principle has never been questioned in making a crucible fusion in the assay furnace, and the blast furnace is simply a crucible on a large scale. The reactions taking place in it are practically the same as those occurring in the crucible. In the blast furnace we attempt to correct the disturbing effect of the moving charge column and the upward rush of the blast by adopting the methods of preparation which have been described.

The use of sintered ore in the lead blast furnace has proved so successful that at least one plant, that of the International Smelting & Refining Co., at Tooele, Utah, has adopted the plan of sintering a large proportion of the total blast-furnace charge. The tonnage of the blast furnaces is high, but little information as to the metal recoveries has been published.

Dwight-Lloyd sintering machines are used. These machines undoubtedly represent the most advanced type of sintering apparatus. It has been urged against them that they are extremely sensitive. That is true; but it is also true of the blast furnace. In both cases the work is continuous and there must be a constant adjustment of the conditions necessary to good results. In the sintering process the sulphur must be maintained within rather narrow limits. In addition, the crushing, mixing, moistening, and feeding of the charge demand close attention. The

working out of the mechanical details in connection with the process has been a difficult problem, and no doubt further improvements will be made. Most of the inferior results ascribed to the Dwight-Lloyd process can be attributed to the defective supervision and to irregularities in the feed.

To offset the troubles caused by frozen ore and concentrates various methods for thawing have been devised. In severe climates this becomes quite a problem, and much remains to be done before it is satisfactorily worked out.

The chemical composition of the slags produced has always been an important point in lead smelting. In recent years, however, the views of metallurgists in general upon the subject have been considerably modified. Experience has shown that many of the irregularities in furnace work, formerly thought to be due to the composition of the slag, are the results of other causes. Laboratory experiments have thrown some light upon the constitution of lead slags. It is now known that these slags, instead of being definite chemical compounds, are in reality mixtures of a number of compounds and oxides, probably existing in a different state of combination while in the molten condition than when solidified. These compounds can exist in a great variety of proportions without materially changing the fusibility or fluidity of the slag. It is recognized, also, that certain oxides, such as those of aluminum and zinc, may not be combined with silica at all, but may be in igneous solution in the slag.

The physical properties of the slag are dependent upon the number and kind of its various components, but, within reasonable limits, there is not so much variation as was formerly supposed. The present-day practice is to place more emphasis upon the questions of temperature and fluidity. No slag, no matter what its composition, will work well if it does not flow freely from the furnace and does not remain liquid long enough to permit of a good separation of the slag and matte. In many cases this is simply a question of heat. With the larger furnaces and the improved methods of preparing the charge now in use hotter slags are produced. This has led to the employment of slags formerly considered to be too refractory for lead-furnace work. The slags still conform generally to the singulo-silicate type, but often there is a considerable proportion of bisilicate. In selecting the proper slag the metallurgist is guided very largely by commercial reasons. In some localities, for instance, where there is a surplus of siliceous ores, he will make a slag high in silica, so as to smelt more of the ore which is most abundant. So also he will generally use as much lime as possible, because it is a cheaper flux than iron. In this connection it may be said that high-lime slags are more in use than they were in times past. The higher temperatures employed and the lower zinc contents of the present-day charges have made this possible. Of course it is generally held, also, that high-

lime slags are beneficial metallurgically. The old idea that lime and iron oxide should exist in the slag in certain definite molecular proportions has been discarded by most metallurgists. This so-called "type theory" was supported for a long time, and was considered by some to be extremely essential. Theory and practice have combined, however, to indicate that the things to be considered in adding lime to a charge are the question of cost, and the percentages of silica, alumina, zinc, and magnesia present, rather than the ratio of the lime to the iron.

Ordinarily all the lime possible that will still permit the slag to flow readily will be used.

Lead slags are now regularly made carrying from 35 to 37 per cent. of silica and from 24 to 26 per cent. of lime, including magnesia and barium oxide figured as lime. In these slags the zinc and alumina are generally low. High-silica slags are usually considered as being liable to contain a rather high percentage of lead. To some extent this is true, but such a slag runs well, chills slowly and keeps the furnace in good condition. It is conducive to long campaigns. Furnace accretions are always basic, and high-silica charges discourage their formation.

A discussion of slag composition naturally leads to a consideration of the subject of metal losses. With the introduction of bag houses, better methods of sampling, and the general use of the wet assay for lead, a much closer check can now be made upon the efficiency of a smelting plant. These refinements have shown that in most cases the metal losses are greater than they were formerly supposed to be. They have led the metallurgist, also, to modify his views somewhat as to what constitutes a good, economical slag. Twenty years ago it was usually the aim of the lead metallurgists to produce slags and mattes containing the lowest possible percentages of the valuable metals. Comparatively little attention was paid to any other feature. The evils of over-reduction were recognized by some, but not very clearly. The character of the work done was judged mainly by the slag and matte assays. About ten years ago a large smelting plant in Colorado was noted for the extremely clean slags which it was producing. The average lead content of the waste slag for one year was 0.67 per cent. The mattes also were very low in lead. The metallurgist congratulated himself upon the figures. There was no bag house, but careful determinations of the lead smelted and recovered for the period showed that the lead losses really were very high. This was confirmed later by analyses of the flue gases. The furnace tonnage was low. At another plant, operating upon the same class of ores, the speed of the furnaces was greatly increased by screening most of the oxidized ores and briquetting the fine. The slags and mattes were unusually clean. A low-silica slag was made, and a considerable production of speiss showed a condition of very strong reduction all the time. Yet the lead losses were extremely high, as shown by the balance sheets at the end

of each month. The work not improving, experiments were begun for the purpose of determining the flue-dust and volatilization losses. The dust at the foot of the blast-furnace stack was found to contain 60 per cent. of lead, and analyses of the gases passing out of the stack showed where the lead was going. The furnaces were low and a high blast pressure was maintained to force the tonnage. Some improvement was effected by lowering the blast pressure and tightening the charge. These illustrations show that the slag and matte assays are not the only points to be considered in lead smelting.

The question of flue dust and fume has always been a difficult one for the lead metallurgist. While bag houses are usually effective, they are expensive to build and to operate. Often they show practically no profit. Collecting, as they do, all of the volatile solids, such as arsenic, cadmium, selenium, and tellurium, they have introduced new problems into the industry. In the latest practice roaster gases are neutralized and filtered also, thus increasing the sulphur and zinc contents of the bag-house fume. The epigram of Anton Eilers, enunciated years ago, that "the best way to handle flue dust is not to make any," still holds good as the ideal toward which the metallurgist should strive. Modern practice has succeeded in reducing the flue-dust and fume losses very materially. This has been the result largely of the changes in methods that have been referred to.

At Midvale, Utah, the blast-furnace bag-house dust formerly carried from 40 to 55 per cent. of lead. This has been cut to an average of under 20 per cent., on a smaller amount of dust. In the meantime the arsenic has risen to about 40 per cent. The arsenic is eliminated almost entirely through the flue as sulphide. Practically no speiss is made and the bullion is remarkably free from arsenic, even though the percentage of that element is high in the charge. As this smelter is equipped with an arsenic plant the large percentage of arsenic in the fume is no disadvantage. Where there are no facilities for collecting and refining arsenic it simply forms one more undesirable product of the bag house.

The formation of arsenical speiss is an evil. It consumes iron, reduces the furnace speed, increases the volatilization of lead, causes trouble in the crucible and settling boxes, and constitutes a troublesome by-product which has to be rehandled. It is the accompaniment of low-silica slags, low sulphur in the charge, and powerful reducing action. Its production can be controlled in almost every case.

The question of coke consumption in the lead blast furnace has been the cause of some controversy. The literature of the subject is vague and often misleading. It is usually stated that a certain percentage of coke is used under certain conditions. No information is given as to the chemical and physical characteristics of the coke. Every metallurgist knows how serious are the troubles caused by irregularities in the grade

of the coke. With the charges as now prepared, the main function of the fuel is to produce heat. The reducing action, except of lead carbonates and iron oxides, is largely secondary. But there are other sources of heat in the charge. The oxidation of sulphur and arsenic, the combination of metallic iron with sulphur, arsenic, and oxygen, and the energetic reactions which take place in the formation of the matte and slag all produce heat. The amount of coke to be added, therefore, depends to some extent upon the heat-producing power of the charge itself. In addition, there must be considered the chemical and physical constitution of the coke, the physical condition of the charge, the temperature of the atmosphere, the amount of moisture in the charge and in the air, the condition of the furnace, and the composition of the slag. There must be sufficient fuel to produce a hot, easily running slag. Experiments have shown that considerable lead and silver can be separated from some slags by simply subjecting them to a very high temperature. This causes a still more complete series of reactions between the various oxides, sulphides, and silicates in the slag. With a good charge and with good coke, an amount equivalent to 10 per cent. of fixed carbon, as figured against the weight of the charge (less slag and fuel), may be considered the minimum that can be used. Sometimes less is sufficient. Often it will be necessary to use more, up to $11\frac{1}{2}$ or 12 per cent. In any case the metallurgist must be guided by conditions, and vary the amount of coke as he would lime rock or any other item.

The excessive amount of fine ore formerly smelted led to the practice of wetting the charge before putting it into the blast furnace. As in the sintering processes, this moisture renders fine ore porous by preventing it from packing. It keeps the furnace top cool and reduces the amount of flue dust made. In slow running there is always a tendency for the heat to creep up in the furnace. This is counteracted by the water in the charge. Owing to the fineness of the ore particles and the slowness with which they descend, the charge is perfectly dried by the waste heat of the furnace before it reaches the fusion zone. Experience has shown that with fine ore a wet charge can be smelted with less coke than a dry one. With the coarse charges and rapid running which is now the practice there is need for but little water. The pieces of ore are so large and descend so rapidly that if they are saturated with moisture they may not be dried out in time, and thus may chill the furnace. With a coarse charge and fast running, also, there is little tendency for the heat to creep up, and little flue dust is made. Sufficient water should be added, however, to moisten the fine. I have known a heavy rain or snow storm to be followed by cold slags and poor reduction at the blast furnaces. To remedy this condition it is necessary to add more coke.

The volume and the pressure of blast required vary so much that it is hard to give any definite figures. In general it may be stated that

the pressures now used are not so high as they were when tight charges were the rule, but comparatively high pressures are still used to increase the furnace tonnage, when the other conditions are such as to permit it. The air pressure should be kept as constant as possible, and should come from the power house in rapid pulsations. At most of the larger plants the blast pressures used range from 30 to 40 oz. per square inch.

To produce a sufficiently heavy matte fall, coarse iron pyrite, low in zinc, is often added to the charge. This usually raises the slag and matte temperatures and facilitates the elimination of arsenic. As at present conducted there is often considerable pyritic action in the lead blast furnace, with a large production of sulphur dioxide.

Much time and money have been devoted to the problem of a proper separation of the matte and slag. Settling furnaces have been devised on the reverberatory principle, using fuel to keep the slag hot and fluid. Most of these have been abandoned. Usually there has been trouble on account of zinc, which gradually accumulates in the furnace. The volatilization losses have been high and there has been also the extra cost of the fuel used. In most cases the slag and matte are separated in the fore hearth. This requires considerable attention and must be changed frequently. It is effective, however, when the furnaces are working properly. Experiments have shown that prolonged settling is unnecessary in the case of hot, properly made slags. It is now known that slags have a slightly solvent action for the sulphides of silver, lead, and copper. Beyond a certain point, therefore, these sulphides cannot be settled out. An earlier recognition of this fact would have saved much time and money.

In the foregoing review of the subject only the more important features in the modern practice of lead smelting have been considered. It is impossible to refer to many important details. In the industry there are many factors, the right adjustment of which is necessary to success. As already intimated, the lead blast furnace is sensitive to adverse conditions. In a smelting plant the ores and fluxes received are continually changing, and the metallurgist finds that he must accommodate himself to a new situation almost daily. Usually there is a shortage or a surplus of some important ingredient of a good charge. At the silver-lead plants there is nearly always a deficiency in the supply of lead, and an excessive amount of sulphides. Often the metallurgist puts a charge on his furnaces that he knows will not give the best results. But he must keep his furnaces going, and he smelts what he has.

As the object of smelting is to make money, the commercial side of the industry must be kept uppermost. Apparently poor metallurgical work may mean the greatest financial success. As between processes, that one will be selected which gives the largest returns in money. At most plants there are local considerations that may modify what is generally held to

be the best practice. In hardly any other metallurgical industry is it so difficult to lay down general rules. But the underlying principles have been worked out and the conditions that are essential to good practice are known. That the results obtained are often not what they should be is due to the fact that these conditions could not be realized.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

The Appraisal of Coal Land for Taxation

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(Salt Lake Meeting, August, 1914)

WITHIN the last 10 years the subject of mine taxation in its relation to coal-mining interests has come to have growing importance, not only to those engaged in the mining of coal, but also to the owners of land underlain by coal. A disposition has developed among legislators to accept the principle, which has been laid down by many judicial decisions, that coal underlying the surface must be regarded as a part of the real value of all properties so underlain, and as such is chargeable with such burdens of taxation as the necessities of the government may impose. These necessities arise from the normal expense of maintaining State and local governments, and in the past have been met principally by taxes based upon assessments of real estate values, which values in many cases have been confined principally or wholly to the value of the surface, without taking into consideration the value of minerals underlying the surface.

It is not proposed here to deal with the question of Federal taxation, whether levied on production, or on products transported from one State to another, or exported from the country, or upon the gross or net earnings of corporations engaged in interstate commerce, or upon the production of materials which may be transported from one State to another, or levied upon the profits or upon the obligations of corporations. The present discussion will be confined to the appraisal of coal lands as assessed for State, county, township, or borough taxation.

The methods heretofore and at present in use in the different States vary widely and have resulted in more or less litigation. Discussions appearing from time to time in the technical and trade journals and in the proceedings of mining and engineering associations frequently direct attention to the need for the adoption of fixed principle in levying taxes, and of uniform methods for applying such principles, but there does not seem to be a general demand from the public for the adoption of such methods.

The greatest burden of taxation must always be from taxes levied to meet local requirements, because the maintenance of schools, of the poor, of roads and bridges, of local courts, of prisons and hospitals, of the

police, of roads and highways and their lighting, and of sanitary supervision, will always cost far more per capita than the cost of maintaining the Federal or State governments.

As taxes for local purposes are collected under authority conferred by State legislation, and as the constitutions of the several States contain provisions concerning the rights of persons and property which require uniformity in methods of taxation, and as the courts have generally affirmed the principle that coal, if of material or determinable value, must be regarded as assessable and taxable property, differing in no respect from other forms of property, this principle is supposed to underlie the methods in general use.

In connection with matters relating to the application of this principle in the determination of taxable values, an investigation was made recently to learn as fully as possible the details of present practice in the States in which coal mining is an important industry. This inquiry resulted in the compilation of a large mass of data, giving perhaps a better insight into these conditions than anything that has yet been available. It disclosed the fact that few of the States have definitely adopted the principles upon which mineral (or coal) land shall be appraised for the purposes of taxation, and that few States have uniform methods applying to all parts of the State for fixing such values.

The investigation upon which this article is based included the principal coal-mining portions of Alabama, Arkansas, Colorado, Illinois, Iowa, Kansas, Kentucky, Missouri, Montana, Ohio, Pennsylvania, Tennessee, Texas, Utah, Virginia, West Virginia, and Wyoming. It involved correspondence with about 350 State, county, and township officials engaged directly or indirectly in the assessment of the value of coal land for taxation, and includes more or less useful and authoritative information obtained from about 120 of these officials.

While the inquiry disclosed the fact that uniformity in making such assessments is in practice almost unknown, entirely different methods often being used in adjoining townships in the same county, or in adjoining counties in the same State, it did, however, clearly indicate that in nearly all of the coal-mining States certain broad principles were recognized, and these tend to establish a basis to which future practice will doubtless be made to conform. While these principles, however, are theoretically assumed as governing factors in fixing the basis for assessments, the methods by which they are applied, or are attempted or supposed to be applied, present extraordinary variations and inconsistencies. This condition, in some cases, has led to attempts to remedy such inconsistencies by the creation of State commissions, or boards, or courts, charged with the duty of equalization.

The principles that appear to have received most general recognition, and the exceptions to the same, are as follows:

1. That the coal, or coal-mining right, when not owned by the owner of the surface, shall be assessed as the property of the individual or corporation claiming ownership. Exceptions to this rule are found in Kansas, Kentucky, Iowa, Illinois, Missouri, Michigan, Utah, Wyoming, and possibly some other States, such exceptions generally applying to localities in which the coal, through lack of knowledge as to its thickness, quality, etc., has no definable or appraisable value. This principle has been affirmed and reaffirmed by judicial decisions in many States.

2. When the coal and surface are owned by the same individual or corporation, the assessed value is usually fixed at a sum per acre supposed, intended, or assumed to cover the value of both coal and surface; assessors rarely attempting to assess the value of the coal and surface separately. The exceptions to this practice are found principally in those States and districts in which coal has the largest values; examples of such exceptions are found in Pennsylvania, Kentucky, Ohio, Virginia, and Wyoming.

3. Coal, or the coal-mining right, is generally assumed to be assessable at a value representing the price obtainable at voluntary sale, or at a definite percentage (ranging all the way from 20 up to 80 per cent.) of such value. While the legal validity of this principle has been sustained by many judicial rulings and it is presumed, or assumed, to be applied in many States, exceptions are found almost everywhere. Some of the Western States have substituted for this principle, methods of fixing the taxable value based partly or wholly upon the value of the output, or upon the net or gross returns, or upon an arbitrarily assumed value per ton of such output. Thus in Wyoming the value of the output (fixed at from 60c. to 95c. per ton) determines the assessable value. Exceptions to the application of this principle are found in Alabama, Arkansas, Illinois, Iowa, Kansas, Kentucky, Ohio, Pennsylvania, Tennessee, Virginia, and West Virginia, the exceptions being most numerous in localities where the owners of the surface (farmers) still own the coal. In such localities the owners of the coal, generally being anxious to sell it, and believing that high assessments of value will tend to retard sales, usually succeed in keeping the assessed values far below the price obtainable at voluntary sale.

4. Great differences exist both as to principle and practice in assessing the value of coal leaseholds; that is, the equity of a lessee in the coal which he controls under a lease. The equity of a lessee in the coal is not assessed as taxable property in Alabama, Arkansas, Colorado, Kansas, Missouri, Utah, and Virginia. Such equity may be assessed as taxable property in Illinois, Iowa, Kentucky, Tennessee, Ohio, Pennsylvania, West Virginia, and Wyoming, but the practice is not uniform throughout any one of these States; in the aggregate only a comparatively small number of leaseholds being assessed for taxation. The whole value of

coal held under lease is usually assessed to the owner of the property, the equity of the lessee being disregarded. This practice, in many cases, doubtless originates in the difficulty experienced by assessors in attempting to determine the value of the equity of the lessee, and also because the assessors assume that as the coal value (in such case perhaps being determined by the royalty per ton paid by the lessee) is assessed to the owner of the leased premises, and the value of the improvements owned by lessee is assessed to the lessee, the operation as a whole is thus bearing its full share of the burdens of taxation. Cases in which the terms of the lease provide that any increase in the taxes assessed after the execution of the lease shall be borne by the lessee, also tend to prevent the assessment of leasehold equities separately from the coal value assessed to the owner of the land. On the other hand, leases which are so drawn as to constitute a sale of the coal in the ground, tend to encourage assessors to assess a portion of the coal value to the lessee as the owner of the same.

5. The proposal to base the assessment of coal value arbitrarily upon the quantity of coal (as defined in foot-acre or other units) and a fixed value per ton or per foot per acre, to apply uniformly over a prescribed district, is steadily losing ground, probably because it does not seem to be in conformity with the principle laid down by the courts, that assessments must be fixed by the price obtainable at voluntary sale. This proposed method, therefore, is hereafter likely to be used only as a auxiliary means of determining the voluntary sale value, and not as a method for directly fixing the assessable value.

Many officials are inclined to be pessimistic as to the possibility of securing greater uniformity in methods of making assessments, apparently because the subject appeals to them, and to the residents of each district, more in the light of a local issue than of one of broader importance. Some of the suggestions illustrating the trend of thought as to proposed changes and the securing of better or more satisfactory results may be of interest thus:

"By making the tax directly upon the output."

"Now being considered by the State Tax Commissioners."

"By revising method of taxation for all property."

"By estimates of quantity of coal in each tract."

"Most any other way would be an improvement."

"Revision of laws to provide assessment at fair sales value."

"Revision of laws to determine assessments by amount invested."

"By assessing surface and coal separately."

"Revision desired, but no prospect of getting revision."

"Tax should be on the production."

"State officials now considering method, but no conclusions reached."

"Should be controlled by a Tax Commissioner."

"No one seems to be thinking about it, are satisfied with present methods."

"By the appointment of County Tax Commissioners."

"Assessments should be low enough to permit operators to hold (unworked) coal land."

"By classifying coal land as working, reserved and undeveloped acreage."

"Depth, length of time to reach, accessibility and other conditions should be considered."

"By assessing each field separately at unit prices governed by the average selling price."

"By employing engineers to assist Tax Commissioners."

"By distinguishing between coal to be mined soon and that not to be operated for some time."

"By assessing all coal lands at their commercial value."

"By taxing on an output basis."

"A tonnage tax is inequitable and should not be favored."

"Opposed to a tonnage tax."

"According to output, value of plant and profits, all to be considered in fixing assessment."

"Should be fixed by the net profits per ton."

"Mining engineers should be employed to assist in making assessments."

"There should be a better way found; present methods most unsatisfactory."

"By classification of lands and the actual production."

Of those officials who were willing to express a definite opinion, about 85 per cent. believe that while the methods are faulty, the results are generally satisfactory to land owners, operators, and the general public; about 10 per cent. believe that both methods and results are unsatisfactory; and about 5 per cent. think that the methods and results are partly satisfactory. It will be understood that these are the expressed opinions of officials engaged in assessing coal land for taxation and may not represent the opinions of coal-land owners and operators. An explanation of the statement that the assessments as made are generally satisfactory to owners and operators is found in the fact that in many mining districts the assessors are influenced in greater or lesser degree by the views and desires of a majority of the residents of the district, and adjust the basis for making assessments to meet the approval of a majority of those most interested. This condition accounts for the wide divergence in methods and results often noted in adjacent districts, and again suggests the conclusion that the assessment of coal land for taxation is at present largely regarded as a local issue and is determined independently in different districts by considerations of local expediency.

That the values at which coal land or coal-mining rights are actually assessed for taxation in the principal coal-producing States seldom bear any fixed relation to the value as determined by prices realized at voluntary sale, is illustrated by the following condensed summary:

County Assessments

State		Highest Assessed Value of Coal, Not Including Surface, per Acre	Lowest Assessed Value of Coal, Not Including Surface, per Acre
Alabama	"Supposed to be assessed at 60 per cent. but rarely is assessed at more than 25 per cent. of value."...	\$1.00 to \$ 40.00	\$0.12 to \$3.00
Arkansas	"Supposed to be assessed at 50 per cent., or less, of value at voluntary sale."	5.00	2.50
Colorado	"Depends on accessibility to railroads.".....	60.00	25.00
Illinois	"Usually at about 20 per cent. of voluntary sale value.".....	75.00	2.00
Iowa	"Undeveloped lands not assessed as coal land."....	10.00 to 30.00	6.00 to 25.00
Kansas	"Supposed to be at its market value.".....	20.00 to 60.00	5.00 to 10.00
Kentucky	"Assessors often adopt statement of owners as to value."..... (Data not complete.)	2.00 to 15.00	1.00 to 4.00
Ohio	"Attempt to approximate value." "More guesswork than anything else." "Actual values tried to be ascertained.".....	20.00 to 80.00	10.00 to 20.00
Pennsylvania	Methods vary greatly.... (Bituminous region only.)	10.00 to 900.00	5.00 to 50.00
Tennessee	Data incomplete.....	20.00	3.00
Utah	Data incomplete.....	20.00	10.00
Virginia	At fair market value as per Act of March 7, 1912.	100.00 to 500.00	1.00 to 8.00
West Virginia	"Supposed to be at voluntary sale value.".....	6.00 to 180.00	3.00 to 15.00
Wyoming	"On net value of output." (Data incomplete.)	20.00 to 30.00	20.00

This summary is not thought to be complete, or free from error. Many of the wide variations in assessments, perhaps most of them, merely reflect actual differences in the real values of coal land. Individual cases in which the assessments are higher or lower than those given in this summary doubtless occur in each of the States above enumerated. The table is introduced to show the maximum assessments in the two counties that show the highest and lowest maximum value and the minimum assessments in the two counties that show the highest and lowest minimum value, for the counties of which a record is available.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Mining Claims within the National Forests

BY E. D. GARDNER, MISSOULA, MONT.

(Salt Lake Meeting, August, 1914)

WHEN the National Forests were created, all lands embraced in their boundaries were exempted from all forms of entry, except mineral claims. Later, by Act of June 11, 1906, and as amended by Act of Aug. 10, 1912, provision was made allowing homesteads on areas chiefly valuable for agriculture and not necessary for the public use. Patents to mining claim in the Forests are granted the same as on land not in the Forests, irrespective of timber or other values.

It is the policy to encourage the highest use of the Forests. As a business proposition, it is a good policy to have mines developed in the Forests. Their development in remote or inaccessible regions causes the country to be opened up and, in addition to the local market which the mines create, means of transportation are afforded to outside markets.

No examination of mining claims is made until patent has been applied for, unless they are actively interfering with the administration of the Forest. In this case, an examination is made, and if there is evidence that the claims are being held in good faith they are in no way interfered with. Timber is never sold from a mining claim except with the consent of the claimant, or unless it is clearly shown, after an examination, that the claim is abandoned. In the case of fire-killed timber, which rapidly deteriorates by standing, a contract bond arrangement can be made for the sale of the timber which protects the interests of both the mining claimant and the government.

The promiscuous locating of mining claims has in the past caused considerable trouble. It has been, and still is, the custom in some parts of the West to locate a mining claim when the surface rights of any particular area are desired, irrespective of the mineral character of the ground. This has been done to hold power sites, town sites, summer residences, springs, etc. When the ground is desired for any of these purposes, what is known as a Special Use Permit will be issued at a nominal sum, to legalize the occupancy. In most instances, it is far cheaper to take out one of these permits than to do the annual assessment

work on a mining claim, but, of course, patent to the ground cannot be obtained in this manner.

If mineral was discovered on the land covered by a Special Use Permit, it would ordinarily be the policy to cancel the permit if it was clearly shown that the land in question was more valuable for mining than for the purpose for which the permit was granted. The burden of proof in such a case must be on the mining claimant. For example, in the case of a permit for a large storage reservoir, it would take a very good mineral showing to justify its cancellation. If a valid mineral claim occupied ground previous to the application for a Special Use Permit, none would be granted without the consent of the mining claimant.

An unpatented mining claim in a National Forest cannot be put to any use inconsistent with the development of the claim, without a permit, which would be charged for at the regular rate.

In the past, before the government inspected ground prior to issuing patent, many valuable power sites, town sites and timber areas have been patented as mining claims when the ground was in no respect mineral.

It has often been stated that the patenting of mining claims is seriously delayed if they are within a National Forest. Out of over 150 claims examined during each of the years 1912 and 1913 in Montana and Idaho, north of Salmon river, only 17 1/2 per cent. were contested. The records show no delay longer than three months except where patent was applied for during a time of year when field examinations could not be made on account of climatic conditions.

Patenting Claims

Very few claims that come up to patent comply with all the Federal and State laws in every particular. Most of this non-compliance, however, is due to ignorance of the law. One reason for this ignorance is that the requirements for staking and patenting mining claims are not the same in all States. Most prospectors and miners operate in more than one State and many times confuse the various regulations.

In accordance with the mining laws, patent on a mining claim should be issued when the claimants show: first, good faith; second, the requisite amount of work done; and third, a mineral discovery made. No protest is made against issuance of patent by the Forest Service when the location is not in accordance with all of the State laws, or for any non-compliance with the minor technicalities of the Federal law. For instance, an application for patent would not be protested for the reason of the vein not being in the middle of the claim or for striking across it.

Good Faith.—On a claim that is chiefly valuable for mining, good faith is assumed to have been shown when the requisite amount of work has been performed and a mineral discovery made. When it is evident

that the ground is more valuable for other purposes than for mining, a merely technical compliance with the requirements of the law does not suffice to make clear the good faith of the applicant.

Development Work.—Five hundred dollars is required to be expended on or for the benefit of a claim before patent is granted. When a claim is examined for patent the value of the work is estimated as closely as possible from the facts at hand, but when any doubt exists the claimant is always given the benefit of the doubt. When work is done on another claim for the benefit of the one in question, the Service does not take a narrow and technical view of what would tend to develop it.

Mineral Discovery.—Application is often made for patent on claims held in apparent good faith but on which no mineral discoveries have been made. Such claims are the so-called protection, or side, claims, and the odd claims in large groups. These protection claims in many cases are really necessary to the group to avoid possible future trouble with later locators and for tunnel sites, etc.; but, no matter how necessary they are, the law requires that a mineral discovery must be made on each claim before a valid location can be made. Most claims which are contested by the government are of this character.

When, after a sufficient expenditure has been made, patent is desired on a group that includes claims which do not contain mineral discoveries, the common improvement work should be credited to all of the locations and application made only for those which meet the requirements of the law. Patent can be asked for the protection claims under a separate and later application after mineral discoveries have been made. As long as the assessment work is kept up, possession of such claims can be retained.

There is a great difference of opinion as to what constitutes a mineral discovery. According to late decisions of the Commissioner of the General Land Office in the East Tintic case of June 15, 1912, and again in the Jefferson Montana Copper Mining Co. case of Sept. 5, 1912, it was held that the following elements are necessary to constitute a valid discovery upon a lode mining claim:

1. There must be a vein or lode of quartz or other rock in place.
2. The quartz or other rock in place must carry gold or some other valuable mineral deposit.
3. The two preceding elements, when taken together, must be such as to warrant a prudent man in the expenditure of his time and money in the effort to develop a valuable mine.

In passing on a mining claim, the Forest Service Mineral Examiner takes into consideration the general geological conditions of the locality. For instance, in the Cœur d' Alêne and the Butte districts the ore or mineral values do not usually outcrop at the surface, and the presence of a well-defined vein with the characteristic iron-stained vein matter

upon a claim, considering the experience gained on other veins in the districts, would afford a reasonable expectation of finding ore at depth.

In determining a mineral discovery on a placer claim, the values found are considered together with the extent of the gravel and mining conditions. A very small value to the pan of gravel would be considered as a mineral discovery where conditions were such that there was any possibility that the ground would ever be mined, but a few isolated colors found on a claim would not in themselves constitute a mineral discovery. In examining a placer, the same as a lode claim, the good faith of the claimant, and the value of the land for other purposes, are taken into consideration.

The Land Office records will show that only a small proportion of the cases on which contests have been initiated by the Forest Service ever come to a hearing, as the claimants do not file answers to the charges against the claims or do not put in an appearance before the Register and Receiver of the local Land Office at the date set for hearing. Final decisions by the Department of the Interior on all mineral cases contested since July 1, 1911, by the Forest Service in District 1, which comprises Montana, northern Idaho, eastern Washington, and western South Dakota, with the exception of one claim, have been favorable to the government.

The examination and protesting of worthless claims in the National Forests often protects the public from "wildcats." There have been instances, particularly in the outskirts of large districts, where companies have been formed and stock has been sold to develop ground without any real mineral value. To further the stock-selling schemes, patents have been desired and applied for, but when the claims were protested they have generally been abandoned.

Timber on a mining claim can be cut and used for the development of the claim. No permit from the Forest Service is necessary and no charge is made. The locator cannot, however, sell the timber or use it for any purpose foreign to the development of the claim. If more timber is needed for development than what is on the claim, it may be obtained elsewhere in the Forest without any charge under a free-use permit. When a mine has become productive and still needs more timber, a charge is made for it. The government, in making timber sales to outside consumers, always considers the local miners and leaves their supply unimpaired.

Had it not been for the establishment of the National Forests, a large part of the timbered areas now embraced within the Forests would by this time have passed into the hands of private interests as timber and stone claims. Once one of these claims is patented, the patentee not only controls the timber and surface rights, but any mineral that may be discovered on the ground. These non-mineral entries not only discourage

prospecting, but tie up the mineral development of large tracts. In addition to being taken up under the non-mineral laws, these claims would gradually come into the possession of large timber companies who have no interests in mining. Even an unpatented timber and stone claim is a drawback and discourages mining.

Any one who has ever had a mining claim conflicting with a claim of this kind can testify to the cost and annoyance of protesting the patent to the timber and stone claim and fighting protests against the issuance of patent on his own claim. The location of timber and stone claims cannot be made within the National Forests and any one is free to enter the Forests for prospecting and the location of legitimate mining claims.

The miners are gradually coming to realize that the reason for the difficulty in patenting some mining claims is in the law itself, and that the Forest Service has no arbitrary power to change the statutes.

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The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada

BY G. H. CLEVENGER, PALO ALTO, CAL.

(Salt Lake Meeting, August, 1914)

(Continuing the discussion of the paper of James Johnston, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 85, January, 1914, pp. 107 to 133.)

THIS paper cannot fail of being of great interest and value to all who are interested in the cyanide process, on account of the important developments which it records. Mr. Johnston is to be commended for the large amount of detail which he gives, and the company and its management for permitting the publication of data of this character, which while of great value to others could well be considered private property.

It might be discussed from a number of standpoints: First, as an example of recent practice in mill and cyanide plant construction; second, the mill practice; third, the cyanide practice; and fourth, the new features in cyanide treatment involved. Perhaps a more vivid way of expressing it would be: First, the equipment or tools with which the various operations are carried on, are they the best and most efficient which could have been selected for their respective uses; and, second, is the best use in every case made of these tools? It may be well inquired is the highest possible recovery of silver made at the lowest cost per ounce recovered? For after all, the making of the greatest ultimate profit is inseparably associated with successful metallurgical practice; there must therefore be at all times a careful balance maintained between recovery and cost of obtaining it.

I wish to call attention in a very general way to the equipment used and to an anticipation of questions which are bound to arise in this connection. However, the bulk of my discussion will be confined to the presentation of certain additional data, together with a more or less general discussion showing the line of development which leads up to the present practice and the applicability of certain features of it to similar problems elsewhere.

There are a number of vital points of broad general interest involved in Nipissing low-grade practice upon which metallurgists will by no means be agreed to which I wish to call particular attention in the hope that it will lead to the presentation of data by others of practice in other districts which will assist in reconciling this divergence of opinion.

It will be noted that the crushing equipment closely follows recent South African practice. The use of a silix lining in the tube mills might be questioned from the standpoint of economy; but quite aside from this point, upon which I am not prepared to give authoritative data, there are perhaps two distinct advantages possessed by this type of lining in this particular case. With the very fine grinding practised at this plant there would be, with iron or steel liners, a much larger proportion of finely divided iron in the pulp. My experience with the treatment of the high-grade ore showed that the presence of iron interfered with the extraction of the silver. It is also a well-known fact that metallic iron in an extremely finely divided state may cause an important cyanide consumption. The installation of the old type of mechanical agitator will be questioned by many operators, particularly as there is an air lift used in each tank in addition to the mechanical stirring gear.

There has been considerable misconception regarding the character of the low-grade ore, and certain writers have made rather sweeping statements regarding the non-suitability of Cobalt ores for vacuum filtration, presumably upon the assumption that the low-grade ore had a heavy gangue similar to the high-grade ore. The specific gravity (2.70 to 2.72) of the low-grade ore is not very different from that of the siliceous ores of other districts. My own experience in making the large-scale tests was that it was a most favorable ore for vacuum filtration. This statement I do not wish misconstrued as meaning that the vacuum type of filter is unqualifiedly the best for Cobalt ores for so far as I am aware comparative tests of the different types of filters have never been made in this district.

Briefly the status of metallurgical practice in the Cobalt district at the time that we began the investigation of the treatment of Nipissing ores was as follows: The high-grade ore was separated by sorting and jigging at surface plants, and sold to smelters outside the district, there not being up to the time of the advent of the Nipissing high-grade mill any of this ore treated locally. The low-grade ore, constituting the portion rejected in separating the high-grade ore, the dumps, and the low-grade material taken directly from the mine, was originally treated exclusively by concentration, the concentrates being sold to smelters. Later certain of the concentration mills provided cyanide annexes for the purpose of making a further saving from their tailings. Then came the building of mills which were primarily cyanide plants in which concentration was to be depended upon for the removal of the refractory minerals, and the major portion of the silver recovered as bullion through the medium of the cyanide process.

While the mills using the cyanide process unquestionably made a higher ultimate recovery, the mills which had confined their efforts to the development of their concentration practice, although realizing a lower recovery, appeared to be making as great and in some cases a greater ultimate profit than the mills employing a combination process. In other

words, the greater recovery, which necessarily required a greater capital expenditure, did not in all cases mean a greater ultimate profit.

Early in the course of our work we considered the treatment of the low-grade ore and in this connection made a number of small-scale tests. These clearly indicated that the ore could be treated by cyanidation, but they had not proceeded far enough for us to determine certain difficulties, and problems, which even at this early date, we were quite sure would arise on account of the experience of others in the district. At this time we had not worked up to the maximum extraction or assured ourselves that concentration could be dispensed with. It might be mentioned that from the first we were determined to treat all the ore produced by the Nipissing company by hydrometallurgical processes and that nothing but refined silver was to leave the premises. Shortly after making the preliminary tests mentioned our efforts were centered upon developing and getting into successful commercial operation the process for treating the high-grade ore. This fully occupied our time until late in the summer of 1911, when the treatment of the high-grade ore was upon a well-established commercial basis, and we again turned our attention to the treatment of the low-grade ore. After a thorough study of the practice and results of other mills in the district, the management and ourselves were unanimous in the opinion that the issue lay squarely between the relative economy of straight concentration and a process, preferably as near straight cyanidation as possible, which would produce only refined silver. The middle ground involving concentration and cyanide treatment did not appear attractive for various reasons.

The object of concentration in cases where it is practised in conjunction with the cyanide process is to recover gold and silver not soluble in cyanide solution, or to remove interfering elements, but experience shows it to do this but imperfectly, and further, it invariably removes gold and silver which are readily soluble in cyanide solution, which could be to better advantage dissolved by the solution and recovered as bullion. In cases where it is feasible to concentrate after cyanide treatment this last objection of course is not valid; but this cannot be readily done for it means repulping of the tailing and provision for a regular feed to the concentration devices, moreover, the conditions of fine grinding which make for the best results in cyanide treatment are not those most conducive to the best work in concentration. A more or less complete concentration plant means a greater capital expenditure and it further means the control of two separate operations which involve entirely different principles and methods of operation. Generally you are attempting to operate a concentrator and a cyanide plant with neither up to full efficiency. The conditions of the two processes are therefore at variance and it is the constantly growing opinion that concentration in connection with cyanide treatment should only be turned to as a last resort.

Many small-scale tests were made involving various combinations of treatment. The results of these, together with the other data which had been collected in the district, led us to make the first large-scale tests along rather conservative lines, particularly as regards the removal of the native silver and dyscrasite. The preliminary examination of these ores indicated that the silver occurred in three general forms: First, native silver, of greater or less purity, and dyscrasite, which on account of their coarseness would not readily pass into solution; second, silver minerals readily dissolved by cyanide solutions; and third, some combination of silver which resisted ordinary cyanide treatment and only was dissolved after extremely fine grinding, and a long period of agitation.

In order to show the general character of the low-grade ore, the results of four large-scale tests made upon lots of ore representative of the type of ore produced by the Nipissing company, and involving a total of about 36 tons are quoted from rather fully.

Each lot of ore was first sent to the sampler where it was weighed, crushed to 4-mesh and the assay sample separated. It was then transported to the point where the tests were to be made. It might be explained that these large-scale tests were run in the high-grade mill and the time available for them was determined by the time necessary to make certain necessary alterations and repairs to the mill. The tube mill had just been relined and therefore contained no amalgam. Agitator tanks, settler and other equipment were cleaned out very thoroughly and before running the tests proper, low-grade ore was passed through the entire system. I have every reason to believe that in general there was no salting except in the case of perhaps four or five samples. At any rate, if there was any error the probability would be that the residues actually assayed higher than the true value, therefore the percentages of extraction noted would be lower than those to be expected in actual practice, when employing the same method.

The general method of procedure in each case was to grind the ore, 2 tons at a time, with a ton of solution, the proper amount of lime, and a part of the cyanide, for a period of 4 hours. The time of grinding has been previously determined by grinding a sample of the ore in the tube mill for varying lengths of time and making sizing tests. The aim was to have practically everything pass a 100-mesh screen. This method of unit grinding had to be adopted on account of there being no arrangement for the ordinary method of continuous feed and discharge. After grinding, each charge was dumped into a settler containing 522 lb. of mercury, and after each charge the tube mill was rinsed out with cyanide solution which was also allowed to run into the settler. The oak-shod muller was raised well above the bottom of the settler so that there would be no grinding effect upon the mercury. Each charge was agitated in the settler for a period of one hour. It will be noted that no mercury was used

during the grinding. The idea was to give an opportunity for any particles of metallic silver or dyscrasite, which might have resisted grinding, to settle out so that they would not be carried down into the treatment tanks where they would cause trouble. The mercury at the bottom of the settler simply acted as a collector for these particles. Particles of native silver and dyscrasite fine enough to remain in suspension in the pulp during agitation were assumed to be fine enough for satisfactory extraction by solution in cyanide. My idea of this operation was that it was a mechanical method, involving settling rather than amalgamation, of separating a portion of the silver which would either not be dissolved in the treatment tanks or would require an undue amount of grinding to reduce it to a state of subdivision fine enough to be readily dissolved. After agitation in the settler the pulp was drawn off into agitator tanks of the ordinary mechanical type fitted with means of introducing air and so connected that the charges could also be agitated with a centrifugal pump. The progress of the treatment was recorded by determinations made upon samples taken every six hours. The effect of various forms of agitation was studied and in this connection pump agitation was found to be the most effective for the refractory silver minerals which dissolved very slowly at the last. A small amount of air was introduced during the whole agitation period. Pump agitation was not resorted to until at the last.

The following assays of the four lots of ore treated show the distribution of the silver:

	Lot A Ounces per Ton	Lot B Ounces per Ton	Lot C Ounces per Ton	Lot D Ounces per Ton
Pulp.....	29.70	30.10	47.80	27.10
Metallics on 100 mesh.....	0.01	0.01	0.16	0.05
Metallics on 20 mesh.....			0.19	0.07
Metallics on 8 mesh.....	4.18	3.23	5.29	2.79
Metallics on 4 mesh.....	1.24	1.33	0.46	0.98
Commercial assay.....	35.13	34.67	53.90	30.99
Correction on pulp.....	1.30	2.00	2.90	1.10
Corrected assay.....	36.43	36.67	56.80	32.09

The following analysis by Denny¹ of a sample of the low-grade ore will show its general composition.

¹ James Denny: Desulphurizing Silver Ores at Cobalt, *Mining and Scientific Press*, vol. cvii, No. 13, p. 487 (Sept. 27, 1913).

	Per cent.		Per cent.
Silver.....	0.106	Lead.....	0.064
Copper.....	0.270	Calcium oxide.....	9.020
Arsenic.....	1.880	Magnesium oxide.....	4.330
Iron.....	1.920	Aluminum oxide.....	10.030
Sulphur.....	0.640	Carbon dioxide.....	11.060
Bismuth.....	0.010	Mercury.....	trace
Nickel and cobalt.....	0.730	Insoluble.....	59.840

An analysis made by Johnson and Sons² upon a sample of the residues from the high-grade ore, shows antimony 3.8 per cent., bismuth 0.09 per cent., tellurium 1.39 per cent., and traces of tin, zinc, and manganese. These elements are all undoubtedly present in the low-grade ore as there is, unavoidably mingled with it, more or less high-grade ore.

I have also observed the presence of a small amount of graphite in Nipissing ores.

The extraction of the silver by settling averaged 46.7 per cent., and the total extraction by settling and cyanide treatment at the end of four hours of grinding and one hour of settling, averaged 66.5; 19.8 per cent. of the silver was dissolved during this period by cyanide. At the end of the first six hours of agitation, the total extraction had reached 76.3 per cent. At the end of the next 30 hr. of agitation or a total agitation period of 36 hr., the extraction was 87.6 per cent. To raise the extraction 4.2 per cent. or to reach a total extraction of 91.8 per cent. required a long additional agitation period. Of the 8.2 per cent. of silver remaining after long-continued agitation, 3.7 per cent. was soluble in hot dilute nitric acid. The average total percentage of silver which could be extracted by combined settling, cyanide treatment, and dilute nitric acid was 95.5 per cent. The proportion of silver extracted during four hours of grinding and one hour of settling with lots *C* and *D*, which were similar in character, was considerably higher than with lots *A* and *B*, but the extraction curves for *C* and *D* as the cyanide treatment continues, straighten out and soon fall below the curves for *A* and *B*. A continued slow increase in extraction was particularly noted with lot *C*, which contains a somewhat higher proportion of silver. The peculiar character of these extraction results is doubtless due, at least to a certain extent, to the presence of a large proportion of dyscrasite, the coarse portion of which settles readily, while the finely divided portion dissolves very slowly in cyanide solution.

Although these results clearly indicated that a cyanide plant would be most advantageous, it was the general consensus of opinion that more experimental work should be carried on to simplify and improve the proc-

² R. B. Watson: Nipissing High-grade Mill, Cobalt, *Engineering and Mining Journal*, vol. xciv, No. 23, pp. 1078-1079 (Dec. 7, 1912).

ess, if possible. From the first, there was a strong prejudice against the use of mercury in the low-grade mill. This is best appreciated by those who have had experience in handling large amounts of mercury in mill practice. Although certain of my early small-scale experiments had not been very promising when it was attempted to treat the whole ore directly by cyanidation, Mr. Butters strongly adhered to the idea of grinding the whole ore, native silver, dyscrasite, etc., to such a degree of subdivision that all the silver could be dissolved by cyanide. Tests were accordingly made in this direction, and through the guidance of Mr. Butters, gradually led into the present system of treatment where the same idea of grinding the native silver and dyscrasite to a state of subdivision such that it can be dissolved by cyanide solutions is practised, the desulphurizing treatment being only effective in altering the sulphides and sulpho-antimonides.

Now I wish to make this point very clear, for this problem of handling metallic gold and silver in ores which are cyanided, is one that has often perplexed metallurgists and one that has resulted in a variety of solutions. The usual answer to this problem although other methods have been used, is either amalgamation or concentration. On the one hand, we have Mr. Butters who has taken the extreme view that an ore containing native silver and dyscrasite, some of which will remain upon a 4-mesh screen, can be ground and cyanided direct, on the other hand, we find some metallurgists amalgamating comparatively low-grade gold ores preliminary to cyanide treatment, where the actual weight of metallic gold to be ground would be many times smaller than the weight of the metallic silver in Nipissing ore and everything considered much easier to grind and dissolve than Nipissing silver.

Mr. Butters has demonstrated that his plan is possible with Nipissing ore, for the results being obtained in the mill bear testimony to that fact; but quite naturally, it will be asked at what cost? Is the ore ground excessively simply that the more resistant metallic silver and dyscrasite may be reduced fine enough to pass readily into solution and the whim of recovering everything by cyanidation be satisfied? It is true of course that selective grinding can be practised to a considerable extent with heavy brittle minerals, but this is less true of the metallic silver and dyscrasite of Nipissing ore. By selective grinding, I mean grinding of an ore consisting of a relatively light gangue and a heavy brittle mineral so that the mineral becomes more finely divided than the gangue. This automatically takes place in a closed circuit of classifiers, other than screens, and a tube mill. This often causes a mineral to be ground down sufficiently fine for satisfactory extraction of gold and silver without excessive grinding of the gangue.

As a result of the first experiments which were made upon this ore, I called particular attention to the necessity of extremely fine grinding and

this point has been confirmed by all who have since investigated its treatment. Now bearing in mind the necessity for this extremely fine grinding, presumably for certain of the brittle silver minerals, it is readily conceivable that when this necessity for fine grinding is satisfied in a closed circuit of tube mills and classifiers, all of the metallic silver and dyscrasite will also have been reduced to a degree of subdivision, such that it readily passes into solution. After all, Mr. Butters may be right in this case, due to the formation of the ore by "Dame Nature" to fit his metallurgical ideas, or he may have recognized this peculiarity from the outset, and as all wise men should do, has taken advantage of it. Under other conditions than those which *possibly* obtain with Nipissing ore, such a procedure could readily lead to poor economy through the excessive grinding of worthless gangue in order that metallic gold or silver might be dissolved by cyanide solutions, and other methods of recovery dispensed with.

The so-called desulphurizing process is of interest as being the first large-scale application of the use of a reducing agent to effect certain chemical changes prior to cyanide treatment. To the thinking operator, it is indeed refreshing in this day of excessive oxidation and aeration in cyanide treatment, to note that at times there may be virtue in reduction. I might mention that this mania for oxidation has even gone so far that certain operators have aerated their pregnant solutions prior to precipitation. When it is remembered that precipitation is distinctly an operation of reduction, the fallacy of this course is apparent. That the deleterious effect of air upon precipitation is no mere theory, will be confirmed by operators using zinc-dust precipitation who have experienced air leaks in the suction of their precipitation pump.

The idea of using a preliminary treatment involving the use of aluminum and caustic soda in connection with the treatment of precious-metal ores, is by no means new or original as many have supposed. Over 10 years ago Walker³ published an account of experiments made in this direction by himself and Martin. As cyanide operators generally are not familiar with this work, I will quote from it at some length.

"In an experiment designed to determine the effect of chlorine and cyanogen at the moment of liberation at the anode, ore was spread on a piece of platinum foil immersed in a solution of salt and potassium cyanide. The electrode intended to act as cathode was suspended from above. Through an error the feed wires were crossed and the platinum plate on which rested the ore was made the cathode. Almost immediately after turning on the current a purplish-red color was observed around the plate and subsequently a black cloud was thrown off. Upon removing the ore the gold was found securely plated upon the platinum, while the black cloud, when collected and analyzed, proved to be tellurium. To prove that the anions Cl and CN played no part in the

³Walker, W.: H. Note on the Electro-Metallurgy of Gold. *Transactions of the American Electrochemical Society*, vol. iv, pp. 51-53 (1903).

reaction, caustic soda was substituted as electrolyte and identical results obtained. Evidently the gold was being recovered by cathodic reduction alone. *It was found, that in general, any method which subjected the telluride to the action of nascent hydrogen in an alkaline solution would effect the reduction; or to put it differently, whenever gold telluride in the presence of alkali came in contact with a metal from which hydrogen was being liberated, tellurium was set free, and metallic gold appearing on the metal.* Thus for example, when powdered telluride is placed upon metallic aluminum and moistened with caustic soda decomposition immediately takes place. The same is true of zinc, although the reaction is much slower. Sodium amalgam breaks up the grains of telluride with great ease. When an electric current is used the reaction seems to be independent of the metal forming the cathode, and to take place in any neutral or alkaline solution.

"When a dilute acid was used as an electrolyte a reduction of the gold was effected with the greatest difficulty. Only by employing a very dilute acid and a large current density could any apparent decomposition be obtained.

"A possible explanation of the reaction taking place is the following: The hydrogen evolved first unites with the tellurium, forming hydrogen telluride and free gold. In the presence of alkali this compound is immediately broken up with the formation of an alkaline telluride which is easily soluble, and thus removes the tellurium from the sphere of action. This alkaline telluride is in turn oxidized with the formation of free alkali, and metallic tellurium. In the presence of acids, however, the hydrogen telluride is immediately broken up and the free tellurium deposited *in situ*, as it were, upon the gold, thus protecting it from further action. This is indicated by the fact that a silver-white piece of gold telluride becomes black when placed upon the cathode in an acid electrolyte, and also by the fact that when the amount of activity of the acid is decreased by diluting the solution a slight deposition of gold can be obtained.

"When the rate of decomposition is rapid the gold in the small pieces of mineral is not evenly deposited, but frequently retains the original shape of the mineral fragment.

"Quantitative runs were made on a rather rich sulpho-telluride with caustic soda as electrolyte, and mercury for the cathode. By maintaining gold agitation it was not difficult to obtain an extraction of 93 per cent. of the assay value. The presence of finely divided tellurium in the electrolyte seemed to cause excessive flowering of the mercury and much trouble was experienced from this cause. When lead or other solid metal was used as cathode much gold was lost by the abrasion of the moving ore.

"Good results were also obtained by agitating the ore with aluminum shavings in the presence of caustic soda solution, washing out the tellurium and amalgamating the free gold."

Reed⁴ has pointed out that reduction of a metal from one of its compounds by solution of another metal, is not due to nascent hydrogen evolved by dissolving the metal added but to the electromotive force which is maintained by the energy of the dissolving metal.

I was familiar with the work of Walker and Martin a number of years before we began the Nipissing investigation and had considered the possible application of this method of reduction as a preliminary treatment for silver ores which were to be cyanided. But it had seemed to me that as the practice of fine grinding and various other refinements in

⁴Reed, C. J.: The Nascent State, *Transactions of the American Electrochemical Society*, vol. i, pp. 69-78 (1902).

the cyanidation of silver ores, or ores in which silver predominated, developed, extractions had reached such a high stage in the majority of cases that there was little room for the introduction of an extra operation with its attendant complication and expense, even though at the outset it were conceded that a slightly better, or possibly a more rapid extraction, could be obtained. It must be remembered that most silver minerals (there are exceptions) are soluble in cyanide solutions of proper strength when reduced to a sufficiently finely divided state and given the proper time of contact. This is even true of Nipissing low-grade ore: But with certain of the ores, excessively fine grinding is required and an almost prohibitive time of contact to recover the last of recoverable silver. Judging from a rather extensive experience in the treatment of silver by the cyanide process, together with the fragmentary and at times contradictory data available upon the solubility of silver minerals in cyanide solutions, I am forced to believe that there is a great difference in the rate of solubility of the same silver mineral occurring in different localities.

At Virginia City, Nev., the predominant silver combinations are the sulphides and sulpho-antimonides. My experience with Comstock ores covering many small-scale tests as well as actual mill operation, for a considerable period, is that these ores are favorable for cyanide treatment either direct or in combination with concentration.

The more recent experience of Symmes⁵ at the Mexican mill, Virginia City bears this out, as he has very successfully treated these ores without concentration or other preliminary treatment. This is often true of the ores from other districts in the United States and Mexico where the sulphide and sulpho-antimonide minerals of silver predominate. It is, however, true that the same degree of comminution, strength of solution in alkaline cyanide, and a period of contact is not in all cases equally effective. It is therefore evident that there is a very marked difference regarding the solubility of the sulphide and sulpho-antimonide minerals occurring in different districts, and even at times in the same mine. There are unquestionably many plants now treating ores in which the sulphide and sulpho-antimonide of silver predominate where it would not prove profitable to introduce this preliminary treatment. However, it should be realized that there might be special cases where it would be advantageous.

The use of this preliminary reduction in other districts should therefore be approached with caution and only adopted after it has been clearly demonstrated that the extraction is sufficiently improved or facilitated to produce an additional profit above the cost of the extra operation involved.

The conditions obtaining in the treatment of telluride ores are some-

⁵Symmes, Whitman: Personal communication.

what different in that on account of the supposed insolubility of the telluride minerals some kind of preliminary treatment is deemed necessary if the gold or silver is to be dissolved by cyanide, however, here reduction would have to compete with other forms of preliminary treatment already in successful operation.

The tellurides of gold and silver are generally assumed to be insoluble in cyanide solutions, although certain experimental data and mill results lead me to think that this statement is not strictly true. At any rate even in the treatment of the lowest grade ores, as for example in the Cripple Creek district, where concentration precedes cyanide treatment, the average extraction of the gold does not perhaps exceed 75 per cent. Aside from these very low-grade ores where the extraction must be sacrificed for costs in order that the greatest ultimate profit may be obtained, the practice has been and still is to roast preliminary to fine grinding and cyanide treatment. It will thus be seen that the preliminary treatment of roasting for the breaking up of the telluride combination (chiefly with gold in this case) or the partially effectual operation of concentration for the removal of the telluride minerals in the case of the low-grade ore is deemed absolutely necessary for the most satisfactory results in present-day practice. However, there is no telling what the future will bring forth.

Ores containing the telluride minerals of gold and silver are not of great commercial importance in this country except in the Cripple Creek district. Therefore a consideration of possibilities in this district will be of interest. With these ores the issue would be between preliminary treatment by reduction followed by cyanide treatment as against, of the case of the low-grade ore, concentration and cyanide treatment (in most cases the concentrates are shipped to outside reduction plants) and in the case of the higher-grade ore, roasting followed by cyanide treatment.

Due to favorable location and cheap fuel the cost of roasting of the Cripple Creek ores has been brought to a very low figure and the percentage of extraction is very high upon the properly roasted ore. Of course it is only fair to say that there are losses occurring during roasting through dusting and volatilization which should be added to the cost of roasting; but in the case of the two modern mills operating at Colorado Springs these losses are doubtless small, so that in the present discussion as they are not definitely known, they will have to be ignored. It therefore appears that the extraction would have to be fully as high by an all-wet method as when roasting was used. At the present time we have no means of knowing what it would be with telluride gold ores. The problem also arises as to whether reduction would be as effective upon the sulpho-tellurides as upon the simple tellurides. It would therefore appear that where this method has to compete with the lowest cost roasting practice in the world (dead oxidizing roast of gold ores) that it might not

prove advantageous. The relative cost of a roasting plant and a plant for the wet treatment would of course be a factor which would have to be given due consideration in building new plants. If we assume that both methods are equally effective it would appear that the wet treatment plant for a given tonnage would cost less than the roasting plant but in the case of Cripple Creek ores where the roasting plants are already installed, in view of the relative cost of the two operations, it would appear difficult for the wet process to supplant roasting in this district, unless at least as good an extraction be conceded to be possible with the wet process. However, long range predictions are extremely difficult to make and a careful investigation of this subject in connection with Cripple Creek ores would prove most interesting and might lead to important developments.

In a district producing telluride ores, where the cost of roasting was high due to local conditions, and such a district might be discovered in the future, this process of reduction would merit particular consideration.

The data now available upon preliminary reduction is limited to its application upon a working scale to the treatment of the sulphides and sulpho-antimonides of silver at one property, and the use in this case of aluminum and caustic soda as a reducing agent. Small-scale experiments have been made upon gold tellurides in which reduction has been obtained by aluminum and caustic soda as well as by electrolysis. The effect of reduction upon other combinations and associations of the precious metals as well as an inquiry into the effectiveness and economy of other methods of obtaining reduction should be made in a thorough manner.

One question which at once arises is the reduction selective, or in other words what other combinations than those of the precious metals are decomposed? And in what order are they decomposed? This would seem to be a vital point for if in certain cases it would be necessary to decompose a large proportion of valueless minerals before the reagent became effective upon the precious-metal combinations, it would certainly be a most serious drawback. There might also in certain cases be reduction products formed which would interfere with cyanide treatment or cause a high cyanide loss.

Another question which arises, particularly in regard to the sulpho-antimonides: What is the relative effect of using a preliminary treatment with caustic soda alone as against the use of caustic soda in combination with aluminum. There is a certain amount of evidence which seems to indicate that there are cases where all that would be necessary would be to use a preliminary treatment with caustic soda solution or caustic soda in combination with the cyanide solution.

A number of years ago, an investigation was carried on under the direction of C. W. Merrill⁶ in connection with the treatment of the so-

⁶Personal Communication.

called refractory blue ores of the Black Hills of South Dakota. In the course of this work it was discovered that a preliminary treatment with caustic soda solution in conjunction with air rendered the ore amenable to cyanide treatment. There has been at least one case in Australia where an antimonial gold ore was rendered amenable to cyanide treatment by a preliminary treatment with caustic soda. At about the same time that the Nipissing process was being developed, I had under consideration the treatment of some roasted telluride concentrate which had already been treated by the cyanide process. Realizing that perhaps the maximum effect had been obtained in the direction of an oxidizing treatment and remembering the work of Walker and Martin, I made a series of tests in which reduction treatment with aluminum and caustic soda, and zinc and caustic soda preceded cyanide treatment. The extractions were most satisfactory, but a further series of tests in which the aluminum and zinc were omitted gave equally good extractions. It might be mentioned that the effect was not due to the presence of tellurides, for all the tellurides had been decomposed and the tellurium removed by roasting, as proved by determinations made for tellurium, but to another condition which I am not at present at liberty to enlarge upon.

Johnston states in the article under discussion, that the working solutions at the Nipissing mill are more active than freshly made up solutions. Denny⁷ attributes this to the presence of caustic soda in the solution, and its effect in aiding the decomposition of dyscrasite (an antimonide of silver of varying composition). It is also well to bear in mind that this mineral is not affected by the reduction treatment.

To quote further from Walker, it will be noted that he greatly magnified the difficulty of applying reduction upon a large scale. It must however be remembered that his work was done over 10 years ago and at that time fine grinding was not generally used in connection with the treatment of gold and silver ores. In other words, the time was not ripe for the practical application of the idea.

"The enormous practical difficulties in the way of using a method of this kind is appreciated when we consider that every minute a particle of mineral scattered through the relatively enormous quantity of gangue must be brought into direct contact with the surface, liberating the hydrogen before any reaction will take place. The fact that tellurides of gold are insoluble in potassium cyanide, and so far as electrolytic methods are concerned are affected only by cathodic reduction may explain why many electrolytic processes have promised well when experimenting with rich ores but have failed when worked upon poor or lean ores. With a rich ore on a small scale it is not difficult to bring a large percentage of the individual particles into direct contact with the active surface and thus get a high extraction, even though the experimenter was

⁷ Denny, James: Desulphurizing Silver Ores at Cobalt, *Mining and Scientific Press*, vol. cvii, No. 13, p. 488 (Sept. 27, 1913).

⁸ Walker, W. H.: Note on the Electrometallurgy of Gold, *Transactions of the American Electrochemical Society*, vol. iv, pp. 53-54 (1903).

ignorant of the reaction by which the values are recovered. In a lean ore, however, on a practical scale each particle of ore is surrounded by such a great quantity of gangue that an amount of agitation sufficient to keep the ore in suspension and to allow the passage of the electric current is not capable of effecting complete reduction at the cathode or other active surface."

There would seem to be several conditions which would have to be fully met to make the operation both effective and economical. First, every particle of mineral should be brought directly in contact with the reacting substances; this presents quite a problem as the mineral particles to be decomposed form a very small proportion of the total mass which it is necessary to subject to reduction; second, the presence of any excess of the metal used in the pulp to be cyanided, would cause premature precipitation and would therefore be fatal to the best results; third the reaction would be sufficiently slow so that reduction would precede without undue waste of the reagent. To sum up: The reduction of the refractory mineral should take place as completely as possible with a minimum consumption of reagents and without any excess of the metal used remaining in the pulp at the end of the treatment and where zinc precipitation was used caustic soda should be eliminated so far as possible.

The passage of the pulp through the tube mill containing the aluminum slugs seems to fulfill all these conditions.

The author seems to be strongly of the opinion that zinc dust could not have been used as a precipitant in the Nipissing low-grade mill on account of the fouling of the solution as solutions containing arsenic and zinc were shown to give a diminished extraction.

Hamilton⁹ is also of the same opinion, but Denny¹⁰ is not so sure that it would not have been possible to overcome the difficulties which arose when zinc was used as a precipitant. In some of the experiments which he made in which zinc dust was used, he found that after the solution was allowed to stand eight days, it recovered its original dissolving power, but he was at a loss to explain this phenomenon. A very plausible explanation for this is apparent when the behavior of zinc precipitation in treating the high-grade ore is considered. Apparently, so far as I have been able to observe, this difficulty does not arise in treating the high-grade ore although the amount of deleterious impurities as well as zinc passing into solution is many times that encountered with the low-grade ore. The cyanide concentration is of course much higher, which is not without its influence; but further than that the real explanation seems to be that the arsenic, antimony, etc., separate during and after precipitation from the solution to a concentration which does not interfere under the condi-

⁹ Hamilton, E. M.: Aluminum Precipitation at Nipissing, *Engineering and Mining Journal.*, vol. xciv, No. 19, p. 935 (May 10, 1913).

¹⁰ Denny, James J.: Desulphurizing Silver Ores at Cobalt, *Mining and Scientific Press*, vol. cvii., No. 13, p. 48 (Sept. 27, 1913)

tions of treatment maintained. The percentage of zinc is also reduced upon standing as well as by reactions which take place when the solution comes in contact with a fresh lot of ore and the mercury during amalgamation.

Shortly after starting the high-grade plant, I observed that there was a considerable amount of finely divided black precipitate being carried over from the foot of the precipitation boxes into the sump. This was at first thought to be a silver precipitate in suspension due to the violent action in the boxes, but it was soon demonstrated that it contained a relatively small proportion of silver. At times the solution leaving the foot of the boxes would be a clear wine color. A sample of this colored solution when allowed to stand in a glass beaker for some time became the color of the ordinary solution while at the bottom of the beaker there was deposited a black precipitate. Watson¹¹ gives the following analysis of the precipitate which collects in the bottom of the sumps at the High-Grade mill:

	Per Cent.		Per Cent.
Silver.....	0.394	Iron.....	5.040
Mercury.....	2.510	Nickel.....	9.060
Antimony.....	3.300	Cobalt.....	7.030
Arsenic.....	32.640	Lime.....	9.240
Sulphur.....	16.130	Carbon dioxide.....	7.259
Silica.....	5.362	Manganese.....	trace
Zinc.....	2.257		

It is evident that these impurities are much more difficult to precipitate than the silver for they seem to separate largely at the foot of the precipitation boxes, and indeed a part of it goes over as a colloidal solution from which the precipitate separates completely, only after standing several days in the sump tanks. It might be mentioned that agitation seemed to facilitate this separation. In precipitating the solutions from the first large-scale tests by zinc-shaving precipitation, the wine-colored colloidal solution was noted at times at the foot of the precipitation boxes. However, in general, it was not apparent with the low-grade ore on account of the much smaller percentage of the deleterious elements which it contains. In general zinc-shaving precipitation throughout my tests gave very complete and satisfactory precipitation; but it must be remembered that conditions were somewhat different than those obtaining in the treatment later developed. My own observation together with facts given by Denny, lead me to think that zinc precipitation under proper conditions would not have been impossible in the cyanidation of Nipissing low-grade ore. However, it would, no doubt, have required a very careful investigation of the conditions governing the separation of the undesirable impurities from solution during and after precipitation.

¹¹ Watson, R. B.: Nipissing High Grade Mill, *Engineering and Mining Journal*, vol. xciv, No. 23, p. 1080 (Dec. 7, 1912).

Perhaps the chief interest in the discussion of this feature of Nipissing practice centers about its applicability elsewhere. This is a subject which must be approached with considerable caution for despite the general prejudice which has existed against zinc since the cyanide process was first introduced, zinc has in the great majority of cases been giving most excellent results, particularly when used in the form of zinc dust. This statement is made as a result of a most critical consideration of the whole field of precipitation with a view of finding a substitute for zinc which would perform all its functions. The use of aluminum or even aluminum dust as a precipitant, is by no means a new idea so that if it did not possess some very obvious disadvantages it would have come into general use as a precipitant for gold and silver from cyanide solutions long ere this. I am inclined to think that its future use will be still confined to special cases.

Moldenhauer¹² first proposed the use of aluminum as a precipitant and obtained a patent for its use in 1893.

Early experiments with it by Julian¹³ in South Africa in the form of plates and shavings with solutions containing gold, did not prove satisfactory. The following quotation from Julian will give an idea of the difficulties encountered:

"It is necessary for efficient precipitation that either free alkali or free acid should be present in the solution. For obvious reasons, the latter is difficult to apply, if not impracticable, in the cyanide process. It was found after the process had been at work some weeks that aluminum began to deposit on the shaving, which retarded precipitation of the gold. This was easily rectified by the addition of a little alkali at the head of the box, but as time went on the ever increasing quantity of alumina in the working solutions became more and more troublesome. Attempts were made to remove the alumina by precipitating it in the solution tanks, and then leave it behind in the ore, but it was found that *whenever alumina was precipitated in a cyanide solution it also carried down a large proportion of the free cyanide*. Methods have been suggested for overcoming the difficulty, but at a prohibitive cost."

Kirkpatrick¹⁴ began experiments in 1906 with aluminum dust as a precipitant and states that, when used in the form of dust, the difficulties experienced by the earlier experimenters are overcome. Nevertheless, it is difficult to see how this could have any effect upon the formation and separation of alumina from the solutions. The use of aluminum dust was first applied commercially by the Deloro Mining & Reduction Co. in 1908 and was in use as late as June, 1913, in connection with the cyanidation of the speiss resulting from the smelting of Cobalt high-grade ores. The same interests introduced its use at the O'Brien mill, in the Cobalt

¹² Moldenhauer, Carl: New Zealand, Patent August 31, 1893.

¹³ Forbes, Julian H. and Smart, Edgar: *Cyaniding Gold and Silver Ores*, 2d ed., pp. 164 and 165. (London, 1907.)

¹⁴ Kirkpatrick, S. F.: Aluminum Precipitation at Deloro, Canada, *Engineering and Mining Journal*, vol. xcv, No. 26, p. 1277 (June 28, 1913).

district where, although in use about three years, it attracted but little attention. The general character of the ore treated at the O'Brien mill is similar to that treated at the Nipissing low-grade mill. Rough concentration is practised prior to cyanidation.

The high cost of aluminum dust as compared with the cost of the metal in other forms as well as a part of the difficulty of wetting it and the comparatively long period of violent agitation necessary, even in precipitating silver solutions due to a coating of the oxide and stearine, is largely explained by a consideration of the method of manufacture.¹⁵

It is necessary to start with the foil or leaf as a raw material. This is manufactured by a special system of rolling or combined rolling and hammering, and it is about 0.01 mm. in thickness. With either process, the percentage of first class foil is only 33 to 35 per cent. of the 0.04-mm. metal started with. The imperfect foil is converted into dust by comminution in two series of special stamp mills. The product from the first series of stamps is classified and the finest of the product separated and passed on to the second series of stamps for the completion of the operation. During this second stamping, the difficulty due to the agglomeration and welding of the particles is overcome by the addition of 2 per cent. of *stearine*. The product of the second series of stamps after being screened through No. 200 silk bolting cloth, is further classified by a winnowing process. Although the finished product is polished in a special cylinder by brushes, a considerable proportion of the 2 per cent. of stearine used to prevent agglomeration must still remain as a coating upon the particles of the finished product.

Butters¹⁶ has sought to overcome some of the difficulties of aluminum-dust precipitation by a system employing granulated aluminum in a tube mill through which the solution to be precipitated is passed during rotation of the mill. The bulk of the precipitate is to be collected by subsequently passing the solution through a filter press.

In order to come to the point, let us ask a few pertinent questions: first, is aluminum-dust precipitation cheaper than zinc-dust precipitation? (of course in this connection we must give due consideration to all factors, even those having an indirect bearing upon the subject); second, is there any function which it would not perform which zinc does? third, is it as effective for dilute solutions containing a small proportion of metal as is often encountered in treating low-grade gold ores?

The average consumption of aluminum dust over a period of nine months, is given as being 1 lb. avoirdupois for each 3.104 lb. avoirdupois of silver precipitated. In the Pachuca district¹⁷ it has been regular practice with a Merrill zinc-dust precipitation system to precipitate 1 lb. of

¹⁵ Process used at La Praz and Charleville—see Anelle by the Société française de Couleurs métalliques.

L. Guillet: *Revue de Métallurgie*, vol. ix, pp. 147 to 159 (1912); also *Mineral Industry*, vol. xxi, pp. 23, 24 (1912).

¹⁶ U. S. Patent No. 1,092,765, Apr. 7, 1914.

¹⁷ Merrill, C. W.: Personal communication.

fine bullion per pound of zinc dust, and with a double circuit system of precipitation even better results have been attained. There is no doubt that this result could be duplicated or bettered at the Nipissing low-grade mill, for in general with zinc precipitation, the greater the proportion of silver in the solution to be precipitated, other conditions being the same, the higher the efficiency.

Hamilton¹⁸ gives the cost of zinc dust laid down at the Nipissing plant at 7c. per pound while aluminum dust costs 35 to 39c. per pound. Suppose that no charge is made for the caustic soda necessary with aluminum or the extra power necessary for agitation, or no allowance for the cyanide carried down with the alumina which sooner or later must separate from the solution, and no credit is given the aluminum for cyanide regenerated, we find that the actual cost of precipitating a pound of silver with aluminum would be 11.3 to 12.5c. while with zinc dust, the cost of precipitating a pound of silver would be 7c. Upon this hypothetical basis, the costs are very much in favor of zinc. The cyanide regenerated has of course a most important bearing upon the cost of precipitation as the value of all that is saved should be deducted from the total cost of aluminum precipitation. I prefer to use the term, saved, rather than regenerated, for the reason that cyanide may appear to be regenerated, as indicated by titration at various stages of the process, and still this may not constitute a real saving in the amount eventually used.

The regeneration of cyanide noted is 1.67 lb. per ton or 408 lb. per day. And as this is given as one of the practical benefits gained by a change in the mill plan from zinc dust to aluminum-dust precipitation, it deserves careful consideration. (Zinc dust was never used in the completed mill). In other words, it is claimed that there would have been required 1.67 lb. per ton of ore treated or a total of 408 lb. per day, additional cyanide if zinc precipitation had been used.

This claim for cyanide regenerated is evidently based upon the difference between titrations made upon the solutions before and after precipitation. There will be considerable difference of opinion regarding just how much weight should be given determinations of this kind as indicating the amount of cyanide which would actually be saved to a plant through aluminum or any other form of precipitation. As having a direct bearing upon this question, the extreme view of Hamilton¹⁹ is quoted:

"Some may dispute the ground I take when I count as lost the cyanide that remains combined with zinc, because it is often stated that the double cyanide of zinc and potassium is almost as efficient for dissolving purposes as the simple cyanide. My experience is, however (at any rate in the case of silver ores), that the reading obtained by the use of potassium iodide indicator with excess of caustic, is worthless as a

¹⁸ Hamilton, E. M.: Aluminum Precipitation at Nipissing, *Engineering and Mining Journal*, vol. xcv, No. 19, p. 939 (May 10, 1913).

¹⁹ *Idem*, p. 936.

measure of the dissolving power of a cyanide solution, the efficiency being for practical purposes proportional to the "free" cyanide reading obtained by stopping at the first faint opalescence without the use of potassium iodide indicator."

Now if we concede this saving in cyanide, claimed through precipitation with aluminum, it is necessary for us to agree with Hamilton's statement regarding the absolute loss of all cyanide combined with zinc, and concede the entire effectiveness of that liberated by aluminum.

Most operators will disagree with this extreme view, but regarding the degree to which cyanide combined with zinc becomes, later, available for extraction there would of course be a great diversity of opinion.

There is no accurate method for the determination of the uncombined alkaline cyanides in mill solutions. The method invariably used in present-day mill work, is the well-known Liebig titration which employs a solution of silver nitrate. Practice in its application differs somewhat in that at times, certain constituents in the cyanide solution are depended upon to act as an indicator, and again an independent indicator is added. A great variety of results will be obtained from the same mill solution, particularly if much zinc is present, depending upon the indicator used and the extent to which the solution has been diluted prior to titration. The method of titration without dilution and without an indicator generally gives the lowest result and perhaps the one which, in most cases, is the most nearly a measure of the simple alkaline cyanide present, but let no man delude himself into thinking that it gives an accurate result in a mill solution, especially one containing zinc, whereas the other methods are inaccurate. This procedure, while relatively more accurate than that in which potassium iodide (the most generally used indicator) is added and the titration carried to the first yellow opalescence instead of stopping at the first faint white opalescence as in the method when an indicator is not used is more difficult to carry out. Its proper performance requires considerable experience, absolutely clean apparatus, and a good light. As the cyanide indicated by any modification of the Liebig method, is only relative, a great many operators prefer to use potassium iodide as an indicator on account of the greater ease with which the determination is made. If proper allowance be made for the higher result obtained when potassium iodide is used, there is no reason why the operation of a cyanide plant cannot be just as successfully carried on when guided by titrations made with potassium iodide as an indicator as by the other method. In fact, where the ordinary solution man is depended upon to make the titrations, as is usually the case, there is less chance of mistake. *The important point to bear in mind is that in any particular mill the same method should be adhered to throughout so that results are comparable.* A good illustration of this is one which came under my observation of a mine manager who, having made a record for low costs at one mine, was appointed manager of an adjoining mine. In the course of time, the

cyanide plant came in for its share of consideration in the operation of cost shaving, and it required only one glance to satisfy him that too much cyanide was being used, for the solutions all through the mill were being kept at a higher cyanide strength than had been found necessary at the mine he had just left, which was in the same district and had the same general character of ore. He, therefore, ordered that no more cyanide be added until the strength of the solutions fell to the level which his former experience had shown was most advantageous. Remonstrance upon the part of the cyanide chemist was of no avail, for this mine manager was supreme, and above the details of cyanide titrations, and had a record for low costs to maintain. The effect was apparent in a few days, and in as many weeks, the situation became critical and the mine manager was frantically calling for help, for although he was reducing costs upon cyanide consumed, he was not obtaining the extraction and he saw that the end was at hand unless something heroic was done at once. Finally, a chemist at his former plant came to the rescue. Being a wise chemist, and having had considerable previous experience in cyanide plants, he at once titrated the solutions without potassium iodide indicator, and to his surprise found that they contained little free cyanide. Knowing the remedy, he at once began adding cyanide in a most reckless manner and extractions began to improve and were soon back to the old level. The solution strength was maintained the same as in the manager's first plant, the titration employing potassium iodide was discarded, and with it went the former chemist. The manager was happy for he was operating with the same solution strength used in his former plant, and extractions were as good as when he came to the plant. *But there was no saving in the cyanide consumption.*

I was once called upon to investigate the treatment at a mill where potassium iodide was used as an indicator in making the cyanide titrations. In making my tests, the titration without an indicator was used. In the course of time, it was determined that a mill solution of about 0.10 to 0.11 as indicated by this titration was the best, everything considered, for treating the slime. The solutions in the plant as daily recorded by the solution men, who used potassium iodide as an indicator ranged from 0.14 to 0.15 per cent. I was rather surprised when I came to titrate these same solutions to find that they ran 0.10 to 0.11 per cent. as indicated by the method which I was using. This shows that two independent investigations of the treatment of this ore had produced the same result, although a different method of titration had been used by the man who first started the plant and the absolute indications by this method were higher than by the method I used, which was the same as that so strongly advocated by Hamilton.

Hamilton's wholesale condemnation of the use of potassium iodide as an indicator is the result of making too rigid comparisons between titra-

tions made with and without its use. This is proven by the fact that the operation of many cyanide plants is controlled with entire satisfaction by the Liebig titration, employing potassium iodide as an indicator.

Having considered the methods of determining the simple alkaline cyanides, we can proceed with a consideration of the extent to which potassium zinc cyanide is available as a solvent for gold and silver. The following is quoted from Sharwood²⁰ who has made a critical investigation of the properties and behavior of potassium and sodium zinc cyanides.

"The considerable increase in solvent power observed when caustic potash is added (always providing that oxygen is accessible) coupled with the superior effect of free potassium cyanide, is good evidence that some free potassium cyanide is formed upon such addition, and therefore that in dilute solutions potassium zinc cyanide is partially decomposed by caustic alkali, with formation of simple alkaline cyanide, in accordance with the principle of chemical equilibrium between substances in solution."

Williams²¹ states that the free cyanide shown by titration to opalescence without potassium iodide is, within limits, proportional to the amount of alkali present. Williams²² also states that lime (CaO) when added in solution to the double cyanide of zinc and potassium has the same effect as potassium hydrate per equivalent added, with the exception that the zinc is precipitated as hydrated zinc oxide when silver nitrate is added to the solution.

Virgoe²³ gives comparative results showing the extractive power of potassium zinc cyanide, plain cyanide, and potassium zinc cyanide with the addition of lime. He²⁴ comments upon certain of these results as follows:

"The value of the addition of the alkali is obvious, and this salt is an excellent solvent for gold, though comparatively not so good a solvent for silver, even in the presence of lime, though this may be simply a question of duration of treatment."

Again he²⁵ comments upon the results given in Table 2.

"Thus it is seen how excellent a solvent this salt is in certain cases, and how the extractions are accompanied with a much lower cyanide consumption."

The regeneration by the addition of alkaline hydroxides of simple alkaline cyanides which dissolve gold and silver, has been demonstrated

²⁰ Sharwood, W. J.: A Study of the Double Cyanides of Zinc with Potassium and with Sodium, *Journal of the American Chemical Society*, vol. xxv, pp. 580, 581 (1913).

²¹ Williams, Gerad: The Determination of Constants in Working Cyanide Solutions, *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iv, p. 491 (1903-1904).

²² *Ibid*, p. 487.

²³ Virgoe, Walter H.: Consumption of Zinc in Cyanide Plants, Nature, Cause and Effect, *Proceedings of the Chemical Metallurgical and Mining Society of South Africa*, vol. iv, p. 630 (1903-1904).

²⁴ *Ibid*, p. 629.

²⁵ *Ibid*, p. 633.

upon many occasions in mill practice by raising the alkalinity of the solutions and running for short periods without the addition of cyanide or with the addition of a much smaller amount of cyanide than that normally necessary.

A good illustration of this was the announcement made a number of years ago by a mill superintendent, treating a silver-gold ore, of the discovery that cyanide could be regenerated by running with a high alkalinity. This proved effective for some time in reducing the amount of cyanide used, but later after the reserve of cyanide carried with the zinc had been largely depleted, it was found necessary to return to the old proportions of cyanide. In the ordinary mill solution, when operated under normal conditions, the proportion of zinc in solution remains more or less constant, so that all zinc taken into solution through precipitation above this maximum, which is reached shortly after the plant begins operation, is separated out as insoluble compounds through reactions, at least a portion of which result in regeneration of cyanide effective for the solution of gold.

With aluminum precipitation, it is necessary to add caustic soda just prior to precipitation, and as aluminum does not even temporarily combine with cyanogen, all the cyanide regenerated is at once apparent by the ordinary Liebig titration without an indicator.

With zinc precipitation, the addition of an alkali to the solution prior to precipitation, is not necessary and in the great majority of cases would be a distinct disadvantage as it would result in a greatly increased solution of zinc with its attendant disadvantages. However, suppose we add caustic soda to the solution, after zinc precipitation, and then make titrations as before, we find that there has been a considerable regeneration of cyanide, for reasons already made sufficiently clear. Now this is just what happens in practice as the alkali is added after precipitation, or generally with the ore in the form of lime and therefore comes in contact with the solution as each fresh charge of ore comes under treatment. During treatment, regeneration is going on to a greater or less extent during the whole period. The rather slow regeneration which takes place during treatment when zinc precipitation is used, is not without its advantages as the cyanide is gradually freed from the combination, as needed. That this makes for a somewhat lower cyanide consumption, is proven by the general experience of there being a less cyanide consumption in regular mill practice than when starting a new mill when there is no zinc in solution, or as indicated by small-scale tests in which fresh solution is used.

There are perhaps other causes which contribute to this, but this factor undoubtedly accounts for a large part of the difference. Regeneration taking place during treatment is not apparent by titration as decomposition of cyanide is going on at this period at a more rapid rate

than regeneration. While there are other reactions taking place during treatment which cause regeneration such as dilution, etc., yet the principal influence appears to be the addition of alkali. With zinc precipitation it is therefore apparent that the various reactions which cause regeneration only take place when the solution is applied to the treatment of a fresh lot of ore when zinc separates and cyanide becomes available for extraction, while with aluminum precipitation all the factors contributing to regeneration are present at the moment of precipitation.

Unfortunately, this indirect regeneration as we may term it, is not readily capable of direct quantitative determination as in the case of aluminum precipitation, so that the only way to make direct comparisons of the regeneration taking place in each case, would be to run the same plant upon the same ore for separate periods of time of considerable length, employing both forms of precipitation. A comparison between the actual amount of cyanide consumed in each case would tell the tale. When one has all the evidence before him, he is compelled to admit that there is regeneration of cyanide at certain stages of the process when zinc precipitation is employed, and remembering Julian's statement that alumina and this is the form in which aluminum separates from cyanide solutions, carries down with it cyanide, it is very evident that the saving in cyanide at the Nipissing mill, through the use of aluminum precipitation, is not so great as has been claimed.

Zinc, in solution, is not without its advantages, as it serves a useful function in removing soluble sulphides and to a certain extent, in acting as a protector for the cyanide temporarily combined with it. An example of the important function which zinc at times may perform in cyanide solutions, is the experience of Colbath²⁶ when treating ore from a certain stope in El Rayo mine. The extraction decreased to a considerable extent, and trouble arose with precipitation. Coincident with these difficulties, it was noted that the zinc had disappeared from the solutions to the extent that the titration for total cyanide coincided with that for simple cyanide. Lead acetate was tried as a remedy, but it did not prove effective, so the rather startling experiment of adding zinc in the form of potassium zinc cyanide was tried.

This addition of zinc caused the extraction and precipitation to again become normal. This unusual case, where there was not zinc enough supplied to the solutions through precipitation, very forcibly draws attention to the important function which zinc stands ready to perform. Ordinarily, sufficient zinc is supplied to the solutions through solution of zinc during precipitation. Evidently, in certain cases, aluminum precipitation could not perform all the functions that zinc precipitation does.

²⁶ Personal communication.

It seems to me that the process of reduction and aluminum precipitation work remarkably well together under the Nipissing conditions, for the possible disadvantage of caustic soda carried over from the reduction process, if zinc precipitation were used, becomes an advantage with aluminum precipitation. The caustic soda thus carried over and that added during precipitation on account of antimonial compounds, is a decided aid to the extraction of the silver. The preliminary reduction treatment presumably removes compounds which would tend to form soluble sulphides so that the non-effectiveness of aluminum for the purpose of removing them, is of little moment in this particular case.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Unit Construction Costs from the New Smelter of the Arizona Copper Co., Ltd.

BY E. HORTON JONES

(Salt Lake Meeting, August, 1914)

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INTRODUCTION

WE have endeavored in the following "sheets" to give the unit construction costs derived from the building of the Arizona Copper Co.'s new smelter, Clifton, Ariz., starting in February, 1912, and completing February, 1914.¹ In Chapter I—Unit Costs—are to be found the most elementary total unit costs which the accounts provide for. They are usable in the Clifton district. Here too are found the percentages to be added to an estimate for Engineering and General Expense. In Chapter II—Comparative Costs—these elementary costs have been classified, averaged and reported as labor and material unit costs. In such form the labor unit costs, when properly applied to similar conditions as these under which they were derived, are usable anywhere. The material unit costs are better disregarded for more accurate estimates and replaced by a newly priced bill of material. In Chapter III Composite Costs are given. They are unit costs built up from several elementary units, and likewise units of larger dimensions and simpler application, valuable for

¹ The smelter went into service producing copper October, 1913.

checking estimates and obtaining quick approximations of total costs. In Chapters IV, V, and VI are given the Wage Scale, Material Prices, and a description of the conditions surrounding the making of every elementary unit cost, which will enable an estimator to judge of their use under any circumstance. It should be borne in mind that Chapter VI is not a pure description of the plant, nor of any of its parts, such as might be expected in a technical journal, but it is barely enough information for a reader to judge as to the applicability of a unit cost in another place.

These unit costs, as any examination will prove, are not ideal, but actual. They were made by a copper company organization extending over a period of two years. They represent delays in material shipments (serious delays in steel and brick deliveries), delayed plans, changes in plans, labor troubles, with changes in hours and rates, variable weather conditions, and the like. They do represent, however, every cent spent.

Each unit cost has been judged, as it stands, solely on its merits for use by an estimator. Where a unit cost is not given it was of no use, and where given and described, it must be employed accordingly.

During the first few months of construction work only the prime account numbers showing in the sheet were in use. Later, to obtain more accurate figures to measure the work by as it progressed, the decimal account numbers were added. But the decimal accounts were on record only in the engineering costing department at the new smelter, two miles from Clifton. The General Office, at Clifton, neglected all decimal accounts and charged them under the prime numbers. The labor segregation was in all instances made at the new smelter, and every charge for material, from whatever source, passed through the warehouse at the new smelter. The only other source of cost sheet entries was the General Office cash books. Once a month these entries were itemized and sent out in the monthly Cost Sheet issued by the General Office.

CHAPTER I

UNIT COSTS

The costs thus accrued from three sources. The individual labor card was not used until the excavating was well under way and the foundation work had started. Each man, who could, made out his card and told thereon exactly at what and how long he had been working. The labor bosses made out the cards for the Mexicans. The account number was later placed on the card by the time keeper and checked by a competent man in the engineering office. For every bit of material used on the job a requisition was passed through the warehouse, whether steel building or keg of nails. To this a charge number was attached and the requisition was finally checked by a competent man in the Engineering Department.

In regard to cash-book entries, there was little to be done, as all local bills were OK'd and account numbers attached by the Engineering Department in the first instance. The units were compiled by the Engineering Department daily as the work progressed and checked suitably.

Yet owing to the fact that the General Office kept the accounts segregated only under the prime account numbers, the slow development of the idea of making a final accurate unit cost sheet, the delay in using individual time cards and many clerical mistakes, it was finally thought best to check over every labor card and warehouse requisition from the beginning of the job. This was done and the unit costs as they now stand are believed to be accurate.

The making of this cost sheet is largely due to the untiring interest and insistent demand for accuracy upon the part of three members of the smelter construction force, G. H. Ruggles, M. Am. Soc. M. E.; Roy Earling, M. Am. Soc. M. E.; and H. F. Adams. Credit is due also to the ever willing assistance given by the Arizona Copper Co.'s general office, under the direction of J. G. Cooper, Cashier.

General Expense

NUMBER	NAME OF ACCOUNT	TOTAL
7001	General expense at Clifton.....	See page 1581
7004	Personal injuries.....	" " "

Engineering

Engineering and Superintendence at Douglas

7101	Salaries.....	\$39,706.22
7103	Telegraphing and telephoning.....	167.55
7104	Traveling expense.....	1,984.85
7105	Miscellaneous expense.....	1,018.06
		\$42,876.68

Engineering and Superintendence at Clifton

7201	Salaries.....	\$40,587.54
7202	Furniture and fixtures.....	149.43
7203	Telegraphing and telephoning.....	258.64
7204	Traveling expense.....	934.24
7205	Miscellaneous supplies.....	2,205.77
7206	Miscellaneous labor.....	869.02
		\$45,004.64

9000.1	Power plant engineering.....	\$12,768.56	\$12,768.56
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7100	Total engineering expense.....	\$100,649.88
	Total unit cost.....	5.40 per cent.

This percentage is obtained by dividing the engineering expense by the total cost of the smelter, minus engineering and indirect expense.

$$\frac{100,649.88}{1,864,092.47} = 5.40 \text{ per cent.}$$

7300.—Yard Tracks and Industrial System**Tracks**

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
7301	Excavation.....	\$31,311.14	\$4,254.86	\$35,566.00	55,405 cu. yd.	\$0.64
7302	Ties.....	425.13	10,777.79	11,202.92	10,262 ties	1.09
7303	Rails and rail fastenings	392.00	9,839.79	10,231.79	14,813 ft.	0.69
7304	Frogs and switches.....	206.50	2,676.95	2,883.45	18 switches	160.19
7305	Laying, surfacing and ballasting.....	5,873.53	248.88	6,122.41	17,150 ft.	0.36

Trolley System

7306	Poles and setting.....	1,427.27	1,804.73	3,232.00	106 poles	30.49
.1	Brackets and wiring....	1,167.19	1,379.73	2,546.92	7,824 ft.	0.33
.2	Rail bonds.....	304.31	347.13	651.44	521 bonds	1.25
.3	Lighting.....	229.66	222.39	452.05	57 lamps	7.93

Rolling Stock

7307	Cars, elec. locomotives, etc.....	801.29	34,017.63	34,818.92		
.1	Calcine car alteration...	381.02	4.36	385.38		

Trestle Approach to Reverberatory Building

7308	Excavation.....	359.95		359.95	277 cu. yd.	1.30
7308.1	Foundations.....	738.18	1,152.52	1,890.60	254.9 cu. yd.	7.42
.2	Steel structure.....			13,460.84	163.97 tons	82.09
.3	Woodwork.....	703.93	768.92	1,472.85	27.65 m.b.m.	53.27

150-Ton Track Scales in Receiving Yard

7309	Excavation.....	348.91		348.91	388 cu. yd.	0.90
.1	Foundation.....	545.62	1,146.87	1,692.49	186 "	9.10
.11	Cost and erection.....	365.95	3,450.86	3,816.81	150 tons	25.45
.30	Scale house.....	65.66	63.87	129.53	879 cu. ft.	0.15

Bridges, Culverts and Walls

7310	Bridge No. 1 foundation.	1,028.26	1,968.38	2,996.64	339.8 cu. yd.	8.82
.1	" steel work..			377.40	3.70 tons	102.00
7311	Culvert No. 1 masonry..	1,384.85	695.80	2,080.65	354 lin. ft.	5.88
7312	Retaining walls excavation.....	77.66		77.66	60 cu. yd.	1.29
.1	" " concrete.	512.34	734.25	1,246.59	203.5 "	6.13
.2	" " masonry	88.08	47.51	135.59	21.9 "	6.19

40-Ton Track Scales on Calcine Track

7313	Excavation.....	108.44	0.51	108.95	118 cu. yd.	0.92
.1	Foundation.....	207.55	193.60	401.15	41.6 "	9.61
.2	Cost and erection.....	82.99	710.85	793.84	40 tons	19.85
.3	Scale house.....	100.38	88.21	188.59	879 cu. ft.	0.22

Trestles to Receiving Bins

7314	Excavation.....	548.18		548.18	589 cu. yd.	0.93
.1	Foundation.....	2,408.16	3,017.95	5,426.11	754.3 "	7.19
.2	Steel structure.....			9,269.48	109.35 tons	84.77
.3	Woodwork.....	572.23	838.11	1,410.34	27.21 m.b.m.	51.83

Total cost—Yard tracks and industrial system..... **\$156,326.43**

7400.—Receiving Bins

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
7401	Excavation.....	\$2,303.11	\$39.16	\$2,342.27	1,428 cu. yd.	\$1.64
7402	Foundation.....	1,235.51	2,247.70	3,483.21	612.3 "	5.69
7403	Steel structure.....			29,276.63	353.09 tons	82.92
7404	Gates.....	901.15	1,984.93	2,886.08	30 gates	96.20
7405	Conveyor No. 1.....	310.92	2,947.19	3,258.11	97.3 ft.	33.49
.01	Conveyor No. 2.....	355.19	2,498.03	2,853.22	117.3 ft.	24.33
7407	Lighting.....	60.87	24.67	85.54	22 drops	3.89
Total cost—Receiving bins.....				\$44,185.06		

7700.—Crushing Plant

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
7701	Excavation.....	\$689.67		\$689.67	609 cu. yd.	\$1.13
7702	Foundation.....	893.15	\$1,568.47	2,461.62	220.5 "	11.16
7703	Steel structure.....			2,420.36	25.07 tons	96.54
.1	Doors, windows and frames	84.00	170.71	254.71	529 sq. ft. opening	0.48
.2	Painting woodwork.....	15.00	12.33	27.33	70 sq. yd.	0.39
7704	Crushing machinery.....	392.86	1,093.61	1,486.47	500 cwt.	2.97
.1	Chutes.....	325.87	338.09	663.96	118.1 cwt.	5.62
7705	Shafting, pulleys and belt- ing.....	4.17	483.35	487.52	12 lin. ft.	
7706	Motor.....	102.81	513.64	616.45	50 h.p.	12.33
7707	Power wiring.....	28.55	19.35	45.90		
.1	Lighting.....	76.41	38.22	114.63	8 drops	14.33
Total cost—Crushing plant.....				\$9,268.62		

7800.—Sampling Plant

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
7801	Excavation.....	\$274.09	\$20.97	\$295.06	332 cu. yd.	\$0.89
7802	Foundation.....	605.91	649.44	1,255.35	120.7 "	10.40
.1	Concrete ground floor...	105.68	283.92	389.60	1,222 sq. ft.	0.30
.2	Reinforced concrete floors	1,050.61	1,080.30	2,130.91	4,244 "	0.50
7803	Steel structure.....			10,408.12	110.85 tons	93.89
.1	Doors, windows and frames.....	332.94	564.90	897.84	2,086 sq. ft. opening	0.43
.11	Painting doors and win- dows.....	118.94	28.96	147.90	129 sash	1.15
7804	Shafting, pulleys and belting.....	64.01	1,871.07	1,935.08	85 ft. of shafting	22.76
7805	Meters.....	186.76	887.57	1,074.33	90 h.p.	11.94
7806	Power wiring.....	78.11	120.55	198.66		
.1	Lighting.....	184.45	140.57	325.02	36 drops	9.03
7807	Rolls and samplers, cost and erection.....	1,215.83	7,899.14	9,114.97	1,251.1 cwt.	7.29
.1	Cast iron liners and dry- ing pan, and erection of chutes.....	1,240.69	1,001.86	2,242.55	270.6 cwt.	8.29
.20	Elevator.....	20.39	458.22	478.61	1 elevator	478.61
.50	Steel chutes (made at new smelter).....	1,995.70	596.65	2,592.35	198 cwt.	13.09
7809	Keystone plate partitions	131.04	338.32	469.36	1,523 sq. ft.	0.31
7810	Alterations of chutes and machinery.....	128.69	44.34	173.03		
Total cost—Sampling plant.....				\$34,108.74		

7900.—Bedding Plant and Bunker Bins

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
7901	Excavation.....	\$11,577.53	\$681.90	\$12,259.43	12,319 cu. yd.	\$0.99
7902	Foundation.....	6,256.34	14,513.21	20,769.55	2,809.7 "	7.39
7903	Steel structure.....			47,404.86	548.71 tons	86.39
7904	Conveyors 7'-7 ¹ / ₂	564.34	6,373.67	6,938.01	380.2 ft.	18.24
.1	" 8'-8 ¹ / ₂	1,211.01	8,718.98	9,929.99	562 ft.	17.67
.2	" 9'-9 ¹ / ₂					
	10'-10 ¹ / ₂	1,912.20	9,756.19	11,668.39	905.5 ft.	12.88
.3	Bunker bin gates.....	161.58	1,021.64	1,183.22	42 gates	28.17
.4	Chutes for conveyors					
	7 ¹ / ₂ to 10 ¹ / ₂ inc.....	658.91	842.60	1,501.51	276.2 cwt.	5.44
7905	Two reclaimers.....	3,103.28	30,579.45	33,682.73	2 reclaimers	16,841.37
.1	" wiring.....	377.21	131.19	508.40	2 "	254.20
7906	Lighting.....	306.85	127.38	434.23	63 drops	6.89
7907	Transfer car.....	739.12	3,823.19	4,562.31	1 car	4,562.31
7908	Signal system.....	67.68	28.74	96.42		
Total cost—Bedding plant and bunker bins.....				\$150,939.05		

8100.—Roasting Plant

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
8101	Excavation.....	\$1,547.07		\$1,547.07	1,216 cu. yd.	\$1.27
8102	Foundation.....	765.42	1,069.63	1,835.05	250.5 "	7.33
8103	Steel structure.....			37,252.67	445.28 tons	83.66
.1	Elevator.....			2,189.62	5 tons capacity	437.92
8104	Roasters, cost and erection.....	3,716.94	56,326.09	60,043.03	8 roasters	7,505.38
.1	" alterations.....	620.49	81.69	702.18		
8105	" brickwork.....	4,730.85	12,336.35	17,067.20	16,104 cu. ft.	1.06
.01	" unloading					
	brick.....	363.14		363.14	1,231.01 tons	0.29
.02	" centering for brickwork.....	1,389.62	588.25	1,977.87	8 roasters	247.23
8106.01	Roaster flue, spouts.....	116.52	568.99	685.51	10 spouts	68.55
.02	" " tile work.....	374.82	400.36	775.18	2,365 cu. ft.	0.33
.03	" " painting inside.....	8.50	3.97	12.47	73 sq. yd.	0.17
8107	Shafting, pulleys and belting.....	118.24	1,999.89	2,118.13	164 lin. ft.	12.93
8108	Motor.....	277.08	463.96	741.04	30 h.p.	24.70
8109	Lighting.....	340.64	157.70	498.34	67 drops	7.44
8112	Motor-driven fans.....	77.69	1,405.91	1,483.60	2 fans	741.80
8112.1	Blast pipe from fans to roaster.....	1,569.62	656.62	2,226.24	240 ft.	9.28
8113	Conveyor No. 12.....	164.15	805.05	969.20	51.2 ft.	18.93
.1	" 13 ¹ / ₂ and 13 ¹ / ₂	476.29	3,472.23	3,948.62	217.3 ft.	18.17
.2	Stile over conveyors.....	98.91	199.80	298.71	2.40 tons	124.46
Total cost—Roasting plant.....				\$136,734.87		

8120.—Roaster Dust Chamber

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
8121	Excavation.....	\$918.74	\$194.09	\$1,112.83	1,194 cu. yd.	\$0.93
8122	Foundation.....	1,049.71	1,775.05	2,824.76	472.9 "	5.97
8123	Steel structure.....			34,745.41	415.68 tons	83.59
.01	Wire baffles.....	523.63	4,758.23	5,281.86	604.8 c. wires	8.73
.1	Tile work.....	2,268.29	2,585.45	4,853.74	14,980 cu. ft.	0.33
.11	Unloading tile.....	307.72		307.72	525.05 tons	0.59
.2	Painting outside.....	304.75	115.82	420.57	2,851 sq. yd.	0.15
.3	Painting inside.....	93.50	24.37	117.87	950 "	0.12
Total cost—Roaster dust chamber.....				\$49,664.76		

8300.—Reverberatory Plant

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
8301	Excavation.....	\$1,417.88	\$59.78	\$1,477.66	1,890 cu. yd.	\$0.78
.01	Back filling.....	2,742.44	13.11	2,755.55	3,679 "	0.75
8302	Foundation.....	7,715.96	15,044.09	22,760.05	2,810 "	8.10
.1	Concrete counterweights.	88.22	79.27	167.49	8.5 "	19.71
8303	Steel structures.....			40,799.76	461.09 tons	88.48
8304	Reverberatories, brickwork	8,402.75	34,963.88	43,366.63	29,680 cu. ft.	1.46
.01	" unloading brick.....	934.78	329.71	1,264.49	2,279.49 tons	0.55
.02	" centering.....	859.77	547.78	1,407.55	3 reverbera- tories.....	469.18
.05	" rehandling brick....	1,503.85		1,503.85	1,530.92 tons	0.98
.1	" steel work.....	1,366.38	14,552.71	15,919.09	240 tons	66.33
.2	" silica fill.....	3,715.66	5,689.41	9,405.07	1,231.49 tons	7.64
.3	" hoppers and chutes.	150.08	1,263.59	1,413.67	403.35 cwt.	3.51
8305	Cross and header flues, brickwork.....	3,296.65	15,098.97	18,395.62	18,500 cu. ft.	0.99
.01	" unloading brick.....	403.77	93.83	497.60	1,329.23 tons	0.37
.02	" centering.....	316.84	125.97	442.81	3 sets	147.60
.2	" painting brick.....	46.88	21.45	68.33	473 sq. yd.	0.15
8306	Flues from boilers to re- verberatory, flue ex- cavation.....	19.86		19.86	15 cu. yd.	1.32
.1	" foundation.....	74.22	74.02	148.24	9.2 "	16.11
.2	" steel structure.....			2,815.32	34.78 tons	80.95
8307	Reverb. boiler bldg. exca'n	283.14		283.14	306 cu. yd.	0.93
.01	Waste heat boilers "	213.44		213.44	129 "	1.65
.02	Oil-fired boilers "	73.60		73.60	97 "	0.76
.03	Boiler feed pumps "	591.28	10.16	601.44	659 "	0.91
.04	Backfill, back of boiler wall	518.10	30.00	548.10	972 "	0.56
.1	Boiler building foundations	2,181.08	3,846.18	6,027.26	573.7 "	10.51
.11	Waste heat boilers "	789.52	793.97	1,583.49	138.7 "	11.42
.12	Oil-fired boilers "	350.45	686.29	1,036.74	100 "	10.37
.13	Feed pumps foundation....	678.73	1,428.97	2,107.70	214.7 "	9.82
.2	Floor over slag track cut..	1,018.17	1,687.29	2,705.46	7,676 sq. ft.	0.35
.3	Floor around boilers.....	276.29	664.82	941.11	2,705 "	0.35
8308	Boiler building, steel struc- ture.....			25,839.85	292.03 tons	88.48
.5	Platform and brackets....	1,317.18	2,556.45	3,873.63	29.50 tons	131.30
8309	Waste heat boilers, inc. all steel.....	4,078.19	39,335.30	42,413.49	7 boilers	6,059.07
.01	" " brickwork....	3,878.57	13,749.97	17,628.54	18,025 cu. ft.	0.98
.02	" " unloading brick.....	510.74	8.17	518.91	1,073.74 tons	0.48
.03	" " painting.....	140.53	84.29	224.82	1,190 sq. yd.	0.19
.05	" " rehandling .. brick.....	185.76		185.76	1,135.57 tons	0.16
.1	Oil fired boilers, inc. all steel.....	1,892.22	13,125.78	15,018.00	3 boilers	5,006.00
.11	" " brickwork.....	1,697.31	1,154.62	2,851.93	6,561 cu. ft.	0.43
.12	" " unloading brick..	168.44	15.50	183.94	228.11 tons	0.81
.13	" " painting.....	124.59	24.16	148.75	438 sq. yd.	0.34
.15	" " rehandling brick.	137.06		137.06	413.35 tons	0.33
8310	Superheaters waste heat boilers.....	654.41	8,288.71	8,943.12	7 heaters	1,277.58
.1	" " oil fired boilers.....	280.28	2,756.41	3,036.69	3 boilers	1,012.23
8312	Misc. piping boilers and reverb. bldg.....	524.15	1,409.85	1,934.00		
.1	Feed piping from heating plant to feed pumps.....	1,039.91	51.51	1,091.42	1,296 cu. yd.	0.84
.11	" " from conduit and pipe....	386.25	2,757.79	3,144.04	557 ft.	5.64

8300.—Reverberatory Plant (Continued)

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
8312.20	" " from pumps to boilers.....	1,060.53	3,041.00	4,101.53	1,093 ft.	3.75
.50	Blow-off piping and drum.	145.32	527.77	673.09		
8313	Wiring electrical feed pumps.....	135.45	177.89	313.34	2 pumps	156.67
.1	Lighting for reverb. and boiler bldg.....	612.30	473.19	1,085.49	104 drops	10.45
8314	Slag launders.....	433.20	1,157.80	1,591.00	72 ft.	22.10
8315	Matte launders.....	142.66	1,209.08	1,351.74	304 ft.	4.44
8316	Six No. 14 Wilgus oil systems.....	111.48	1,862.29	1,973.77	6 pumps	328.96
8317	Two electrical feed pumps.	240.25	5,633.97	5,874.22	2 "	2,937.11
.1	One steam feed pump....	37.27	499.24	536.51	1 "	536.51
.2	Crawls and chain block in feed pump house.....	2.00	112.05	114.05		
8318	Fettling system.....	1,463.13	1,943.08	3,406.21	3 furnaces	1,135.40
Total cost—Reverberatory plant.....				3,328,945.02		

8400.—Converter Plant

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
8401	Excavation.....	5,163.74	567.37	5,731.11	6,330 cu. yd.	0.91
8402	Foundation.....	1,796.56	2,860.16	4,656.72	776.9 cu. yd.	6.00
8403	Steel structure.....			69,359.60	783.86 tons	88.48
8404	Converter stands—Exca- vation.....	255.48	3.40	258.88	304 cu. yd.	0.85
.1	" " —foun- dation.....	555.53	1,331.03	1,886.56	173.8 cu. yd.	10.85
8405	" " and shells	821.67	22,238.28	23,059.95	162.53 tons	141.88
.01	Repairs to No. 2 stand...	164.44	9.53	173.97		
.1	Converter shells—brick lining..	785.61	8,043.00	8,828.61	4 shells	2,207.15
.11	" " —unload- ing brick.	104.22	82.03	186.25	579.30 tons	0.32
8406	Cranes.....	1,438.50	23,027.65	24,466.15	110.75 tons	220.91
.1	" wiring.....	1,941.44	342.62	2,284.06	2 cranes	1,142.03
8407	Clinkering machines.....	1,715.23	13,981.94	15,698.17	2 machines	7,640.71
.01	" " altera- tion No. 1	33.00	1.43	34.43		
.02	" " altera- tion No. 2.	57.92	40.39	98.31		
.03	Clinkering machine electri- cal alteration.....	65.98	31.10	97.08		
.1	Clinkering machines, wiring.....	392.21	283.79	676.00	2 machines	338.00
8409	Wiring for converter control.....	136.42	318.49	454.91	3 converters	151.64
.1	Lighting.....	451.93	462.01	913.94	60 drops	15.23
8410	Air pipe from power house excav.....	224.06		224.06	331 cu. yd.	0.68
.1	Air pipe from power house laying.....	674.62	2,041.89	2,716.51	422 ft.	6.43
8411	Ladles, boats, bales, tools, etc.....	906.82	4,932.39	5,839.21		

8400.—Converter Plant (Continued)

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
8413	Casting machine excavation...					
		490.39	30.43	520.82	512 cu. yd.	1.02
8414	" " foundation...	1,627.37	1,889.36	3,516.73	291.9 cu. yd.	12.05
8415	" " cost and erection.	3,266.34	24,211.21	27,477.55	2 machines	13,738.7
.1	" " repairs..	221.71	13.88	235.59		
8416	Loading platform excavation.	212.49	.28	212.77	216 cu. yd.	0.99
.1	" " foundation...	311.35	505.68	817.03	93.2 cu. yd.	8.77
.11	" " floors...	381.02	1,053.55	1,434.57	6,803 sq. ft.	0.21
.2	" " backfill.	67.85		67.85	129 cu. yd.	0.53
.3	" " striking plates..	53.09	126.69	179.78	119 sq. ft.	1.51
8417	Hoods and smoke boxes..	2,674.30	4,012.59	6,686.89	3 sets	2,228.96
.1	Hood to protect converter operator.....	62.67	109.66	172.33	1 hood	172.33
8418	Spouts, gates and hoppers at Silica ore bins.....	245.48	1,400.37	1,645.85		
8419.1	10-ton bullion scales excavation.....	19.76		19.76	24 cu. yd.	0.82
.2	" " " foundation.....	58.32	65.19	123.51	10.5 cu. yd.	11.76
.3	" " " cost and erection.....	55.55	736.53	792.08	10 tons	79.21
.4	" " " scale house.....	48.78	41.09	89.87	1 shed	89.87
8425	Conveyor No. 15.....	422.77	2,251.47	2,674.24	165 ft.	16.21
8426.1	Wet pan excavation.....	2.44		2.44	3 cu. yd.	0.81
.2	" " foundation.....	53.30	55.96	109.26	7.5 cu. yd.	14.57
.3	" " cost and erection.	304.76	1,050.10	1,354.86	1 pan	1,354.86
.4	" " bin and spout...	75.64	180.47	256.11	548 cwt.	4.67
Total cost converter plant				\$216,033.37		

8420.—Converter Dust Chamber

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
8421	Excavation.....	\$127.28		\$127.28	265 cu. yd.	\$0.48
8422	Foundation.....	668.85	1239.17	1,908.02	286.4 cu. yd.	6.66
8423	Steel structure.....			20,371.20	238.30 tons	85.49
.01	Wire baffles.....	138.86	1,101.95	1,240.81	166.10 c. wire	7.47
.1	Tile work.....	1,620.05	2,182.40	3,802.45	6,369 cu. ft.	0.60
.11	Tile handling.....	37.61		37.61	155.20 tons	0.24
8424	Iron doors and frames....	1.94	158.93	160.87		
8428	Smoke box track.....	155.32	10.02	165.34		
Total cost converter dust chamber.....				\$27,813.58		

8500.—Conveying System

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
8501	Excavation.....	\$1,824.22		1,824.22	2,286 cu. yd.	\$0.80
8502	Foundation.....	2,550.78	4,006.37	6,557.15	622.3 cu. yd.	10.54
8503	Steel structure.....			19,365.98	211.73 tons	91.47
8504	Woodwork.....	589.51	630.42	1,219.93	13.86 m.b.m.	88.02
.1	Floor battens.....	146.29	51.47	197.76		
8505	Conveyors No.'s 3-4-5-6 11-14.....	1,210.87	12,505.62	13,716.49	1,284.9 ft.	10.68
.1	Chutes.....	665.15	167.13	832.28	23.3 cwt.	3.67
.2	Guides.....	28.00		28.00		
.3	Weightometer.....	65.00	1,329.92	1,394.92	1 weighto- meter	1,994.92
8506	Lighting.....	189.86	84.56	274.42	33 drops	8.32
Total cost—Conveying sys- tem.....				\$4,5411.15		

8600.—Chimney

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
8601	Excavation.....	\$337.44	\$29.61	\$367.05	597 cu. yd.	\$0.61
8602	Foundation.....	654.42	4,199.65	4,854.07	872.7 cu. yd.	5.56
8603	Brickwork.....	891.88	39,358.34	40,250.22	58,644 cu. ft.	0.69
Total cost chimney.....				\$45,471.34		

8610.—Reverberatory Flue

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
8611	Excavation.....	\$916.72	\$20.21	\$936.93	1,588 cu. yd.	\$0.59
8612	Foundation.....	1,657.09	3,886.80	5,543.89	487.8 cu. yd.	11.37
8613	Brickwork.....	1,272.86	1,696.39	2,969.25	6,400 cu. ft.	0.46
.01	Unloading brick.....	57.79		57.79	278.33 tons	0.21
8614	Steel structure.....			3,593.06	41.61 tons	86.35
.1	Clean out doors.....	10.61	153.61	164.22		
.2	Caulking roof.....	184.35	4.21	188.56		
Total cost—Reverberatory flue.....				\$13,453.70		

8620.—Converter Flue

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
8621	Excavation.....	\$168.02		\$168.02	198 cu. yd.	\$0.85
8622	Foundation.....	165.73	652.69	818.42	142 cu. yd.	5.76
8624	Steel structure.....			6,616.44	81.99 tons	80.70
Total cost—Converter flue				\$7,602.88		

8625.—Roaster Dust Chamber Flue

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
8626	Excavation.....	\$225.37		\$225.37	213 cu. yd.	\$1.06
8627	Foundation.....	224.35	551.23	775.58	114.6 cu. yd.	6.77
8628	Brickwork.....	1,018.13	1,573.06	2,591.19	4,231 cu. ft.	0.61
.01	Unloading brick.....	57.60		57.60	171.40 tons	0.34
8629	Steel structure.....			9,209.36	94.46 tons	97.49
Total cost—Roaster dust chamber flue.....				\$12,859.10		

8700.—Boiler and Blacksmith Shop

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
8701	Excavation.....	\$1,142.07	\$44.81	\$1,186.88	1,458 cu. yd.	\$0.81
8702	Foundation.....	416.57	584.49	1,001.06	78.7 cu. yd.	12.71
8703	Steel structure.....			2,913.90	32.72 tons	89.06
.1	Doors, windows and frames.....	693.02	2,456.28	3,149.30	2,581 sq. ft. opening	1.22
.11	Concrete sills.....	119.80	50.87	170.67	251.5 lin. ft.	0.68
.2	Tile walls.....	477.95	612.62	1,090.57	2,297 cu. ft.	0.48
.21	Unloading tile.....	18.89		18.89	69.70 tons	0.27
.22	Coping.....	112.17	2.72	114.89	290 lin. ft.	0.40
.30	Roof.....	286.52	828.24	1,114.76	66.49 squares	16.77
8703.31	Ventilators.....	16.01	261.50	277.51	3 vents	92.50
.40	Dirt floor.....	59.73	1.25	60.98		
.50	Benches.....	87.83	49.43	137.26		
.60	Painting.....	92.53	60.73	153.26	1,574 sq. yd.	0.10
8704	Crane.....	119.60	438.41	558.01	1 crane	558.01
8705	Tools.....	798.51	7,859.36	8,655.87		
8706	Shafting, pulleys, belt- ing.....	105.59	301.16	406.75	51 lin. ft.	7.98
8707	Motor.....	23.22	347.54	370.76	20 h.p.	18.54
8708	Lighting.....	23.41	44.50	67.91	17 drops	4.00
Total cost—Boiler and blacksmith shop.....				\$21,449.23		

8714.—Machine and Carpenter Shop

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
8715	Excavation.....	\$1,615.83	\$325.28	\$1,941.11	1,765 cu. yd.	\$1.10
8716	Foundation.....	792.05	584.06	1,376.11	105.5 cu. yd.	13.04
8717	Steel structure.....			3,431.42	38.23 tons	89.76
.1	Doors, windows and frames.....	923.61	2,992.16	3,915.77	3,037 sq. ft. opening	1.29
.11	Concrete sills.....	111.65	67.70	179.35	295.3 lin. ft.	0.61
.2	Tile walls.....	531.45	571.28	1,102.73	2,397 cu. ft.	0.46
.21	Unloading tile.....	42.06	2.00	44.06	58.80 tons	0.75
.22	Wall coping.....	121.67	23.70	145.37	320 lin. ft.	0.45
.30	Roof.....	297.85	953.04	1,250.89	77.21 squares	16.20
.31	Ventilators.....	11.16	248.24	259.40	3 ventilators	86.45
.40	Floor.....	269.80	593.30	863.10	4,136 sq. ft.	0.21
.50	Benches.....	130.00	35.00	165.00		
.60	Painting.....	118.00	87.40	205.40	1,989 sq. yd.	0.10
8718	Crane.....	25.19	564.36	589.55		
8719	Tools.....	444.07	8,953.13	9,397.20		
8720	Shafting, pulleys and belting.....	289.29	1,513.36	1,802.65	152 lin. ft.	11.86
8721	Motor.....	18.34	477.97	496.31	40 h.p.	12.40
8722	Lighting.....	55.84	135.01	190.85	20 drops	9.54
Total cost—Machine and carpenter shop.....				\$27,356.27		

8800.—General Office

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
8804	Furniture and fixtures..		\$1,394.95	\$1,394.95		

8808.—Warehouse

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
8810	Excavation.....	\$944.59	\$51.49	\$996.08	1,287 cu. yd.	\$0.77
8811	Foundation.....	878.16	856.09	1,734.25	123 cu. yd.	14.09
8812	Steel structure.....			3,734.08	39.76 tons	93.92
.1	Doors, windows and frames	533.02	1,066.31	1,589.33	1,982 sq. ft. opening	0.80
.11	Concrete sills.....	164.72	61.63	226.35	241.5 lin. ft.	0.94
.2	Tile walls.....	438.00	477.86	915.86	2,342 cu. ft.	0.39
.21	Unloading tile.....	15.50	1.00	16.50	74.20 tons	0.22
.22	Coping.....	176.60	36.53	213.13	320 lin. ft.	0.67
.3	Painting roof.....	81.16	65.66	146.82	813 sq. ft.	0.18
.31	Ventilators.....	30.38	207.12	237.50	3 vents	79.17
.4	Floor excavation.....	129.03		129.03	66 cu. yd.	1.96
.41	" concrete.....	558.04	721.60	1,279.64	8,298 sq. ft.	0.15
.5	Lighting.....	45.09	70.48	115.57	26 drops	4.45
8813	Warehouse fixtures.....	548.66	1,541.12	2,089.78		
.1	Painting.....	26.50	14.17	40.67	412 sq. yd.	0.10
.11	" sash.....	122.78	15.34	138.12	189 sash	0.73
Total cost—Warehouse.....				\$13,602.71		

8819.—Laboratory

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
8820	Excavation.....	\$191.11	\$0.16	\$191.27	212 cu. yd.	\$0.90
8821	Foundation.....	448.42	575.13	1,023.55	96.5 cu. yd.	10.61
.1	Plain concrete floors.....	113.78	154.90	268.68	1,026 sq. ft.	0.26
.2	Reinforced floors.....	59.81	114.87	174.68	364 sq. ft.	0.48
.3	Sills and lintels.....	109.53	22.85	132.38	163 lin. ft.	0.81
8822	Tile walls.....	250.71	517.66	768.37	841 cu. ft.	0.91
.2	Carpenter work.....	194.43	338.16	532.59		
.5	Doors, windows and skylights	208.70	480.33	689.03	823 sq. ft. opening	0.84
8824	Wood fixtures.....	486.90	204.02	690.92		
8825	Lighting.....	226.77	61.28	288.05		
8826	Plumbing.....	129.79	97.23	227.02		
8828	Painting.....	151.46	55.05	206.51		
8829	Plastering.....	58.11	30.55	88.66	171.5 sq. yd.	0.52
8830	Apparatus.....	44.82	618.98	663.80		
8831	Oil centrifuge.....	40.92	157.59	198.51		
Total cost—Laboratory.....				\$6,144.02		

8840.—Sample Room

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
8841	Excavation.....	\$61.35		\$61.35	72 cu. yd.	\$0.85
8842	Foundation.....	61.00	64.43	125.43	9 cu. yd.	13.94
.1	Concrete floors.....	63.17	75.25	138.42	489 sq. ft.	0.28
8843	Walls and roof structure.....	101.84	201.81	303.65		
.4	Roof.....	23.00	76.57	99.57	8 squares	12.45
.5	Doors and windows.....	32.39	118.85	151.24	298 sq. ft. opening	0.51
8844	Oven.....	233.74	58.56	292.30		
8845	Benches, motor platform and fixtures.....	128.35	129.60	257.95		
8846	Lighting.....	22.97	52.83	75.80	7 drops	10.82
8848	Painting.....	28.00	8.00	36.00	26 sash	1.38
8849	Machinery.....	53.07	912.00	965.07		
.1	Motor!.....	9.00	116.77	125.77	5 h.p.	25.15
.2	Shafting, pulleys and belting...	23.39	170.17	193.56	26 lin. ft.	7.44
Total cost—Sample room.....				\$2,826.11		

8900.—Miscellaneous Accounts

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
8902	Sewer system, cost of pipe and laying.....	\$778.83	\$1,224.72	\$2,003.55	2,967 ft.	\$0.68
.1	" excavation.....	2,122.84	65.20	2,188.04	2,967 ft.	0.74
.2	" concrete.....	168.18	184.08	352.26	53.8 ft.	6.55
8903	Outside closets.....	879.05	227.78	1,106.83	3 closets	368.94
8905	Permanent outside lighting.....	183.02	177.99	361.01	5 arcs	72.20
8906	Water pipe lines excavation....	868.11		868.11	4,253 ft.	0.20
.01	" concrete.....	17.37	17.86	35.23	2.3 cu. yd.	15.32
.02	" cost and laying.....	2,863.32	2,062.07	4,925.39	4,253 ft.	1.16
.1	6-in. pipe line to Clifton.....	1,474.71	6,914.95	8,389.66	8,988 ft.	0.93
.2	Water supply tank, excavation..	143.68		143.68	116 cu. yd.	1.24
.4	" cost and erection.....			4,137.03	33.67 tons	122.87
8908	Power distribution.....	3,233.02	7,407.21	10,640.23	17,370 ft.	0.61
8909	Permanent air line, excavation..	267.50		267.50	401 cu. yd.	0.67
.1	" laying.....	432.37	623.08	1,055.45	2,316 ft.	0.46
8961	Steam heating system, excavation.....	166.36		166.36	225 cu. yd.	0.73
.1	" cost and installation.....	240.78	305.37	546.15	496 ft.	1.10
Total cost—Miscellaneous accounts.....				\$37,186.48		

8999.—Charges to Indirect Expense

NUMBER	NAME OF ACCOUNT	TOTAL	TOTAL	TOTAL	TOTAL
7001	General expense at Clifton	\$13.58	8941	Temporary railway receiving bins.....	243.78
7004	Personal injuries.....	6,734.01	8942	Water supply.....	2,372.63
8901	Derricks and construction equipment.....	18,718.35	8943	Corral expense.....	
8904	Telephone system.....	3,229.40	8944	Switching and freight from Clifton.....	2,661.35
8905.1	Temporary outside lighting.....	18.30	8945	Office stationery and supplies.....	1,369.68
8907	Watchman.....	1,516.69	8946	Warehouse operating expense.....	10,771.88
8908.1	Temporary oil tanks.....	382.75	8947	Time-keeping expense....	4,346.59
8910	Transmission of power to various departments....		8948	Form lumber.....	
8911	Watchman's house.....	185.39	8949	Cement.....	
8912	Tool shed.....	725.75	8951	Sand and gravel.....	
8913	Barn and corral.....	2,100.53	8952	Employes quarters.....	2,192.98
8914	Temporary blacksmith shop.....	251.71	8953	Crushing plant operating expense.....	
8916	" power plant.....	5,518.11	8954	Concrete, power and repairs	
8917	" crushing plant.....	4,555.07	8955	Mortar sand.....	
8918	" water tank.....	1,991.31	8955.1	Mortar lime.....	
8919	" electrical shop equipment.....	322.16	8955.2	Mortar cement.....	
8920	Wagon roads.....	1,215.33	8955.4	Fire brick mortar.....	
8921	Temporary pumping plant	375.25	8955.5	Silica brick mortar.....	
8922	" pipe lines.....	5,199.73	8956	Operating temporary power house.....	
8923	" warehouse.....	1,810.24	8957	Maintenance of tracks in yards.....	2,800.31
8924	" cement sheds.....	801.10	8958	Ditch at tunnel No. 2....	5,851.89
8925	Horses, harness and carts.	2,646.40	8975	Cleaning up.....	4,996.20
8926	Temporary office.....	1,195.86	8976	Rehandling brick and tile.	53.02
8927	" closets.....	146.65	8998	Direct charges.....	4,316.30
8928	" machine shop.....	166.10			
8929	Employee railroad transportation.....	16,994.57	8999	Total charges to indirect expense.....	\$140,277.72
8930	Clearing land.....	456.67		Total unit cost.....	7.53 per cent.
8931	Test holes.....	109.48		This percentage is obtained by dividing the total charges to indirect expense by the total cost of the smelter, minus engineering and indirect expense.	
8933	Furniture and fixtures....	365.25		\$140,277.72	
8934	Miscellaneous supplies....	4,436.24		1,864,092.47	
8935	Shop equipment.....	657.68			
8936	Overhead shop expense..	8,394.19			
8937	Stock lumber.....				
8938	Powder magazine.....	241.37			
8939	Miscellaneous labor.....	6,825.89			

9000.—Power Plant

Power House

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT Cost
9000.1	Power plant engineering					See page 1571
9001	Excavation	\$7,727.56	\$69.09	\$7,796.65	7,313 cu. yd.	\$1.07
9002	Bldg. foundation piers	1,699.92	1,460.02	3,159.94	231.7 cu. yd.	13.64
.1	" " walls	3,735.78	3,628.81	7,364.59	508.5 cu. yd.	14.48
.2	North tunnel	1,350.79	1,230.37	2,581.36	180.3 cu. yd.	14.32
.3	Concrete drain	205.68	227.37	433.05	34.6 cu. yd.	12.52
.4	Basement floor, concrete	916.41	1,347.78	2,264.19	12,130 sq. ft.	0.19
.45	" painting	81.45	48.81	130.26	830 sq. yd.	0.16
.6	Preparation of concrete for painting	891.73	42.69	934.42	2,459 sq. yd.	0.38
.7	Painting concrete	195.84	301.61	497.45	2,459 sq. yd.	0.20
9003	Steel structure			23,773.30	254.29 tons	93.49
.1	Tile walls	3,856.83	4,510.20	8,367.03	14,343 cu. ft.	0.58
.11	Unloading tile	332.40	0.17	332.57	522.70 tons	0.64
.12	Wall coping	372.69	107.05	479.74	732 lin. ft	0.66
.2	Doors, window and frames	974.38	3,319.93	4,294.31	4,044 sq. ft. opening	1.06
9003.21	Concrete sills	596.33	120.96	717.29	964 lin. ft.	0.74
.3	Ventilators	125.60	439.76	565.36	6 ventilators	94.23
.4	Main floor columns	236.93	626.44	863.37	68 columns	12.70
.41	Main floor slab concrete	1,267.91	3,341.61	4,609.52	10,210 sq. ft.	0.45
.42	Painting under side main floor	181.88	147.58	329.46	2,679 sq. yd.	0.12
.43	" top main floor	95.56	199.32	294.88	1,134 sq. yd.	0.26
.5	Roof, Berger multiplex plate	420.83	3,063.18	3,484.01	214.83 squares	16.22
.51	" concrete	1,723.10	958.51	2,681.61	214.83 "	12.48
.52	" tar	172.70	127.73	300.43	214.83 "	1.40
.53	" down spouts and tile drain	286.17	240.44	526.61	905 ft.	0.58
.54	" painting under-side	692.84	324.55	1,017.39	6,813 sq. yd.	0.15
.55	" P. & B. roofing	577.68	1,317.08	1,894.76	214.83 squares	8.82
.60	Painting sash	290.09	16.72	306.81	299 saah	1.02
.61	Painting woodwork	29.50	4.06	33.56	89 sq. yd.	0.38
9004	Crane	131.89	1,723.27	1,855.16	1 crane	1,855.16
9005	Well grading	1,558.07	517.68	2,075.75	2,600 cu. yd.	0.80
.1	Shaft sinking	765.62	612.10	1,377.72	45 ft.	30.61
.2	Timbering	57.61		57.61	45 ft.	1.28
.31	Aldrich pump installation	74.56	16.62	91.18		
9006.01	Nordberg blowers, foundation	774.06	3,020.83	3,794.89	686.3 cu. yd.	5.53
.1	" cost and installation	1,641.62	32,514.02	34,155.64	2 Nordbergs	17,077.82
.2	" painting	327.57	57.65	385.22	2 "	192.61
9007.01	Turbines, foundation	959.08	1,432.70	2,391.78	196.5 cu. yd.	12.16
.1	" cost and installation	2,297.70	79,586.49	81,884.19	3 turbines	27,294.73
.2	" painting	286.15	41.02	327.17	3 "	109.06
.3	" air pipe making	547.68	200.75	748.43	103 ft.	6.27
.4	" air pipe erection	232.57	64.24	296.81	103 ft.	2.88
.51	Transformer trucks and transfer table	121.63	538.08	659.71	15 trucks	43.98
.52	Auto transformers	735.60	12,044.91	12,780.51	10 transformers	1,278.05
9008.01	Condensers foundations	291.08	285.18	576.26	50.3 cu. yd.	11.45

9000.—Power Plant (Continued)

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
9008.1	" cost and installation.....	415.31	19,563.55	19,978.86	3 condensers	6,659.62
.2	" painting.....	30.00	5.86	35.86	3 "	11.95
9009	Jet condenser hot well, excavation.....	28.82	0.90	29.72	46 cu. yd.	0.65
.01	" foundation.....	66.27	69.99	136.26	16.5 "	8.26
.02	" supporting struct. and tank.....			945.74	5.76 tons	164.19
.03	" cost and erection	128.97	949.68	1,078.65	1 condenser	1,078.65
.12	" dry vacuum pumps....	285.51	2,860.01	3,145.52	2 pumps	1,572.76
.13	" " painting...	30.00	5.86	35.86	2 "	17.93
.21	Circulating pumps foundation.....	560.04	708.93	1,268.97	210 cu. yd.	6.04
.22	" cost and erection	366.90	3,535.68	3,902.58	2 pumps	1,951.29
.23	" painting.....	30.00	5.86	35.86	2 "	17.93
9010.01	Air compressor foundation.....	840.98	1,246.54	2,087.52	238.3 cu. yd.	8.76
.02	" erection.....	642.90	148.67	791.57		
.03	" painting.....	10.58	24.49	35.07		
.04	" all piping except steam.....	298.46	160.65	459.11		
.05	" wrecking and transportation...	457.77	136.06	593.83		
.06	" installation of air receivers.....	49.47	1.43	50.90		
9011.01	2 exciters, 2 air pumps, 2 cir. pumps, found'n	1,439.67	1,875.43	3,315.10	373 cu. yd.	8.89
.02	2 exciters, cost and installation.....	491.01	6,118.26	6,609.27	2 exciters	3,304.64
.03	3 dry vacuum pumps, cost and installation	147.26	3,190.10	3,337.36	3 pumps	1,112.45
.04	3 cir. pumps and engines, cost and installation.....	389.32	8,729.37	9,118.69	3 "	3,039.56
.05	2 exciters, painting....	86.01	14.65	100.66	2 exciters	50.33
.06	3 air pumps, "	50.00	8.79	58.79	3 pumps	19.59
.07	3 cir. " "	81.69	14.65	96.34	3 "	32.11
9012.01	2 motor gen., 1 air pump, 1 cir. pump., foundation.....	296.52	658.91	955.34	107 cu. yd.	8.93
.02	2 motor generators, cost and installation.....	319.06	6,830.33	7,149.39	2 generators	3,574.69
.05	" painting.....	30.00	5.86	35.86	2 "	17.93
9013	Transfer table pit, concrete.....	24.13	58.23	82.36	12 cu. yd.	6.86
.01	Switchboard concrete compartments.....	1,472.21	510.48	1,982.69	1,469 sq. ft.	1.35
.02	" cost and erection	2,730.53	15,520.57	18,251.10		
9014	Steam piping north and south mains, excavation...	249.65		249.65	279 cu. yd.	0.89
.01	" " foundation...	578.24	945.97	1,524.21	194.5 cu. yd.	7.84
.02	" steel supporting structure.....			7,694.58	86.81 tons	88.64
.03	" hangers and anchors.....	1,030.68	337.26	1,367.94	153 rods	8.94
.04	" cost and erection	2,286.31	18,622.25	20,908.56	3,401 ft.	6.15
.05	" covering and erection.....	266.71	5,813.23	6,079.94	3,401 ft.	1.79

9000.—Power Plant (Continued)

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
9015	Exhaust pipe, cost and erection.....	1,745.71	8,715.66	10,461.37	1,541 ft.	6.79
.01	" painting.....	85.05	51.19	136.24	1,541 ft.	0.09
.05	" covering and erection.....	318.25	830.56	1,148.81	746 ft.	1.54
.10	Air piping, cost and erection.....	363.19	554.16	917.35		
.11	" painting.....	31.56	18.66	50.22		
.2	Exhaust pipe foundation.....	63.09	102.81	165.90	18.3 cu. yd.	9.07
.21	" supporting structure.....	197.27	57.93	255.20		
.22	" excavation.....	20.82		20.82	29 cu. yd.	0.72
9016	Water pipe, excavation and backfill.....	1,485.10	0.24	1,485.34	2,406 cu. yd.	0.62
.01	" cost and erection.....	3,747.79	16,437.88	20,185.87		
.02	" painting.....	230.59	25.54	256.13		

Feed-water Heating Power

9017	Excavation.....	239.39	1.70	241.09	274 cu. yd.	10.88
.01	Foundation.....	708.06	551.04	1,259.10	105.4 cu. yd.	1.95
.015	Reinforced floors.....	282.45	273.51	555.97	1,330 sq. ft.	0.42
.016	Water-proofing concrete tanks.....	80.65	14.46	95.11	1,475 sq. ft.	80.06
.02	Steel structure.....			2,262.90	26.63 tons	4.98
.021	Distributing and equalizing tank.....	364.70	260.58	625.28	80.2 cwt.	7.80
.03	Tile work.....	285.99	234.36	520.35	706 cu. ft.	0.74
.031	Unloading tile.....	24.91		24.91	30 tons	0.83
.032	Coping.....	14.83	14.01	28.84	108 lin. ft.	0.27
.033	Sills and lintels.....	7.81	5.57	13.38	60 lin. ft.	0.22
.034	Painting tile walls.....	26.10		26.10	112 sq. yd.	0.23
9017.035	Doors, windows and frames.....	59.83	99.98	159.81	186 sq. ft. opening	0.86
.04	Roofing.....	115.25	109.88	225.13	8.8 squares	25.58
.045	Ventilators.....	142.14	49.20	191.34	2 ventilators	95.67
.05	Treating tank, concrete	637.67	487.19	1,124.86	28.7 cu. yd.	39.19
.06	Receiving tank No. 1.	251.55	167.92	419.47	7.3 cu. yd.	57.46
.07	Receiving tank No. 2..	364.50	265.08	629.58	24.4 cu. yd.	25.80
.08	Calibrating tank.....	188.96	50.21	239.17	12.6 cwt.	18.98
.081	Tipping meter.....	372.46	227.50	599.96	1 tipping meter	599.96
.09	Heaters, recorder.....	685.22	2,813.84	3,499.06		
.1	Sewer excavation and backfill.....	157.19		157.19	266 cu. yd.	0.59
.11	Sewer pipe, cost and laying.....	71.88	203.00	274.88	100 ft.	2.75
.12	Lighting.....	53.35	25.83	79.18	6 drops	13.20
.13	Painting.....	62.78	20.35	83.13		
.14	Wood walkway and tank covers.....	67.08	51.31	118.39	1.56 m.b.m.	75.89
.15	Alterations.....	99.70	3.40	103.10		

Condensed Water Pump House

.20	Excavation.....	220.56	8.93	229.49	236 cu. yd.	0.97
.21	Foundation.....	1,171.89	854.68	2,026.57	171 cu. yd.	11.85
.22	Floor.....	78.41	57.70	136.11	355 sq. ft.	0.38
.24	Doors, windows and frames.....	26.02	22.68	48.70	57 sq. ft. opening	0.85
.242	Tile work.....	98.87	84.28	183.15	257 cu. ft.	0.71
.243	Coping.....	14.50	2.73	17.23	57 lin. ft.	0.30
.25	Roof.....	73.92	60.83	134.75	5 squares	26.95
.26	Pumps and piping....	90.58	691.37	781.95	2 pumps	390.98
.27	Lighting.....	221.16	51.53	272.69		

Power House Miscellaneous Accounts

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
9018.1	Power and lighting transformers.....	397.70	4,714.33	5,112.03	7 transformers	730.29
9019	Lighting.....	307.86	1,449.97	1,757.83	94 drops	18.70
9020	Oiling system.....	1,051.82	401.69	1,453.51	104 outlets	13.98
9021	Benches, bolt racks, etc.	161.22	51.80	213.02		
9022	Instruments and gauges.....	57.97	343.06	401.03		

Cooling Tower

9050	Excavation.....	1,590.85	83.47	1,674.32	1,589 cu. yd.	1.05
.01	Backfill.....	1,395.20		1,395.20	2,415 cu. yd.	0.58
9051	Foundations, sumps and gutters.....	5,212.31	5,640.13	10,852.44	706.3 cu. yd.	15.37
.02	Floor.....	911.93	1,985.03	2,896.96	17,116 sq. ft.	0.17
.03	Water proofing concrete.....	588.76	130.59	719.35	1,606 sq. yd.	0.45
9052	Woodwork.....	3,181.33	5,415.47	8,596.80	128.63 m.b.m.	66.84
9053	Alterations.....	115.40	22.54	137.94		

Total cost—Cooling tower..... \$26,273.01

Total cost—Power plant..... \$434,703.15

9060.—Oil Supply Sump and Pump House

NUMBER	NAME OF ACCOUNT	LABOR	MATERIAL	TOTAL	QUANTITY	TOTAL UNIT COST
9060	Excavation.....	\$1,148.78	\$108.24	\$1,257.02	1,308 cu. yd.	\$0.96
.01	Concrete work.....	2,838.72	3,230.46	5,569.18	340.6 cu. yd.	16.35
.02	Pumps.....	176.20	2,035.58	2,211.78	2 pumps	1,105.89
.03	Inlet piping.....	44.77	126.55	171.32	108 ft.	1.59
.04	Lighting.....	79.71	62.57	142.28	4 drops	35.57
.05	Steel work.....	96.94	129.39	226.33	1.12 tons	202.08
.06	Doors, windows and frames	83.60	65.02	148.62	124 sq. ft. opening	1.20
.07	Roof.....	136.81	162.66	299.47	6.5 squares	46.07
.075	Ventilators.....	95.75	64.32	160.07	2 ventilators	80.03

Two 500,000 Gallon Oil Tanks

9060.10	Wrecking and transportation.....	934.00	465.31	1,399.31	64.40 tons	21.73
.11	Excavation.....	308.20		308.20	554 cu. yd.	0.56
.12	Foundation.....	128.70	210.61	339.31	32.8 cu. yd.	10.35
.13	Erection.....	3,602.89	429.47	4,032.36	64.40 tons	62.62
.131	Roof supports.....	362.90	359.83	722.73	65.56 squares	11.02
.132	Sheathing, lath and plaster	399.05	523.01	922.05	75.50 squares	12.22

Tracks at Oil Sump

9060.14	Railroad grading.....	1,477.27		1,477.27	2,439 cu. yd.	0.61
.15	Laying and ballasting.....	1,107.38	1,092.99	2,200.37	1,362 ft.	1.62
.16	Track bumpers.....	246.74	47.92	294.66	3.16 m.b.m.	93.25
.17	Bridges over wood pipe...	229.22	87.14	316.36		

Oil Supply Tanks for Reverberatories and Boilers

9060.20	Excavation.....	392.13	13.18	405.31	404 cu. yd.	1.00
.21	Foundation.....	875.70	1,685.76	2,561.46	189.5 cu. yd.	13.52
.22	Cost and erection.....			3,926.67	8 tanks	487.94
.23	Piping.....	199.41	282.16	481.57	785 ft.	0.61

Oil Piping

9060.40	Excavation.....	990.73	1.39	992.12	1,150 cu. yd.	0.86
.41	Pipe and laying.....	3,156.14	5,654.50	8,810.64	1,888 ft.	4.67
9060.50	Heating installation.....	167.37	1,068.04	1,235.41	360 ft.	3.43

Total cost—Oil supply system..... \$40,611.88

NEW SMELTER

Recapitulation of Costs

NUMBER	NAME OF ACCOUNT	TOTAL	NUMBER	NAME OF ACCOUNT	TOTAL
7100	Engineering expense....	\$100,849.88	8625	Roaster dust chamber flue.....	12,859.10
7300	Yard tracks and industrial system.....	156,326.43	8700	Boiler and blacksmith shop.....	21,449.23
7400	Receiving bins.....	44,185.06	8714	Machine and carpenter shop.....	27,356.27
7700	Crushing plant.....	9,268.62	8800	General office.....	1,394.95
7800	Sampling plant.....	34,108.74	8809	Warehouse.....	13,602.71
7900	Bedding plant and bunker bins.....	150,939.05	8819	Laboratory.....	6,144.02
8100	Roasting plant.....	136,734.87	8840	Sample room.....	2,826.11
8120	Roaster dust chamber..	49,664.76	8900	Miscellaneous accounts..	37,186.48
8300	Reverberatory plant....	328,945.02	8999	Indirect expense.....	140,277.72
8400	Converter plant.....	216,033.37	9000	Power plant.....	434,703.15
8420	Converter dust chamber	27,813.58	9060	Oil supply sump and pump house.....	40,611.88
8500	Conveying system.....	45,411.15			
8600	Chimney.....	45,471.34			
8610	Reverberatory flue.....	13,453.70			
8620	Converter flue.....	7,602.88			
				Total cost.....	\$2,105,020.07

CHAPTER II

COMPARATIVE COSTS

Plain Concrete

NUMBER	NAME OF ACCOUNT	Cu. Yd.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER Cu. Yd.	AMOUNT	PER Cu. Yd.	AMOUNT	PER Cu. Yd.
7308.1	Trestle approach to reverberatory.....	254.9	\$738.18	\$2.90	\$1,152.42	\$4.52	\$1,890.60	\$7.42
7309.1	Track scales, receiving yard.....	186.0	545.62	2.93	1,146.87	6.17	1,692.49	9.10
7310	Bridge No. 1.....	339.8	1,028.26	3.03	1,968.38	8.82	2,996.64	8.82
7312.1	Retaining wall.....	203.5	512.34	2.52	734.25	3.61	1,246.59	6.13
7313.1	Calcline track scales..	41.6	207.55	4.99	193.60	4.65	401.15	9.64
7314.1	Approach to receiving bins.....	754.3	2,408.16	3.19	3,017.95	4.00	5,426.11	7.19
7402	Receiving bins.....	612.3	1,235.51	2.02	2,247.70	3.67	3,483.21	5.69
7802	Sampling plant.....	120.7	605.91	5.02	649.44	5.38	1,255.35	10.40
8102	Roaster plant.....	250.5	765.42	3.06	1,069.63	4.27	1,835.05	7.33
8122	Roaster dust chamber	472.9	1,049.71	2.22	1,775.05	3.75	2,824.76	5.97
8306.1	Flues from boilers to reverberatory flue.....	9.2	74.22	8.07	74.02	8.04	148.24	16.11
8404.1	Converter stands.....	173.8	555.53	3.19	1,331.03	7.66	1,886.56	10.85
8419.2	Bullion scales.....	10.5	58.32	5.55	65.19	6.21	123.51	11.76
8422	Converter dust chamber	286.4	668.85	2.33	1,239.17	4.33	1,908.02	6.66
8426.20	Wet pan.....	7.5	53.30	7.10	55.96	7.46	109.26	14.56
8602	Chimney.....	872.7	654.42	0.75	4,199.65	4.81	4,854.07	5.56
8622	Converter flue.....	142.0	165.73	1.17	652.69	4.59	818.42	5.76
8627	Roaster dust chamber flue.....	114.6	224.35	1.96	551.23	4.81	775.58	6.77
8702	Boiler and blacksmith shop.....	78.7	416.57	5.29	584.49	7.42	1,001.06	12.71
8716	Machine and carpenter shop.....	105.5	792.05	7.51	584.06	5.53	1,376.11	13.04
8811	Warehouse.....	123.0	878.16	7.14	856.09	6.96	1,734.25	14.09
8842	Sample room.....	9.0	61.00	6.78	64.43	7.16	125.43	13.94
8901	Derricks and cons. equipment.....	53.8	168.18	3.13	184.08	3.42	352.26	6.55

Plain Concrete (Continued.)

NUMBER	NAME OF ACCOUNT	CU. YD.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER CU. YD.	AMOUNT	PER CU. YD.	AMOUNT	PER CU. YD.
8906.01	Water pipe lines, etc.	2.3	17.37	7.55	17.85	7.76	35.23	15.31
8908.2	Power conduits.....	27.6	198.14	7.18	217.15	7.87	415.29	15.04
8908.5	Power conduit branches.....	5.0	25.36	5.07	30.63	6.12	55.99	11.19
8916	Temporary power plant.....	14.4	56.04	3.89	98.69	6.86	154.73	10.75
8917	" crushing plant..	101.1	303.15	3.00	534.13	5.28	837.28	8.28
8918	" oil and water tanks.....	48.8	149.27	3.06	263.01	5.39	412.28	8.45
9002	Power plant.....	231.7	\$1,699.92	\$7.34	\$1,460.02	\$6.30	\$3,159.94	\$13.64
9002.3	" drain.....	34.6	205.68	5.94	227.37	6.57	433.05	12.52
9006.01	" Nordberg engines.....	686.3	774.06	1.13	3,020.83	4.40	3,794.89	5.53
9007.01	" turbines.....	196.5	959.08	4.88	1,432.70	7.29	2,391.28	12.16
9008.01	" condensers.....	50.3	291.08	5.79	285.18	5.67	576.26	11.45
9009.01	" jet condenser...	16.5	66.27	4.02	69.99	4.24	136.26	8.26
9009.21	" 2 air pumps....	210.0	560.04	2.67	708.93	3.37	1,268.97	6.04
9010.01	" air compressor...	238.3	840.98	3.53	1,246.54	5.23	2,087.52	8.76
9011.01	" exciters, etc....	373.0	1,439.67	3.86	1,875.43	5.03	3,315.10	8.89
9012.01	" motor generators.....	107.0	296.52	2.77	658.91	6.16	955.43	8.93
9014.01	North and south steam mains.....	194.5	578.24	2.97	945.97	4.86	1,524.21	7.84
9015.2	Power plant exhaust pipe.....	18.3	63.09	3.45	102.81	5.62	165.90	9.07
Total.....		7,779.4	\$22,391.30	\$2.85	\$37,593.53	\$4.82	\$59,984.83	\$7.67

Miscellaneous Concrete

NUMBER	NAME OF ACCOUNT	CU. YD.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER CU. YD.	AMOUNT	PER CU. YD.	AMOUNT	PER CU. YD.
7702	Crusher plant.....	220.5	\$893.15	\$4.05	\$1,568.47	\$7.11	\$2,461.62	\$11.16
7902	Bedding plant.....	2,809.7	6,256.34	2.23	14,513.21	5.16	20,769.55	7.39
8302	Reverberatory plant	2,810.0	7,715.96	2.74	15,044.09	5.35	22,760.05	8.10
8307.11	Waste heat boilers..	138.7	789.52	5.69	793.97	5.72	1,583.49	11.42
8307.12	Oil-fired boilers.....	100.0	350.45	3.50	686.29	6.86	1,036.74	10.37
8307.13	Feed pumps.....	214.7	678.73	3.16	1,428.97	6.66	2,107.70	9.82
8402	Converter plant.....	776.9	1,796.56	2.31	2,860.16	3.68	4,656.72	6.00
8414	Casting machines...	291.9	1,627.37	5.58	1,899.86	6.47	3,516.73	12.05
8502	Conveying system..	622.3	2,550.78	4.10	4,006.37	6.44	6,557.15	10.54
8821	Laboratory.....	96.5	448.42	4.65	575.13	5.96	1,023.55	10.61
8902.2	Sewer system.....	53.8	168.18	3.13	184.08	3.42	352.26	6.55
9013	Transfer table pit...	12.0	24.13	2.01	58.23	4.85	82.36	6.86
9017.01	Feed water heating plant.....	105.4	708.06	6.72	551.04	5.23	1,259.10	11.95
9017.21	Condensed water pump house.....	171.0	1,171.89	6.85	854.68	5.00	2,026.57	11.85
9060.21	Supply tanks, reverbs. and oil-fired boilers.....	189.5	875.70	4.62	1,685.76	8.90	2,561.46	13.52
Total.....		8,706.1	\$26,055.24	\$2.99	\$46,699.81	\$5.36	\$72,755.05	\$8.36

UNIT CONSTRUCTION COSTS

Reinforced Concrete

NUMBER	NAME OF ACCOUNT	Cu. Yd.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER Cu. Yd.	AMOUNT	PER Cu. Yd.	AMOUNT	PER Cu. Yd.
8307.10	Reverb. boiler bldg.	573.7	\$2,181.08	\$3.80	\$3,846.18	\$6.70	\$6,027.26	\$10.51
8612	Reverberatory flue.	487.8	1,657.09	3.40	3,886.80	7.97	5,543.89	11.37
9002.1	Power house.....	508.5	3,735.78	7.35	3,628.81	7.14	7,364.59	14.48
.2	North tunnel.....	180.3	1,350.79	7.49	1,230.57	6.83	2,581.36	14.32
9051	Cooling tower.....	706.3	5,212.31	7.38	5,640.13	7.99	10,852.44	15.37
9060.01	Oil supply sump and pump house.	340.6	2,338.72	6.87	3,230.46	9.48	5,569.18	16.35
.12	Oil storage tanks..	32.8	128.70	3.92	210.61	6.42	339.31	10.35
Total.....		2,830.0	\$16,604.47	\$5.87	\$21,673.56	\$7.66	\$38,278.03	\$13.53
Total concrete foundations.....		19,315.5	\$65,051.01	\$3.37	\$105,966.90	\$5.48	\$171,017.91	\$8.85

Plain Concrete Floors

NUMBER	NAME OF ACCOUNT	Sq. Ft.	THICK-NESS	LABOR COST		MATERIAL COST		TOTAL COST	
				AMOUNT	PER Sq. Ft.	AMOUNT	PER Sq. Ft.	AMOUNT	PER Sq. Ft.
7802.1	Sampling plant, ground floor.....	1,222	5 in.	\$105.68	\$0.09	\$263.92	\$0.21	\$369.60	\$0.30
8307.3	Around boilers.....	2,705	4 "	276.29	0.10	664.82	0.25	941.11	0.35
8812.41	Warehouse floor and platform....	8,298	4 "	558.04	0.07	721.60	0.09	1,279.64	0.15
8821.1	Laboratory.....	1,026	4 "	113.78	0.11	154.90	0.15	268.68	0.26
8842.1	Sample room.....	489	5 "	63.17	0.13	75.25	0.15	138.42	0.28
9002.4	Power plant basement.....	12,130	5 "	916.41	0.07	1,347.78	0.11	2,264.19	0.18
9017.22	Condensed water pump house.....	355	4 "	78.41	0.22	57.70	0.16	136.11	0.38
		26,225		\$2,111.78	\$0.08	\$3,285.97	\$0.13	\$5,397.75	\$0.21
The above concrete floors were laid in small blocks 5 to 6 ft. square, having a finished top and with sand joints between all squares									
8416.11	Loading platform..	6,803	5 in.	381.02	0.06	1,053.55	0.15	1,434.57	0.21
The floor above was finished as a large slab with no sand joints									

Reinforced Concrete Floors

NUMBER	NAME OF ACCOUNT	Sq. Ft.	THICK-NESS	LABOR COST		MATERIAL COST		TOTAL COST	
				AMOUNT	PER Sq. Ft.	AMOUNT	PER Sq. Ft.	AMOUNT	PER Sq. Ft.
7802.2	Sampling plant....	4,244	4½ in.	\$1,050.61	\$0.25	\$1,080.30	\$0.25	\$2,130.91	\$0.50
8307.2	Over slag track cut	7,676	5 "	1,018.17	0.13	1,687.29	0.22	2,705.46	0.35
8821.1	Laboratory.....	364	4½ "	59.81	0.16	114.87	0.32	174.68	0.48
9017.015	Feed water heating plant.....	1,330	4½ "	282.46	0.21	273.51	0.21	555.97	0.42
		13,614		\$2,411.05	\$0.18	\$3,155.97	\$0.23	\$5,567.02	\$0.41
The floors above were formed, reinforced and finished									
9003.41	Power plant main floor.....	10,210	3½ in.	1,267.91	0.12	3,341.61	0.33	4,609.52	0.45
The floor above was laid on Berger plate, reinforced and finished									
9051.02	Power plant cooling tower.....	17,116	4 in.	911.93	0.05	1,985.03	0.12	2,896.96	0.17
The floor above was reinforced, laid on the ground and straight-edged but with no finish									

Coping

NUMBER	NAME OF ACCOUNT	LIN. FT.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER FT.	AMOUNT	PER FT.	AMOUNT	PER FT.
8703.22	Boiler and blacksmith shop....	290	\$112.17	\$0.38	\$2.72	\$0.01	\$114.89	\$0.40
8717.22	Machine and carpenter shop...	320	121.67	0.37	23.70	0.08	145.37	0.45
8812.22	Warehouse.....	320	176.60	0.55	36.53	0.12	213.13	0.67
9017.243	Condensed water pump house..	57	14.50	0.25	2.73	0.04	17.23	0.30
		987	\$424.94	\$0.43	\$65.68	\$0.07	\$490.62	\$0.50
	This coping was 12 in. deep, projecting 2 in. from tile work							
9003.12	Power house.....	732	\$372.69	\$0.51	\$107.05	\$0.15	\$479.74	\$0.66
	This coping was 18 in. deep, projecting 2 in. from tile work							

Excavation—Type No. 1

This class covers shallow excavation made with picks, shovels, wheelbarrow and slips. The haul is less than 100 ft.

NUMBER	NAME OF ACCOUNT	TOTAL COST.	CU. YD.	COST PER CU. YD.
7308	Trestle approach to reverb. bldg.....	\$359.95	277	\$1.30
7313	Track scales on calcine track.....	108.95	118	0.92
7314	Trestles to receiving bins.....	548.18	589	0.93
7801	Sampling plant.....	295.06	332	0.89
8410	Air pipe line from power house.....	224.06	331	0.68
8416	Loading platform.....	212.77	216	0.99
8419.1	Bullion scales.....	19.76	24	0.82
8501	Conveying system.....	1,824.22	2,286	0.80
8611	Reverberatory flue.....	936.93	1,588	0.59
8621	Converter flue.....	168.02	198	0.85
8626	Roaster dust chamber flue.....	225.37	213	1.06
8812.4	Warehouse, floor.....	129.03	66	1.96
8820	Laboratory.....	191.27	212	0.90
8841	Sample room.....	61.35	72	0.85
8902.1	Sewer system.....	2,188.04	2,808	0.78
8906	Water pipe lines, tanks, etc.....	868.11	1,078	0.81
8908	Power conduit.....	358.53	435	0.82
8908	Power conduit branches.....	49.20	53	0.93
8909	Permanent air lines.....	267.50	401	0.67
8922	Temporary pipe lines.....	169.13	148	1.14
8923	Temporary warehouse.....	47.29	87	0.54
9009	Jet condenser not well.....	29.72	46	0.65
9014	North and south steam mains.....	249.65	279	0.89
9015.22	Exhaust pipe.....	20.82	29	0.72
9017.20	Condensed water pump house.....	229.49	236	0.97
9060.40	Oil piping.....	992.12	1,150	0.86
	Total.....	\$10,774.52	13,272	\$0.81

Excavation—Type No. 2

This class covers excavation made with picks, shovels, slips and carts. The haul is over 100 ft. in every case.

NUMBER	NAME OF ACCOUNT	TOTAL COST	CU. YD.	COST PER CU. YD.
7309	Track scales in receiving yard.....	\$348.91	388	\$0.90
7312	Retaining walls.....	77.66	60	1.29
7701	Crushing plant.....	689.67	609	1.13
8306	Flues from boilers to reverbs.....	19.86	15	1.32
8307.01	Waste heat boilers.....	213.44	129	1.65
8307.03	Feed pumps.....	601.44	659	0.91
8404	Converter stands.....	258.88	304	0.85
8413	Two casting machines.....	520.82	512	1.02
8421	Converter dust chamber.....	127.28	265	0.48
8426.1	Wet pan.....	2.44	3	0.81
9017	Feed water heating plant.....	241.09	274	0.88
9050	Cooling tower.....	1,674.32	1,589	1.05
9060	Oil supply sump and pump house.....	1,257.02	1,308	0.96
9060.11	Storage tanks.....	308.20	554	0.56
Total.....		\$6,341.03	6,669	\$0.95

Excavation—Type No. 3

This class covers excavation made with powder, picks, shovels and wheelbarrows. The haul was less than 100 ft.

NUMBER	NAME OF ACCOUNT	TOTAL COST	CU. YD.	COST PER CU. YD.
8906.2	Water supply tank.....	\$143.68	116	\$1.24
8916	Temporary power plant.....	354.07	388	0.91
8917	Temporary crushing plant.....	57.00	156	0.37
8920	Wagon road.....	924.74	951	0.97
9005	Well grading.....	2,075.75	2,600	0.80
Total.....		\$3,555.24	4,211	\$0.84

Excavation—Type No. 4

This class covers excavation made with powder, picks, shovels, slips, fresnos and carts. The haul was over 100 ft.

NUMBER	NAME OF ACCOUNT	TOTAL COST	CU. YD.	COST PER CU. YD.
7901	Bedding plant.....	\$12,259.43	12,319	\$0.99
8101	Roaster plant.....	1,547.07	1,216	1.27
8307	Reverb. boiler building.....	283.14	306	0.93
8312.10	Feed piping from heating plant to pumps.....	1,091.42	1,296	0.84
9060.20	Supply tank for reverbs. and boilers.....	405.31	404	1.00
Total.....		\$15,586.37	15,541	\$1.00

Excavation—Type No. 5

This class covers excavation done with plows, slips, fresnos, and in some cases powder. The haul was less than 100 ft.

NUMBER	NAME OF ACCOUNT	TOTAL COST	CU. YD.	COST PER CU. YD.
8701	Boiler and blacksmith shop.....	\$1,186.88	1,458	\$0.82
9001	Power house.....	7,796.65	7,313	1.07
9060.14	Railroad grading at oil sump.....	1,477.27	2,439	0.61
Total.....		\$10,460.80	11,210	\$0.93

Excavation—Type No. 6

This class covers excavation made with plows, slips, fresnos, and in some cases powder. The haul was over 100 ft.

NUMBER	NAME OF ACCOUNT	TOTAL COST	Cu. Yd.	COST PER
				Cu. Yd.
8121	Roaster dust chamber.....	\$1,112.83	1,194	\$0.93
8301	Reverberatory plant.....	1,477.66	1,890	0.78
8307.02	Oil-fired boilers.....	73.60	97	0.76
8401	Converter plant.....	5,731.11	6,330	0.91
8601	Chimney.....	867.05	597	0.61
8715	Machine shop.....	1,941.11	1,765	1.10
8810	Warehouse.....	996.08	1,287	0.77
Total.....		\$11,699.44	13,160	\$0.89

Excavation—Type No. 7

These are miscellaneous jobs where a variety of methods were used.

NUMBER	NAME OF ACCOUNT	TOTAL COST	Cu. Yd.	COST PER
				Cu. Yd.
7301	Yard tracks and industrial system.....	\$35,566.00	55,405	\$0.64
8901	Derrieks and construction equip.....	30.32	41	0.74
8905.01	Permanent outside lighting.....	17.87	21	0.85
8952	Employee quarters.....	401.63	318	1.26
8961	Steam heating installation.....	166.36	228	0.73
9016	Power house water pipe.....	1,485.34	2,406	0.62
9017.1	Feed water heating plant.....	167.19	266	0.69
Total.....		\$37,824.71	58,685	\$0.64

Excavation—Type No. 8

This class covers excavation made with picks, shovels, wheelbarrows and carts. A large portion of it was windlassed from deep pits.

NUMBER	NAME OF ACCOUNT	TOTAL COST	Cu. Yd.	COST PER
				Cu. Yd.
7401	Receiving bins.....	\$2,342.27	1,428	\$1.64

Excavation—Type No. 9

This covers backfilling and tamping in 4 to 5-in. layers.

NUMBER	NAME OF ACCOUNT	TOTAL COST	Cu. Yd.	COST PER
				Cu. Yd.
8301.01	Reverberatory plant.....	\$2,755.55	3,679	\$0.75
8307.04	Reverb. boiler building.....	548.10	972	0.56
8416.2	Loading platform.....	67.85	129	0.53
9050.01	Cooling tower.....	1,395.20	2,415	0.58
Total.....		\$4,766.70	7,195	\$0.66
Total excavation.....		\$103,351.08	131,371	\$0.79

Lighting

NUMBER	NAME OF ACCOUNT	No. DROPS	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER DROP	AMOUNT	PER DROP	AMOUNT	PER DROP
7407	Receiving bins.....	22	\$60.87	\$2.77	\$24.67	\$1.12	\$85.54	\$3.89
7707.1	Crushing plant.....	8	76.41	9.55	38.22	4.78	114.63	14.33
7806.1	Sampling plant.....	36	178.80	4.97	146.22	4.06	325.02	9.03
7906	Bedding plant.....	63	306.85	4.87	127.38	2.02	434.23	6.89
8109	Roaster plant.....	67	340.64	5.09	157.70	2.35	498.34	7.44
8313.1	Reverb. and boiler bldg.	104	612.30	5.89	473.19	4.55	1,085.49	10.44
8409.1	Converter plant.....	60	451.93	7.53	462.01	7.70	913.94	15.23
8506	Conveying system.....	33	189.86	5.76	84.56	2.56	274.42	8.32
8708	Boiler and blacksmith shop.....	17	23.41	1.38	44.50	2.62	67.91	4.00
8722	Machine shop.....	20	55.84	2.79	135.01	6.75	190.85	9.54
8812.50	Warehouse.....	26	45.09	1.73	70.48	2.71	115.57	4.45
8846	Sample room.....	7	22.97	3.28	52.83	7.54	75.80	10.82
9017.12	Feed water heating plant	6	53.35	8.89	25.83	4.31	79.18	13.20
9019	Power house.....	94	307.86	3.28	1,449.97	15.42	1,757.83	18.70
Total.....		563	\$2,726.18	\$4.84	\$3,292.57	\$5.85	\$6,018.75	\$10.69

Cost and Erection of Machinery

NUMBER	NAME OF ACCOUNT	CWT.	LABOR COST		ERECTION COST		TOTAL COST		
			AMOUNT	PER CWT.	AMOUNT	PER CWT.	AMOUNT	PER CWT.	
Group No. 1									
8313-8317	Two electrical feed pumps.....	433.45	\$375.70	\$0.87	\$375.70	\$0.87	\$6,187.56	\$14.28	
8316	Six No. 14 Wilgus oil systems.....	84.75	111.48	1.32	123.82	1.46	1,973.77	23.29	
8317.1	Two steam feed pumps.....	35.47	37.27	1.05	38.12	1.07	499.24	14.07	
9006.1	Two Nordberg blowers with air receivers.....	3,832.42	1,641.62	0.43	3,080.52	0.80	34,155.64	8.91	
9007.1	Three Curtis turbines and ten auto transformers.....	4,541.40	2,297.70	0.51	4,442.20	0.98	81,884.19	18.03	
9009.12	Two dry vacuum pumps for jet condenser.....	242.00	285.51	1.18	517.38	2.14	3,145.52	13.00	
9009.22	Two circulating pumps.....	375.60	366.90	0.98	433.78	1.15	3,902.58	10.39	
9010.02	Air compressor....	978.40	642.90	0.66	791.57	0.81			
9011.03	Three dry vacuum pumps.....	140.00	147.26	1.05	196.78	1.42	3,337.36	23.84	
9011.04	Three pumps and engines.....	972.55	389.32	0.40	568.75	0.58	9,118.69	9.38	
9060.02	Two 5 by 8 vertical triplex pumps...	113.54	176.20	1.55	195.07	1.72	2,211.78	19.48	
			11,749.58	\$6,471.86	\$0.55	\$10,763.69	\$0.92	\$146,416.33	\$13.59

Group No. 2

NUMBER	NAME OF ACCOUNT	CWT.	LABOR COST		ERECTION COST		TOTAL COST	
			AMOUNT	PER CWT.	AMOUNT	PER CWT.	AMOUNT	PER CWT.
8406	Two 40-ton Morgan cranes.....	2,215.00	\$1,438.50	\$0.65	\$3,813.34	\$1.72	\$23,027.65	\$10.40
8407	Two chinking machines.....	1,692.13	1,715.23	1.01	2,435.73	1.44	15,697.17	9.28
8415	Two casting machines.	2,692.20	3,266.34	1.21	3,682.40	1.37	27,477.55	10.21
		6,599.33	\$6,420.07	\$0.97	\$9,931.47	\$1.50	\$66,202.37	\$10.03

Group No. 3

7704	Farrell crusher, 36 by 18.....	500.00	392.86	0.79	401.31	0.80	1,486.47	2.96
8112	Two motor-driven fans at roaster building..	61.40	77.69	1.27	81.11	1.32	1,483.60	24.16
8718	Traveling hand crane, 5 ton.....	30.00	25.19	0.84	44.90	1.50	589.55	19.65
9004	" 20 ton.....	252.00	131.89	0.52	169.16	0.67	1,855.16	7.36
9008.1	Three surface condensers.....	1,157.00	415.31	0.36	542.82	0.47	19,978.86	17.27
9009.03	One barometric condenser.....	81.32	128.97	1.59	185.33	2.28	1,078.65	13.26
		2,081.72	\$1,171.91	\$0.56	\$1,424.63	\$0.68	\$26,472.29	\$12.72

Group No. 4

9011.02	Two exciters.....	543.00	491.01	0.90	864.31	1.59	6,609.27	12.17
9012.02	Two 150 Kw. synchronous generator sets.....	418.98	319.06	0.76	699.23	1.67	7,149.39	17.09
		961.98	\$810.07	\$0.84	\$1,563.54	\$1.63	\$13,758.66	\$14.30

In the above average costs an effort has been made for a logical grouping, yet it is somewhat arbitrary. Group 1 contains the erection of engine machinery. It was here necessary, in addition to handling heavy weights and placing on the foundation, to clean, adjust, and line up many mechanical parts. Group 2 is very similar to 1, but the machinery is not of the engine type and not so heavy in proportion to the labor required to put it in working order. Group 3 composes machinery that required little other labor in the main than the lifting of heavy loads into place. Group 4 is somewhat similar to Group 3, but the labor is principally electrical. The above costs are reported as labor, erection, and total costs. The labor cost is self explanatory. The erection cost is the labor cost plus the needed small supplies, such as waste, oil, small tools, and the like. The total cost is also self-explanatory.

Masonry

NUMBER	NAME OF ACCOUNT	CU. YD.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER CU. YD.	AMOUNT	PER CU. YD.	AMOUNT	PER CU. YD.
7312.2	Retaining wall.....	21.9	\$88.08	\$4.03	\$47.51	\$2.17	\$135.59	\$6.19

Cost of Painting Concrete

NUMBER	NAME OF ACCOUNT	Sq. Yd.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER Sq. Yd.	AMOUNT	PER Sq. Yd.	AMOUNT	PER Sq. Yd.
9002.45	Power house basement floor..	830	\$81.45	\$0.10	\$48.81	\$0.06	\$130.26	\$0.16
	Two coats of Toch cement filler							
9002.7	Power house walls and foundations.....	2,459	195.84	0.08	301.61	0.12	497.45	0.20
	One coat of Wadsworth Howland Bay State cement paint							
9003.43	Power house, top of main floor slab.....	1,134	95.56	0.08	199.32	0.18	294.88	0.26
	Two coats of Toch cement filler—one coat of Toch cement paint							

Painting Corrugated Iron

NUMBER	NAME OF ACCOUNT	Sq. Yd.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER Sq. Yd.	AMOUNT	PER Sq. Yd.	AMOUNT	PER Sq. Yd.
8812.30	Warehouse roof, underside... Two coats white lead and oil	813	\$81.16	\$0.10	\$65.66	\$0.08	\$146.82	\$0.18

Painting Berger Plate

9003.42	Power house main floor slab, underside.....	2,679	181.88	0.07	147.58	0.06	329.46	0.12
.54	Power house roof, underside... Two coats white lead and oil	6,813	692.84	0.10	324.55	0.05	1,017.39	0.15

Square yards derived by developing the Berger plate. This is three times the superficial area

Miscellaneous Painting

NUMBER	NAME OF ACCOUNT	Sq. Yd.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER Sq. Yd.	AMOUNT	PER Sq. Yd.	AMOUNT	PER Sq. Yd.
8703.60	Boiler shop.....	1,574	\$92.53	\$0.05	\$60.73	\$0.04	\$153.26	\$0.10
8718.60	Machine shop.....	1,989	118.00	0.06	87.40	0.04	205.40	0.10
8813.10	Warehouse.....	412	26.50	0.06	14.17	0.03	40.67	0.10
		3,975	\$237.03	\$0.06	\$162.30	\$0.04	\$399.33	\$0.10

This was two-coat work on wood

Painting Window Sash and Doors

NUMBER	NAME OF ACCOUNT	No. of Sash	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER SASH	AMOUNT	PER SASH	AMOUNT	PER SASH
7803.11	Sampling plant.....	129	\$118.94	\$0.92	\$28.96	\$0.22	\$147.90	\$1.14
8813.11	Warehouse.....	189	122.78	0.65	15.34	0.08	138.12	0.73
	Window sash and frames, two-coat work							
9003.60	Power house.....	299	290.09	0.97	16.72	0.05	306.81	1.02
	Window sash only, three-coat work							
	Doors figured as two sash							

Roofs

NUMBER	NAME OF ACCOUNT	SQUARES	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER Sq.	AMOUNT	PER Sq.	AMOUNT	PER Sq.
8703.30	Boiler and blacksmith shop.....	66.49	\$286.52	\$4.31	\$828.24	\$12.46	\$1,114.76	\$16.77
8717.30	Machine and carpenter shop.....	77.21	297.85	3.86	953.04	12.34	1,250.89	16.20
		143.70	\$584.37	\$4.06	\$1,781.28	\$12.40	\$2,365.65	\$16.46
	Consists of 2 by 8 tongue and groove sheathing, asbestos roofing and nailing strips. No painting included							
8843.4	Sample room.....	8.0	\$23.00	\$2.88	\$76.57	\$9.57	\$99.57	\$12.45
	Consists of 1 by 12 sheathing and asbestos roofing. No painting included							
9003.5	Power house.....	214.83	\$3,587.15	\$16.70	\$5,791.05	\$26.96	\$9,378.20	\$43.65
	Consists of Berger plate, concrete, tar, and P and B roofing. Painting the underside of the Berger plate is included							
9017.04	Feed water heating plant	8.80	\$115.25	\$13.10	109.88	\$12.48	\$225.13	\$25.58
9017.25	Condensed water pump house.....	5.00	73.92	14.78	60.83	12.17	134.75	26.95
		13.80	\$189.17	\$13.71	\$170.71	\$12.37	\$359.88	\$26.06
	Consists of 2 by 8 sheathing with asbestos roofing							
9060.07	Oil pump house.....	6.50	\$136.81	\$21.05	\$162.66	\$25.02	\$299.47	\$46.07
	Consists of "hyrib," concrete and plaster							
9060.131	Oil storage tanks.....	65.56	\$761.95	\$11.62	\$882.84	\$13.47	\$1,644.79	\$25.09
	Consists of wood supporting structure, 1-in. sheathing and metal lath and plaster							

Shafting, Pulleys and Belting

NUMBER	NAME OF ACCOUNT	LIN. FT. OF SHAFTING		LABOR COST		MATERIAL COST		TOTAL COST	
		AMOUNT	PER FT.	AMOUNT	PER FT.	AMOUNT	PER FT.	AMOUNT	PER FT.
7804	Sampling plant.....	85	\$64.01	\$0.75	\$1,871.07	\$22.01	\$1,935.08	\$22.76	
8107	Roasting plant.....	164	118.24	0.72	1,999.89	12.20	2,118.13	12.92	
8706	Boiler and blacksmith shop.....	51	105.59	2.07	301.16	5.91	406.75	7.98	
8720	Machine and carpenter shop.....	162	289.29	1.79	1,513.36	9.34	1,802.65	11.13	
8849.2	Sample room.....	26	23.39	0.89	170.17	6.55	193.56	7.44	

Structural Steel

NUMBER	NAME OF ACCOUNT	AMOUNT	TONS	COST	
				PER TON	
7308.2	Trestle approach to reverb. building.....	\$13,460.84	163.97	\$82.09	
7310	Bridge No. 1.....	377.40	3.70	102.00	
7314	Trestles to receiving bins.....	9,269.48	109.35	84.77	
7403	Receiving bins.....	29,276.63	353.09	82.92	
7703	Crushing plant.....	2,420.86	25.07	96.54	
7803	Sampling plant.....	10,408.12	110.85	93.89	
7903	Bedding plant.....	47,404.86	548.71	86.39	
8103	Roasting plant.....	37,252.67	445.28	83.66	
8123	Roaster dust chamber.....	34,745.41	415.68	83.59	
8303	Reverberatory plant.....	40,799.76	461.09	88.48	
8306.2	Flues from boilers to reverb. flue.....	2,815.32	34.78	80.95	
8308	Boiler building.....	25,839.85	292.03	88.48	
8403	Converter plant.....	69,359.60	783.86	88.48	
8423	Converter dust chamber.....	20,371.20	238.30	85.49	
8503	Conveying system.....	19,365.98	211.73	91.47	
8614	Reverberatory flue.....	3,593.06	41.61	86.35	
8624	Converter flue.....	6,616.44	81.99	80.70	
8629	Roaster dust chamber flue.....	9,209.36	94.46	97.49	
8703	Boiler and blacksmith shop.....	2,913.90	32.72	89.06	
8717	Machine and carpenter shop.....	3,431.42	38.23	89.76	
8812	Warehouse.....	3,734.08	39.76	93.92	
9003	Power house.....	23,773.13	254.29	93.49	
9014.02	North and south mains.....	7,694.58	86.81	88.64	
9017.02	Feed water heating plant.....	2,262.90	26.63	84.98	
Total		\$426,396.35	\$4,839.99	\$87.13	

Tile Walls

NUMBER	NAME OF ACCOUNT	CU. FT.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER CU. FT.	AMOUNT	PER CU. FT.	AMOUNT	PER CU. FT.
8106.02	Roaster flues.....	2,365	\$374.82	\$0.16	\$400.36	\$0.17	\$775.18	\$0.33
8123.10	Roaster dust chamber.....	14,980	2,268.29	0.15	2,585.45	0.17	4,853.74	0.32
8423.10	Converter dust chamber.....	6,369	1,620.05	0.25	2,182.40	0.34	3,802.45	0.60
8613	Reverberatory flue....	6,400	1,272.86	0.20	1,696.39	0.26	2,969.25	0.46
8628	Roaster dust chamber flue.....	4,231	1,018.13	0.24	1,573.06	0.37	2,591.19	0.61
8703.20	Boiler and blacksmith shop.....	2,397	477.95	0.21	612.62	0.27	1,090.57	0.48
8717.20	Machine and carpenter shop.....	2,397	531.45	0.22	571.28	0.24	1,102.73	0.46
8812.20	Warehouse.....	2,342	438.00	0.19	477.86	0.20	915.86	0.39
9003.10	Power house.....	14,343	3,856.83	0.26	4,510.20	0.31	8,367.03	0.58
9017.03	Feed water heating plant.....	706	285.99	0.41	234.36	0.33	520.35	0.74
9017.242	Condensed water pump house.....	257	98.87	0.38	84.28	0.33	183.15	0.71
Total.....		56,687	\$12,243.24	\$0.22	\$14,928.26	\$0.26	\$27,171.50	\$0.48

Unloading Brick and Tile

NUMBER	NAME OF ACCOUNT	TOTAL COST	TONS	COST PER TON
8105.01	Roaster brick.....	\$363.14	1,231.01	\$0.21
8123.11	Roaster dust chamber tile.....	307.72	525.05	0.58
8304.01	Reverberatory furnaces.....	1,264.49	2,279.49	0.55
8305.01	Cross and header flue.....	497.60	1,329.23	0.37
8309.02	Waste heat boilers.....	518.91	1,073.74	0.48
8309.12	Oil-fired boilers.....	183.94	228.11	0.81
8405.11	Converters.....	186.25	579.30	0.32
8423.11	Converter dust chamber.....	37.61	155.20	0.24
8613.01	Reverberatory flue.....	57.79	278.33	0.21
8628.01	Roaster dust chamber flue.....	57.60	171.40	0.34
8703.21	Boiler and blacksmith shop.....	18.89	69.70	0.27
8717.21	Machine and carpenter shop.....	44.06	58.80	0.75
8812.21	Warehouse.....	16.60	74.20	0.22
9003.11	Power plant.....	332.57	522.70	0.64
9017.031	Feed water heating plant.....	24.91	30.00	0.83
Total.....		\$3,911.98	8,606.26	\$0.45

These costs cover such costs as cleaning the site for unloading, building runways where needed, constructing brick sheds, and checking the quantities of the shipment, as well as the unloading of the various shapes in separate piles.

Ventilators

NUMBER	NAME OF ACCOUNT	NO. OF VENTI- LATORS	LABOR COST PER AMOUNT VENT	MATERIAL COST PER AMOUNT VENT	TOTAL COST PER AMOUNT VENT
8703.31	Boiler and blacksmith shop.....	3	\$16.01 \$5.33	\$261.50 \$87.17	\$277.51 \$92.50
8717.31	Machine and carpenter shop.....	3	11.16 3.72	248.24 82.74	259.40 86.46
		6	\$27.17 \$4.53	\$509.74 \$84.96	\$536.91 \$89.49
	48-in. Burt ventilators, square base, set on wooden roof				
8812.31	Warehouse.....	3	\$30.38 \$10.13	\$207.12 \$69.04	\$237.50 \$79.17
	48-in. Burt ventilators, round base, set on corrugated iron roof				
9003.3	Power house.....	6	\$125.60 \$20.94	\$439.76 \$73.29	\$565.36 \$94.23
	48-in. Burt ventilators, square base, set on concrete roof				
9017.045	Feed water heating plant.....	2	\$142.14 \$71.07	\$49.20 \$24.60	\$191.34 \$95.67
	42-in. ventilators made in new smelter shops, set on wooden roof				

Windows and Doors

NUMBER	NAME OF ACCOUNT	Sq. Ft. OPEN- ING	LABOR COST PER AMOUNT Sq. Ft.	MATERIAL COST PER AMOUNT Sq. Ft.	TOTAL COST PER AMOUNT Sq. Ft.
7703.1	Crushing plant.....	529	\$84.00 \$0.16	\$170.71 \$0.32	\$254.71 \$0.48
7803.1	Sampling plant.....	2,086	332.94 0.16	584.90 0.27	897.84 0.43
		2,615	\$416.94 \$0.16	\$735.61 \$0.26	\$1,152.55 \$0.44
	The above accounts cover wooden sash and frame set in steel and corrugated iron building				
8843.5	Sample room.....	298	\$32.39 \$0.11	\$118.85 \$0.40	\$151.24 \$0.51
	The above account covers wooden sash and frame set in wood and corrugated iron building				
8703.1	Boiler and blacksmith shop.....	2,581	\$693.02 \$0.27	\$2,456.28 \$0.95	\$3,149.30 \$1.22
8717.1	Machine and carpenter shop.....	3,037	923.61 0.30	2,992.16 0.99	3,915.77 1.29
		5,618	\$1,616.63 \$0.29	\$5,448.44 0.97	\$7,065.07 \$1.26

The above accounts cover steel sash and frame set in steel and tile curtain wall building

Windows and Doors (Continued)

NUMBER	NAME OF ACCOUNT	Sq. Ft. OPEN- ING	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER Sq. Ft.	AMOUNT	PER Sq. Ft.	AMOUNT	PER Sq. Ft.
8812.1	Warehouse.....	1,982	\$533.02	\$0.27	\$1,056.31	\$0.53	\$1,589.33	\$0.80
8822.5	Laboratory.....	823	208.70	0.25	480.33	0.59	689.03	0.84
9017.035	Heater house.....	186	59.83	0.32	99.98	0.54	159.81	0.86
9017.24	Cond. water pump house.	57	26.02	0.45	22.68	0.40	48.70	0.85
		3,048	\$827.57	\$0.27	\$1,659.30	\$0.54	\$2,486.87	\$0.81
	The above account covers wooden sash and frame set in steel and tile curtain wall building							
9060.06	Oil pump house.....	124	\$83.60	\$0.68	\$65.02	\$0.52	\$148.62	\$1.20
	The above account covers wooden sash and frame set in concrete walls							
9003.2	Power house.....	4,044	\$974.38	\$0.24	\$3,319.93	\$0.82	\$4,294.31	\$1.06
	The above account covers steel frame and wooden sash set in steel and tile curtain wall building							

Wire Baffles

NUMBER	NAME OF ACCOUNT	C. WIRES	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER C. WIRES	AMOUNT	PER C. WIRES	AMOUNT	PER C. WIRES
8123.01	Roaster dust chamber..	604.80	\$523.63	\$0.87	\$4,758.23	\$7.87	\$5,281.86	\$8.73
8423.01	Converter dust chamber	166.10	138.86	0.84	1,101.95	6.63	1,240.81	7.47
		770.90	\$662.49	\$0.86	\$5,860.18	\$7.66	\$6,522.67	\$8.46

Woodwork

NUMBER	NAME OF ACCOUNT	M.B.M.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER M.B.M.	AMOUNT	PER M.B.M.	AMOUNT	PER M.B.M.
7308.3	Trestle approach to reverb. bldg.....	27.65	\$703.93	\$25.45	\$768.92	\$27.81	\$1,472.85	\$53.27
7314.3	Trestle to receiving bins.....	27.21	572.23	21.03	838.11	30.80	1,410.34	51.83
		54.86	\$1,276.16	\$23.26	\$1,607.03	\$29.29	\$2,883.19	\$52.55
9052	Cooling tower.....	128.63	3,181.33	24.73	5,415.47	42.10	8,596.80	66.84
9060.16	Track bumpers.....	3.16	246.74	78.08	47.92	15.16	294.66	93.25

Wooden Floors

NUMBER	NAME OF ACCOUNT	Sq. Ft.	LABOR COST		MATERIAL COST		TOTAL COST	
			AMOUNT	PER Sq. Ft.	AMOUNT	PER Sq. Ft.	AMOUNT	PER Sq. Ft.
8717.40	Machine and carpenter shop.	4,136	\$269.80	\$0.07	\$593.30	\$0.14	\$863.10	\$0.21
	See sketch No. 24							

CHAPTER III

COMPOSITE COSTS

Cost of Building per Square Foot of Floor Space

NUMBER	NAME OF ACCOUNT	Sq. Ft. of FLOOR SPACE	TOTAL COST	
			AMOUNT	PER Sq. Ft.
7700	Crushing plant.....	1,650	\$5,968.32	\$3.62
	Includes accounts 7701 to 7703.2 inclusive and 7707.1			
7800	Sampling plant.....	6,140	16,299.16	2.65
	Includes accounts 7801 to 7803.11 inclusive, 7806.1 and 7809			
8100	Roasting plant.....	28,740	43,322.75	1.51
	Includes accounts 8101 to 8103.1 inclusive, and 8109			
8300	Reverberatory plant.....	20,370	50,687.28	2.49
	Includes accounts 8301 (\$784.32), 8302 (\$8,560.45), 8303 and one-half of 8313.1			
8307	Reverberatory boiler building.....	14,310	36,887.67	2.58
	Includes accounts 8307; 0.04; 0.1; 0.2; 0.3; 8308, one-half of 8313.1			
8400	Converter building.....	26,084	87,231.14	3.34
	Includes accounts 8401, 8402, 8403, 8409.1, 8413, 8414, 8416, 8416.1, 8416.11 and 8416.3			
8700	Boiler and blacksmith shop.....	4,424	11,320.58	2.56
	Includes accounts 8701 to 8703.40 inclusive, 8703.60 and 8708			
8714	Machine and carpenter shop.....	5,144	14,905.56	2.90
	Includes accounts 8714 to 8717.50 inclusive; 8717.60 and 8722			
8809	Warehouse.....	5,040	11,512.93	2.28
	Includes accounts 8810 to 8812.50 inclusive; 8813.10 and 8813.11			
8819	Laboratory.....	1,492	11,363.77	2.92
	Includes accounts 8820 to 8822.5 inclusive; 8825, 8828 and 8829			
8840	Sample room.....	600	991.46	1.65
	Includes accounts 8841 to 8843.5 inclusive; 8846 and 8848			
9000	Power plant.....	32,096	77,452.56	2.41
	Includes accounts 9001 to 9003.61 inclusive, excluding 9002.2			

Cost of Buildings per Cubic Foot

NUMBER	NAME OF ACCOUNT	CU. FT. IN BUILDING	TOTAL COST	
			AMOUNT	PER CU. FT.
7700	Crushing plant.....	27,040	\$5,968.32	\$0.22
7800	Sampling plant.....	80,547	16,299.16	0.20
8100	Roasting plant.....	410,140	43,322.75	0.11
8300	Reverberatory plant.....	474,350	50,687.28	0.11
8307	Reverberatory boiler building.....	500,850	36,887.67	0.07
8400	Converter building.....	1,529,636	87,231.14	0.06
8700	Boiler and blacksmith shop.....	86,268	11,320.58	0.15
8714	Machine and carpenter shop.....	100,308	14,905.56	0.15
8809	Warehouse.....	83,160	11,512.93	0.14
8819	Laboratory.....	16,140	4,363.77	0.27
8840	Sample room.....	6,000	991.46	0.16
9000	Power house.....	784,000	77,452.56	0.10

In the above costs the same account numbers are used as in computing the cost of buildings per square foot of floor space.

Cost of Buildings Equipped Per Square Foot of Floor Space

NUMBER	NAME OF ACCOUNT	Sq. Ft. OF FLOOR SPACE	TOTAL COST	
			AMOUNT	PER Sq. Ft.,
7700	Crushing plant.....	1,650	\$9,268.62	\$5.62
	Accounts 7701 to 7707.1 inclusive			
7800	Sampling plant.....	6,140	34,108.74	5.56
	Accounts 7801 to 7810 inclusive			
8100	Roasting plant.....	28,740	136,734.87	4.76
	Accounts 8101 to 8113.2 inclusive			
8300	Reverberatory plant.....	20,370	172,171.55	8.45
	Accounts 8301 to 8305.2 inc.—8307.2, 8312 and 8313, deducting one-half of each—8314 8315 and 8318—8316, deducting two-thirds of this account			
8307	Reverberatory boiler building.....	14,310	159,716.26	11.16
	Accounts 8306 to 8317.2 inc. deducting one-half of 8307.2, 8312 and 8313.1, and two-thirds of 8316			
8400	Converter building.....	26,084	216,033.37	8.28
	Accounts 8401 to 8419.4 and 8425 to 8426.4 inclusive			
8700	Boiler shop.....	4,424	21,449.23	4.85
	Accounts 8701 to 8708 inclusive			
8714	Machine and carpenter shop.....	5,144	27,356.27	5.32
	Accounts 8715 to 8722 inclusive			
8809	Warehouse.....	5,040	13,602.71	2.70
	Accounts 8810 to 8813.11 inclusive			
8819	Laboratory.....	1,492	6,144.02	4.13
	Accounts 8820 to 8831 inclusive			
8840	Sample room.....	600	2,826.11	4.71
	Accounts 8841 to 8849.2 inclusive			
9000	Power house.....	32,096	359,590.10	11.20
	Accounts 9001 to 9004 and 9006.01 to 9016.01 inclusive, deducting one-half of 9014 to 9014.05 inclusive.			

Cost of Buildings Equipped per Cubic Foot.

NUMBER	NAME OF ACCOUNT	Sq. Ft. OF FLOOR SPACE	TOTAL COST	
			AMOUNT	PER Sq. Ft.,
7700	Crushing plant.....	27,040	\$9,268.62	\$0.34
7800	Sampling plant.....	80,547	34,108.74	0.42
8100	Roasting plant.....	410,140	136,734.87	0.33
8300	Reverberatory plant.....	474,350	172,171.55	0.36
8307	Reverberatory boiler building.....	500,850	159,716.26	0.32
8400	Converter building.....	1,529,636	216,033.37	0.14
8700	Boiler and blacksmith shop.....	86,286	21,449.23	0.24
8714	Machine and carpenter shop.....	100,308	27,356.27	0.27
8809	Warehouse.....	83,160	13,602.71	0.16
8819	Laboratory.....	16,140	6,144.02	0.38
8840	Sample room.....	6,000	2,826.11	0.47
9000	Power house.....	784,000	359,590.10	0.46
	The amounts shown above are the same as those used in the cost of buildings equipped per square foot			

Bedding Plant and Bunker Bins. Cost per Cubic Foot of Capacity of both Beds and Bins

CAPACITY		TOTAL COST	
		AMOUNT	PER Cu. Ft.
228,440 cu. ft.....		\$150,939.05	\$0.66
	This cost consists of accounts 7901 to 7908 inclusive		
	The capacity was obtained by actual measurement		

Three Spreading Beds. Cost per Cubic Foot of Capacity

CAPACITY		TOTAL COST	
		AMOUNT	PER Cu. Ft.
160,380 cu. ft.....		\$120,177.94	\$0.75
	This is partly as estimated cost. It was obtained by deducting the cost of the bunker bins installation from the total bedding plant, and substituting therefor such excavation, founda- tion, steel work, etc., as would be necessary to provide for conveyors 7' and 10'		

Receiving Bins—Cost per Cubic Foot of Capacity

CAPACITY	TOTAL COST	
	AMOUNT	PER CU. FT.
11,386 cu. ft.....	\$38,073.73	\$3.34

This cost consists of accounts 7401 to 7404 inclusive, and 7407

The capacity of the receiving bins was obtained by assuming that the material lay on a one and one-half to one slope on falling from the cars

Conveyors—Cost per Ton of Capacity

CONVEYORS	CAPACITY, TONS PER HOUR	TOTAL COST		
		AMOUNT	PER TON	
No. 1.....	100	\$3,258.11	\$32.58	Account 7405
" 2.....	150	2,853.22	19.02	" 7405.01
" 7-7 ¹	250	6,938.01	27.75	" 7904
" 8-8 ¹ -8 ²	450	9,929.99	22.07	" 7904.1
" 9-9 ¹ -9 ² -10-10 ¹	500	11,668.39	23.33	" 7904.2
" 12.....	100	969.20	9.69	" 8113
" 13-13 ¹	200	3,948.62	19.74	" 8113.1
" 15.....	100	2,674.24	26.74	" 8425
" 3-4-5-6-11-14.....	700	13,716.49	19.59	" 8505

The above costs do not include any steel supporting structure for the conveyors. Capacities are taken from the designers' drawings

Complete Conveyor—Cost per Lineal Foot

CONVEYOR No.	TOTAL LENGTH	AMOUNT	COST PER FOOT
No. 3, 4, 5, 6, 11, 14.....	1,284.9 ft.	\$44,290.65	\$34.47

Cost consists of accounts 8501 to 8505.2, and 8506. The total length is the sum of the lengths of the various conveyors measured from center of head pulley to center of tail pulley

Cost of Cooling Tower per Thousand Gallons a Minute

M GALLONS PER MIN.	TOTAL COST	
	AMOUNT	PER M GALLONS
12 M.....	\$26,273.01	\$2,189.42

Cost consists of accounts 9050 to 9053 inclusive

Cost of Dust Chambers per Cubic Foot

NAME	CU. FT.	TOTAL COST		
		AMOUNT	PER CU. FT.	
Roaster dust chamber.....	256,860	\$49,664.76	\$0.19	Accounts 8121 to 8123.3 inclusive
Converter dust chamber.....	67,210	27,813.58	0.41	" 8421 to 8428 "

Cost of Flues per Cubic Foot

NAME	CU. FT.	TOTAL COST		
		AMOUNT	PER CU. FT.	
Reverberatory flue.....	63,420	\$13,453.70	\$0.21	Accounts 8611 to 8614.2 inclusive
Roaster dust chamber flue.....	29,527	12,859.10	0.44	" 8625 to 8629 "
Flues from boilers to reverb. flues.....	6,734	2,983.42	0.44	" 8306 to 8306.2 "
Converter flue.....	10,705	7,602.88	0.71	" 8621 to 8624 "

Cost of Flues per Lineal Foot

NAME	LIN. FT.	TOTAL COST		
		AMOUNT	PER FT.	
Reverb. flue.....	250	\$13,453.70	\$53.81	In obtaining these costs the same accounts were used as in computing the cost of flue per cubic foot.
Roaster dust chamber flue.....	170	12,859.10	75.64	
Flues from boilers to reverb. flues.....	246	2,983.42	12.13	
Converter flue.....	208	7,602.88	36.55	

Cost of Power Plant per Indicated Horse Power, Boiler Plant Included

I.H.P.	TOTAL COST	
	AMOUNT	PER I.H.P.
10,660.....	\$589,717.16	\$55.32
Cost consists of accounts 8306 to 8313.1 inclusive—deducting one-half of 8307.2, 8312 and 8313.1. 8316 to 8317.2 inclusive—deducting two-thirds of 8316. 9001 to 9004 inclusive. 9006.01 to 9053 inclusive. 9060.20 to 9060.23 inclusive—deducting three-fourths of these accounts.		
	I.H.P.*	
3 turbines.....	9,460	
2 Nordberg blowers.....	1,000	
1 air compressor.....	200	
Total.....	10,600	

Cost of Power Plant per Indicated Horse Power, Boiler Plant not Included

I.H.P.	TOTAL COST	
	AMOUNT	PER I.H.P.
10,660.....	\$398,631.17	\$37.40
Cost consists of accounts 9001 to 9004 inclusive. 9006.01 to 9016.02 inclusive—deducting one-half of 9014 to 9014.05. 9017.20 to 9017.27 inclusive. 9018.1 to 9022 inclusive. 9050 to 9053 inclusive		

Cost of Boiler Plant per Boiler Horse Power

BOILER HORSE POWER	TOTAL COST	
	AMOUNT	PER HORSE POWER
6,143.....	\$191,085.99	\$31.11
Cost consists of accounts 8306 to 8313.1 inclusive—deducting one-half of 8307.2, 8312 and 8313.1. 8316 to 8317.2 inclusive—deducting two-thirds of 8316. 9014 to 9014.05 inclusive—deducting one-half of these accounts. 9017 to 9017.15 inclusive. 9060.20 to 9060.23—deducting three-quarters of these accounts.		

Boiler Horse Power

7 waste heat @ 713.....	4,991
3 oil-fired @ 384.....	1,152

Total boiler horse power..... 6,143

Cost of Reverberatory Plant—per Ton of Capacity

TONS PER 24 HR.	TOTAL COST	
	AMOUNT	PER TON
1,200.....	\$172,171.55	\$143.47
Cost includes accounts 8301 to 8305.2 inclusive. 8307.2, 8312, 8313.1—(one-half of each). 8314, 8315, and 8318. 8316—(two-thirds). 3 reverberatories at 400 tons per 24 hr. capacity.		

Cost of Complete Roasting Installation per Roaster

NO. OF ROASTERS	TOTAL COST	
	AMOUNT	PER ROASTER
8.....	\$136,734.87	\$17,091.86
Cost consists of accounts 8101 to 8113.2—Roaster building and roasters		
8.....	\$199,258.73	\$24,907.34
Cost consists of accounts 8101 to 8113.2 inclusive—Roaster building and roasters. 8121 to 8123.3 inclusive—Roaster dust chamber. 8026 to 8029 inclusive—Roaster dust chamber flue.		

Tracks—Cost per Foot

LABOR COST		MATERIAL COST		TOTAL COST	
LENGTH OF TRACK	AMOUNT PER FOOT	AMOUNT PER FOOT	PER FOOT	AMOUNT PER FOOT	PER FOOT
14,116 ft.	\$38,190.18	\$2.71	\$27,335.46	\$1.94	\$65,525.64
This cost consists of accounts 7301 to 7305 inclusive. The 697 ft. of track which was on trestles was deducted from account 7303.					

CHAPTER IV

WAGE SCALE

OCCUPATION	FEB. 28, 1912, TEN HOURS		APRIL 1, 1912, NINE HOURS		JULY 24, 1912, NINE HOURS		JULY 1, 1913, NINE HOURS		SEPT. 1, 1913, EIGHT HOURS	
	A.	M.	A.	M.	A.	M.	A.	M.	A.	M.
Blacksmiths.....		\$2.50	\$4.00		\$4.50		\$4.50		\$4.25	
" helpers.....				2.25	3.00	3.00	2.50	2.50		
							3.00	3.00	2.50	2.50
							3.50		3.25	3.25
Boilermaker boss.....							5.50		5.50	
Boilermakers.....					4.50		4.50		4.25	
" " layer-out							4.75		4.75	
" " helpers..					3.00		3.00	3.00	3.00	3.00
Brick masons.....							16.50		16.50	
" tenders.....								12.25		12.25
Carpenter boss.....					5.00		5.00		5.00	
" 1st class.....	4.50		4.00		4.50		4.50		4.25	
" 2nd class.....	4.00	4.00								
" helpers.....	3.00	3.00	3.00	3.00	3.00		3.00		2.50	
" (with tools)....					3.50		3.50		3.00	
" helpers.....				2.25						
Cart drivers.....					2.25	2.25				
Cement finisher boss..									5.00	
" " ".....					4.50			2.50		
							3.50	3.50		
							4.00	4.00		
Concrete boss.....					4.50		4.50			
" mixers.....				2.25		2.25		2.25		
Corral boss.....			3.50		3.50		3.50		3.50	
" men.....					2.25	2.25				
Drillers.....		2.25		2.00						
Electrician boss.....							5.00		4.50	
									5.00	
Electrician.....							4.50		4.25	
" helpers.....							3.00	2.25	2.75	
									3.00	
Engineers, locomotive	3.25									
" compressors....					3.50		3.50		3.50	
" stationary.....					3.00					
Janitors.....				2.00		2.25				
Labor bosses.....			3.25	2.00	4.00		4.00		4.00	
			4.00	2.50			4.50			
				3.00						
Laborers.....		1.75		1.75		2.00		2.00		1.75
										2.00
Machine shop boss...							5.00		5.00	
" " ".....										
(outside).....							5.50			
Machinist.....			4.00		4.50		4.50		4.25	
" helpers.....			3.00		3.00	3.00	3.00	2.50	3.00	3.00
Miners, underground..				12.75		2.75				
" surface.....				2.25		2.25				
Office boys.....								2.00		2.00
										2.25
Pipe fitter boss.....							5.50		5.50	
Pipe fitter.....		2.50					4.50		4.25	
" " helpers.....							3.00	2.25	3.00	

A—American.

M—Mexican.

1 Eight hours.

2 Nine hours.

Wage Scale

OCCUPATION	FEB. 28, 1912, TEN HOURS	APRIL 1, 1912, NINE HOURS	JULY 24, 1912, NINE HOURS	JULY 1, 1913, NINE HOURS	SEPT. 1, 1913 EIGHT HOURS
Plow holders.....		2.25 2.25			
Plumbers.....			4.50		
" helpers.....			3.00		
Rigger boss.....		4.50	5.00		
Rigger.....				3.50	4.25
" helpers.....				4.50	
" helpers.....		2.25 2.25		2.25 2.25	2.25 2.25
" helpers.....				2.50 2.50	3.50 3.50
" helpers.....				3.00 3.00	
Stone mason boss....		3.50			
" ".....	3.00	3.00	3.00		
Steam fitters.....			4.50		
" helpers.....			3.00		
Teamsters, 4 and 6					
head.....		3.00	3.00		
" 2 head.....		2.00 2.00	2.25 2.25	3.00	3.00
" frescos and slips.	2.25 2.25	2.00 2.00	2.50 2.50		
" plow.....	2.75	2.25 2.25			
Tinners.....			4.00	4.50	4.25
" helpers.....			2.50 2.50		3.00
Tool room man.....		2.00	2.25		
Tool sharpeners.....		2.50	3.50	4.00	
Track boss.....			4.25		
Water boys.....	1.30	1.00	2.00 2.00	2.00 2.00	1.75 1.75
Warehouse help.....				3.00 2.50	3.00

CHAPTER V

RAW MATERIAL PRICES

Prices of Raw Material F. O. B. Clifton During New Smelter Construction

NAME	PRICE	UNIT
Asphalt (Trinidad).....	\$3.62	Cwt.
Asbestos sheets.....	5.68	"
Brick red standard, 2" x 4" x 8".....	16.986	M.
" silica A1 straights, 2½" x 6" x 12".....	98.10	"
" B 2 " 2½" x 4½" x 9".....	52.20	"
" C 2 " 3" x 6" x 20".....	184.19	"
" D 1 " 3" x 6" x 18".....	148.62	"
" Fire "Star", 2½" x 4½" x 9".....	49.68	"
" "Athens", 2½" x 4½" x 9".....	40.00	"
" Magnesite straights, 2½" x 4½" x 9".....	174.40	"
Special shapes at rate of \$189.40 per M brick 2½" x 4½" x 9", figuring from cubical contents, e.g. "A" special is 2.3796 larger than a 2½" x 4" x 9" brick, therefore would cost 2.3796 x \$189.40.		
Bolts, carriage ½" x 2".....	0.97	C.
" machine ½" x 2".....	1.01	"
" " ½" x 2".....	2.20	"
" " ½" x 24".....	13.04	"
Belted conveyor rubber, 7-5-1/8", 30" wide.....	3.63	Lin. ft.
" " 5-3-1/8", 20" ".....	1.9168	"
" " 6-4-1/8", 20" ".....	2.1898	"
Castings, rough iron.....	2.00 to 3.00	Cwt.
Cement, "El Toro" brand.....	0.7225	sack

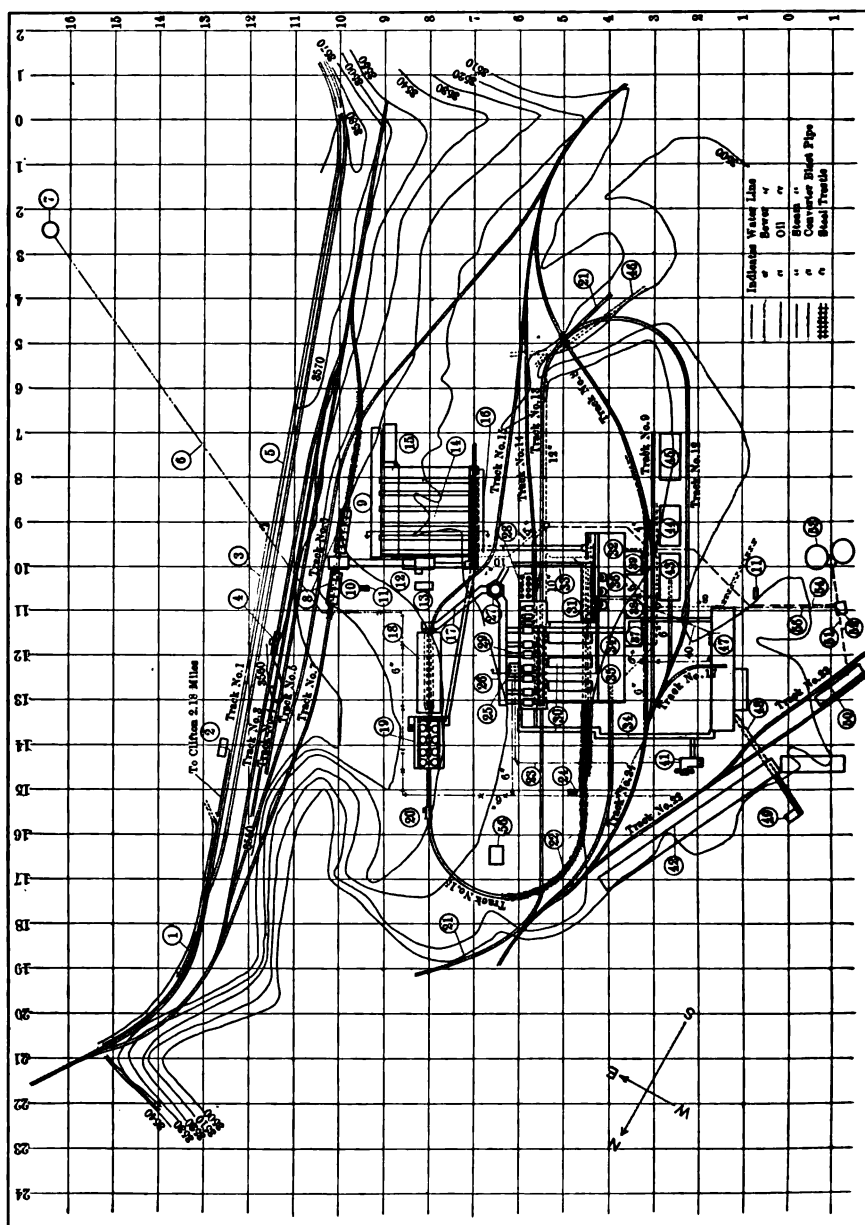
Prices of Raw Material F. O. B. Clifton During New Smelter Construction (Con.)

NAME	PRICE	UNIT
Clay, fire.....	\$7.00	ton
Coal, steam.....	8.00 to 7.175	"
Conduit $\frac{1}{2}$ " galvanized.....	6.39	C. ft.
Gasoline.....	0.23	gal.
Grease "Arctic" No. 4.....	6.50	cwt.
Glass, factory ribbed, $13\frac{1}{2}$ " $\times$ 20".....	13.28	C.
Hay, alfalfa.....	20.35	ton
Iron, corrugated No. 22, 10-ft. lengths.....	0.9729	sheet
" galvanized No. 20.....	3.49	cwt.
" round and square, base.....	2.25	"
" flat base.....	2.35	"
" sheets.....	2.35	"
Lumber, common R. O. P.....	26.00	M.
" No. 2 T & G flooring.....	31.76	"
Lead, white.....	8.80	cwt.
Lime, burnt.....	10.00	ton
Nails, common, base.....	2.84	cwt.
Oil, linseed.....	0.6163	gal.
" coal.....	0.12	"
Pipe, black 1 in.....	47.53	M. ft.
" " 2 in.....	91.84	"
" " 4 in.....	286.20	"
" " 6 in.....	516.10	"
" " 8 in.....	776.87	"
" sewer 6 in.....	27.94	C. ft.
" " 12 in.....	52.80	"
" " 24 in.....	193.00	"
Powder, black.....	7.1008	cwt.
" "Hercules," 30 per cent. $\frac{1}{4}$ " $\times$ 8".....	11.576 to 12.0754	"
Rails, 60, second-hand.....	27.50	ton
Rope, Manila, base.....	10.50	cwt.
Sand and gravel.....	1.00	ave. cu. yd.
Shafting 3- $\frac{1}{2}$ " C.R.....	3.78	cwt.
Silicate of soda.....	12.182	bbl.
Steel structure.....	3.11	cwt.
Ties, white oak.....	1.05	each
Tile, 4 $\times$ 8 $\times$ 12", "El Paso".....	58.742	M.
Valves, gate brass "Crane" 1 in.....	71.96	C.
" " " " 2 in.....	208.50	"
" " " " 3 in.....	6.20	each
" " 1B " 3 in.....	4.98	"
Wire, No. 12 weatherproof.....	23.18	cwt.
" No. 4 double braid stranded.....	56.50	M. ft.
" No. 12 galvanized.....	6.195	cwt.

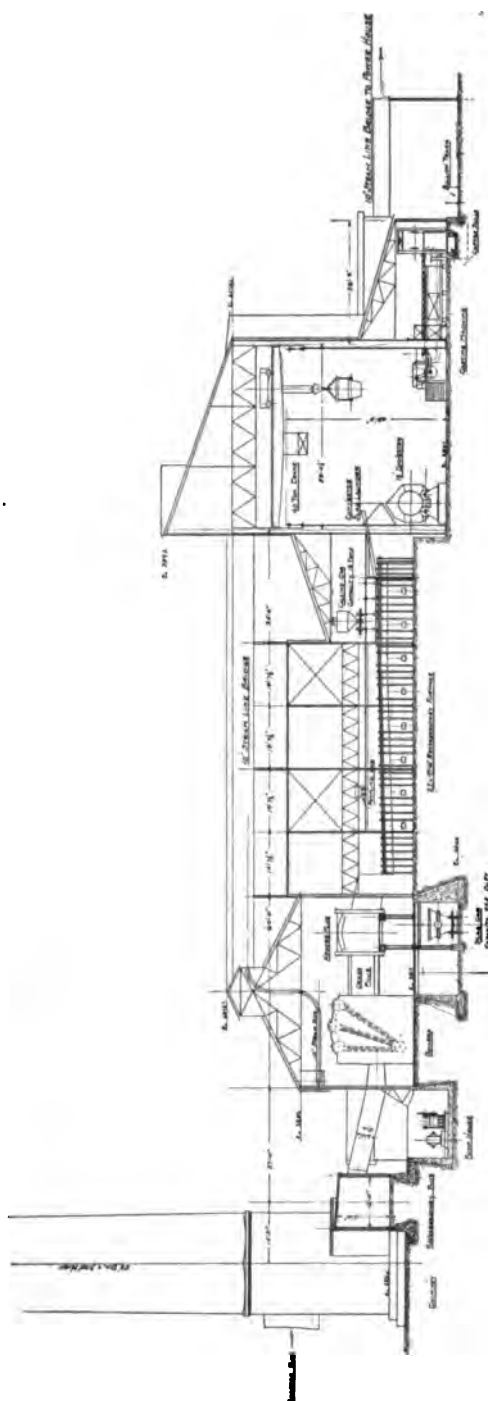
REFERENCE LIST FOR GENERAL PLAN OF SMELTER

(See Opposite Page)

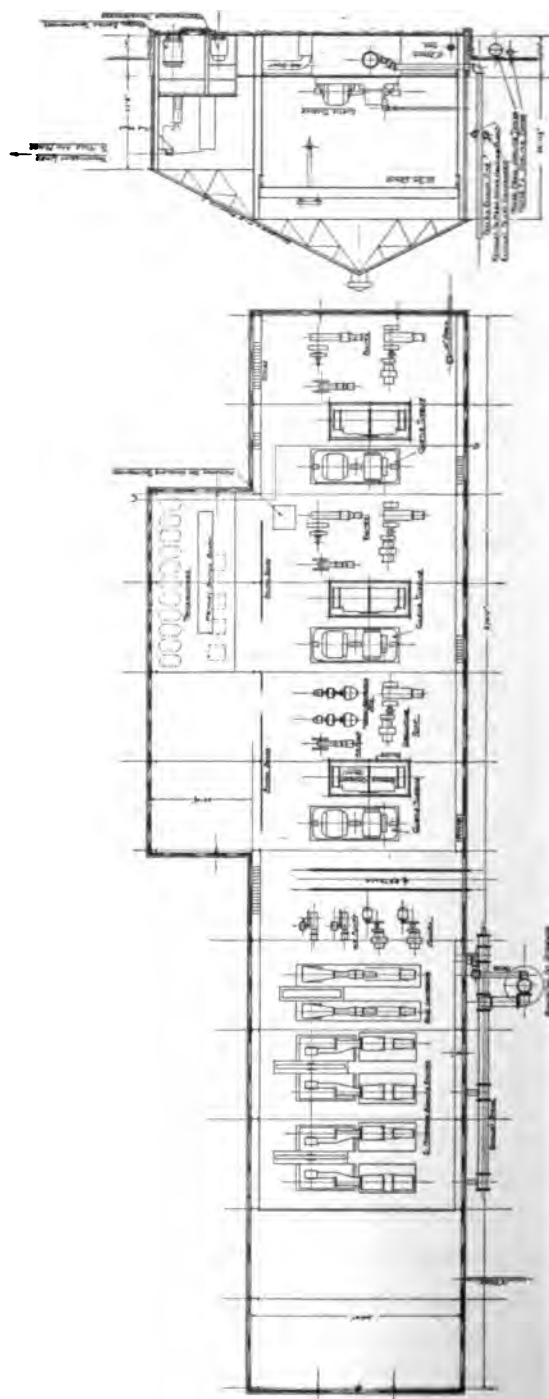
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|----------------------------------|--------------------------------|
| 1. Drainage Ditch | 29. Boiler Building |
| 2. Depot | 33. Converter Dust Chamber |
| 3. 6-in. Water Main from Clifton | 32. Silica Bins |
| 4. 150-ton Track Scales | 33. Converter Flue |
| 5. A. N. M. Main Line | 34. Steam Line |
| 6. 6-in. Water Pipe | 35. Converter Building |
| 7. 250,000-gal. Water Tank | 36. Converters |
| 8. Receiving Bins | 37. Casting Shed |
| 9. Transfer Pit | 38. Bullion Scales |
| 10. Crushing Plant | 39. Casting Shed |
| 11. Outside Closet | 40. 24-in. Blast Pipe |
| 12. Sampling Mill | 41. Heater House |
| 13. Sample Room | 42. Cooling Tower |
| 14. Three Spreading Beds | 43. Machine Shop |
| 15. Reclaimer House | 44. Boiler Shop |
| 16. Bunker Bins | 45. Warehouse |
| 17. Roaster Flue | 46. Culvert |
| 18. Roaster Dust Chamber | 47. Power House |
| 19. Roasters | 48. Power House Tunnel |
| 20. 40-ton Track Scale | 49. Condensed Water Pump House |
| 21. To Slag Dump | 50. Oil Sump |
| 22. Calcine Trestle | 51. 16-in. Oil Pipe |
| 23. 8-in. Hot-water Line | 52. Oil Pump House |
| 24. Outside Closet | 53. 500,000-gal. Oil Tanks |
| 25. Feed-water Pump House | 54. 8-in. Oil Pipe |
| 26. Reverberatory Flue | 55. 2 1/2-in. Steam Pipe |
| 27. Chimney | 56. Assay Office |
| 28. 163-barrel Oil Tanks | |



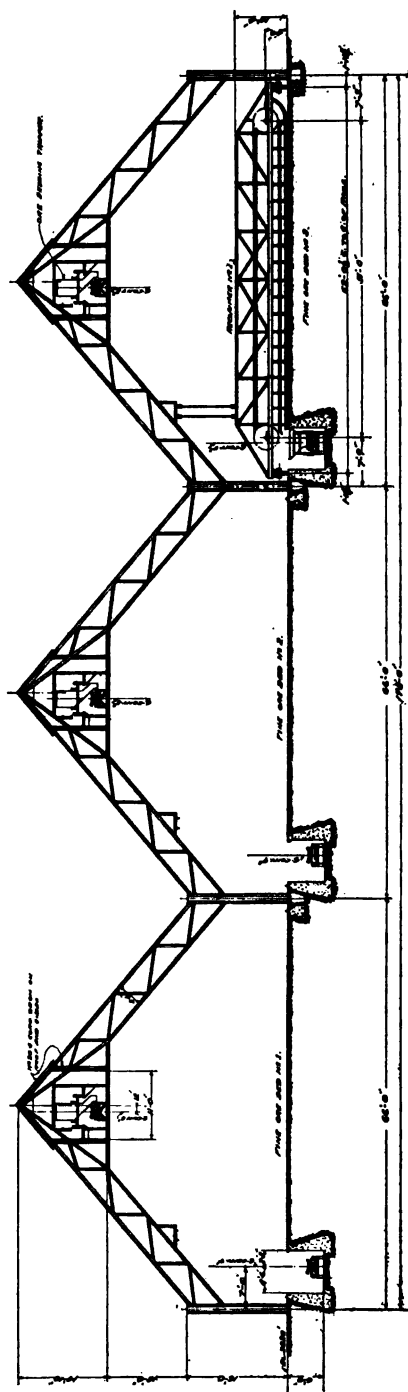
GENERAL PLAN OF SMELTER.
Numbers in circles refer to list on opposite page.

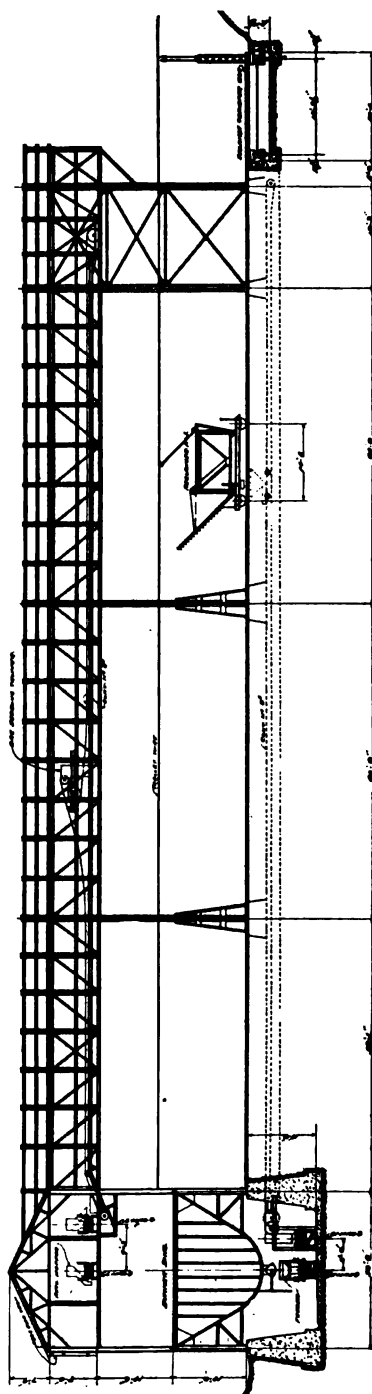


CROSS-SECTION THROUGH REVERBERATORIES AND CONVERTERS.

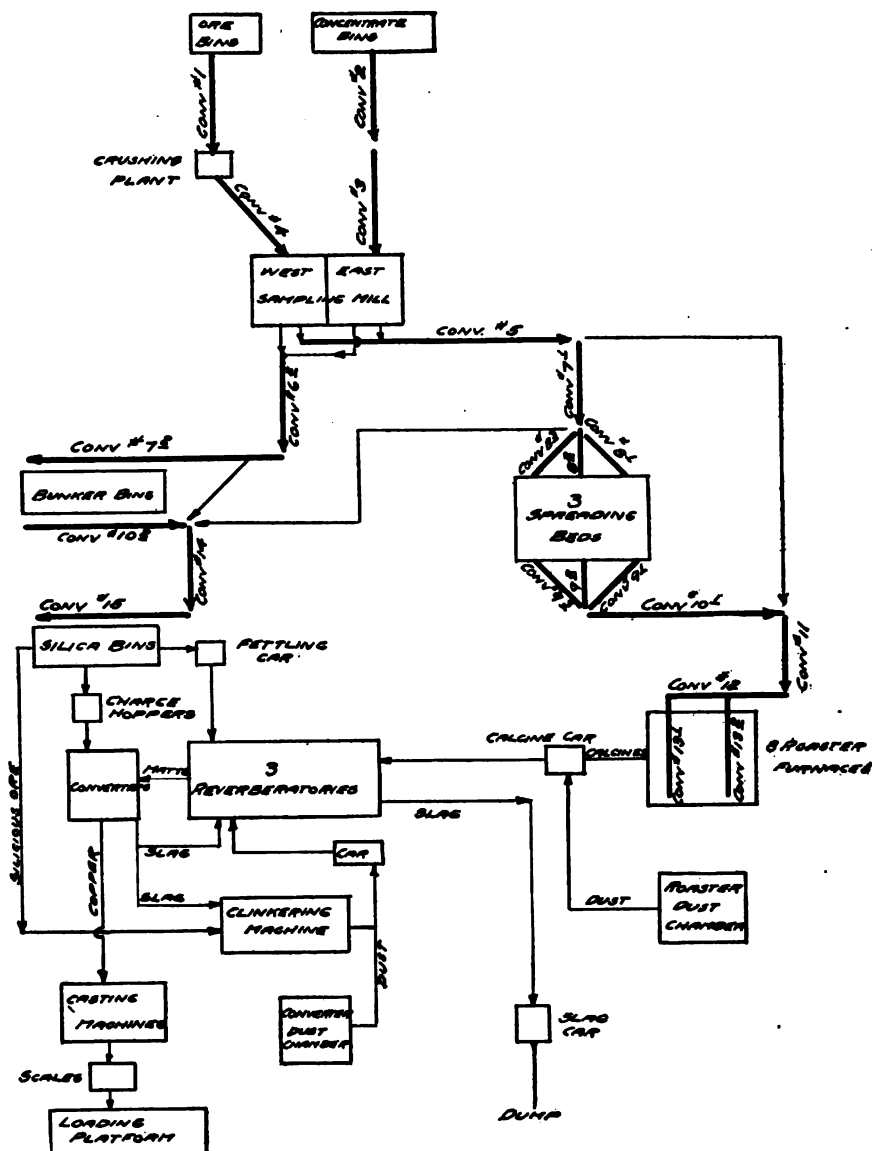


PLAN AND CROSS-SECTION OF POWER HOUSE.

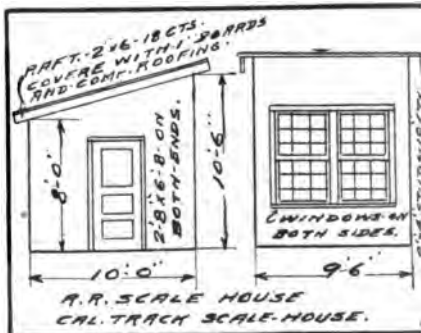
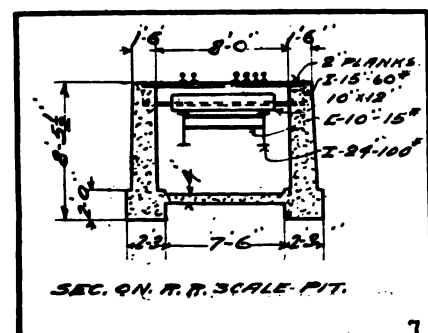
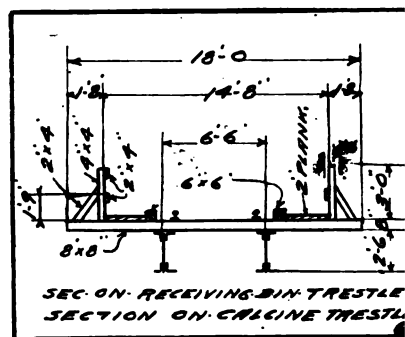
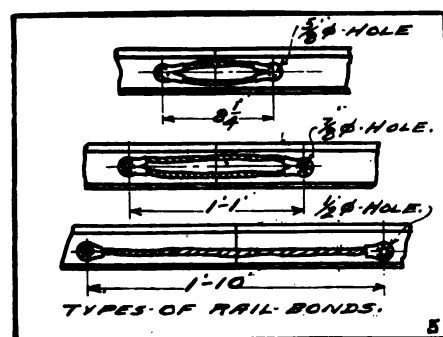
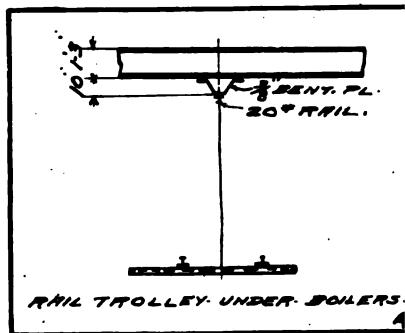
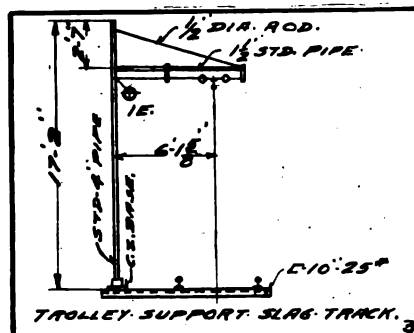
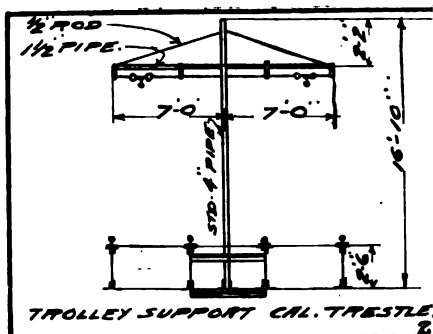
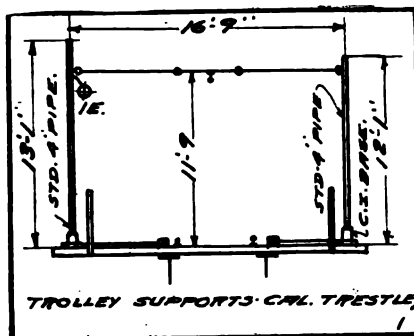


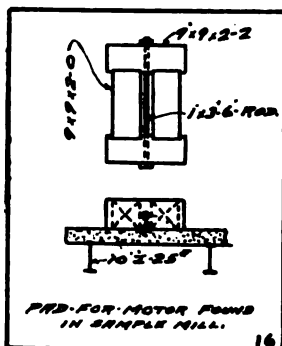
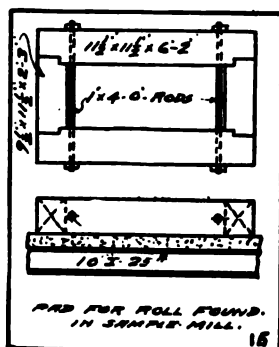
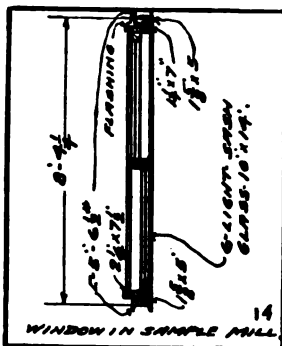
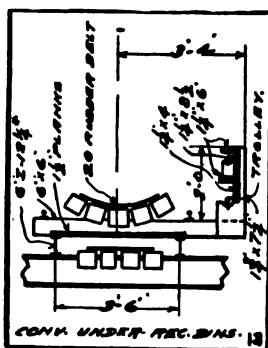
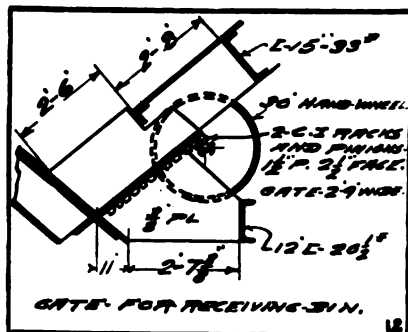
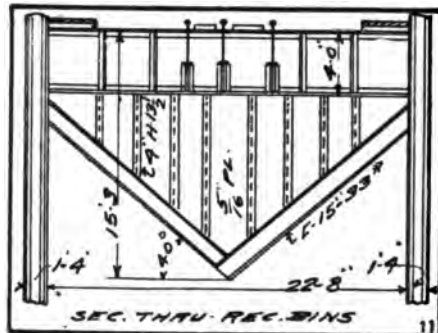
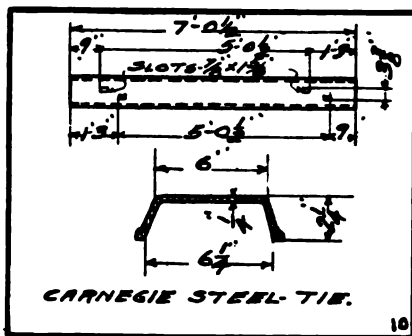
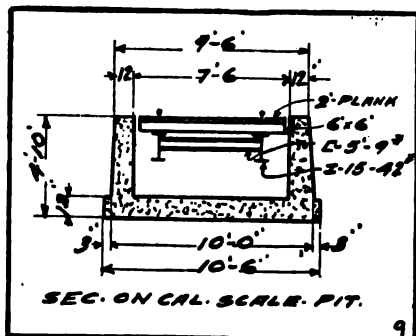


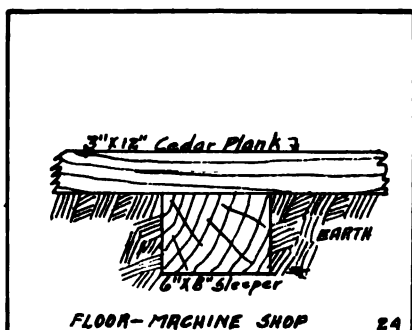
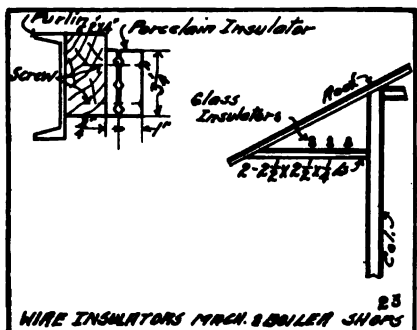
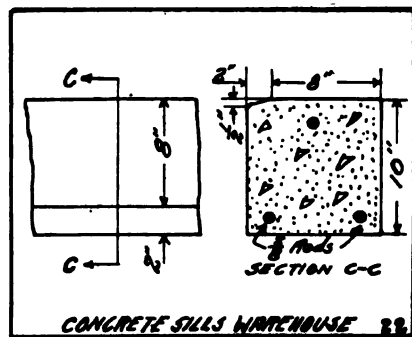
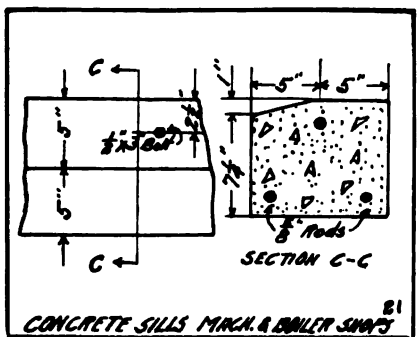
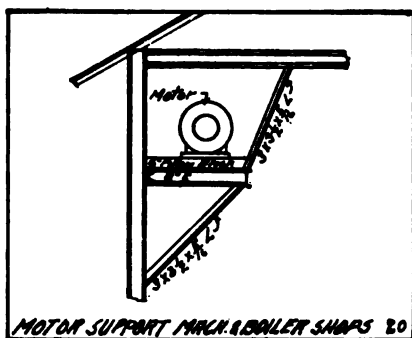
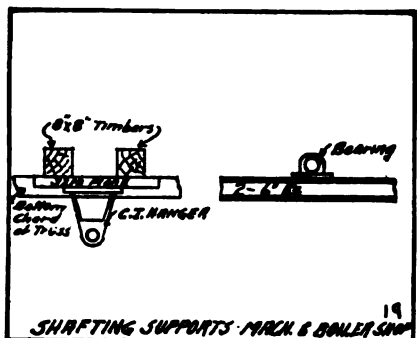
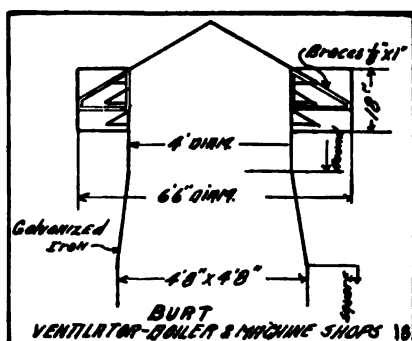
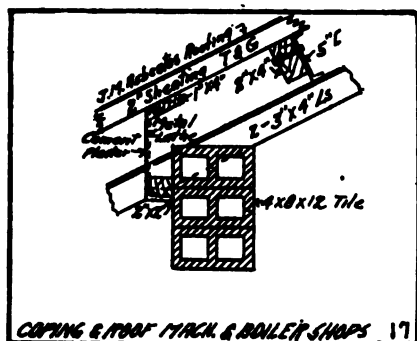
LONGITUDINAL SECTION THROUGH BEDDING PLANT, LOOKING NORTH.

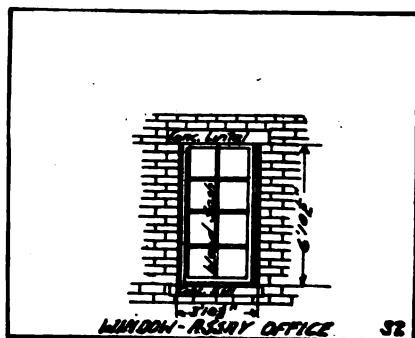
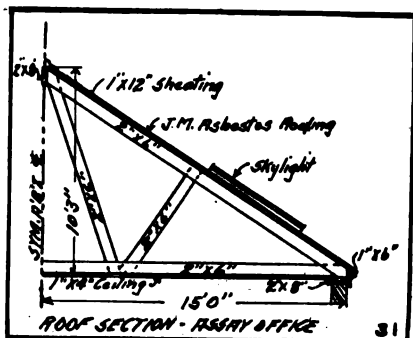
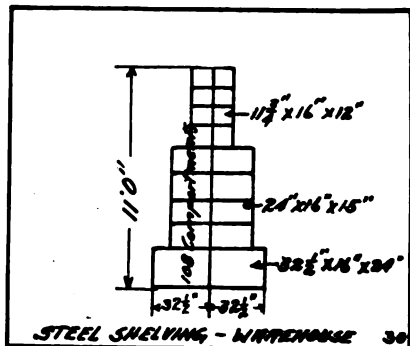
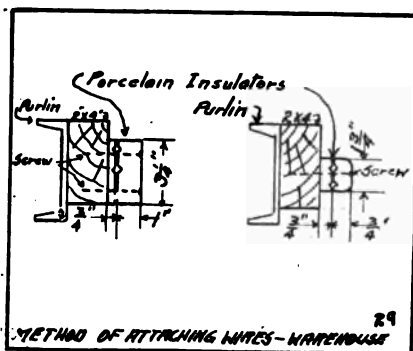
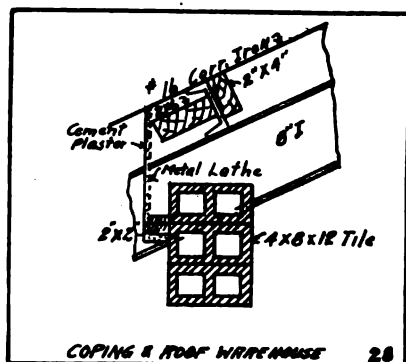
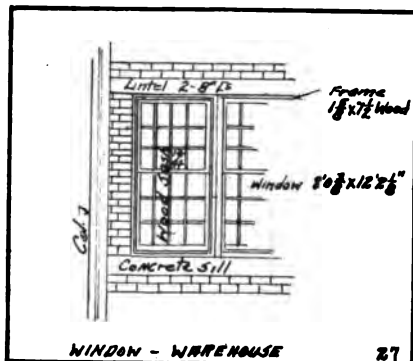
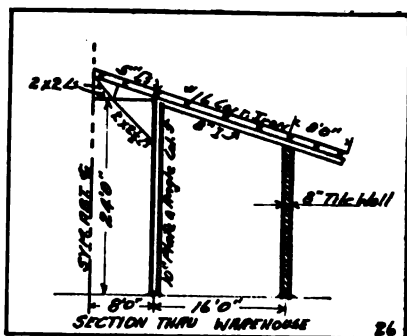
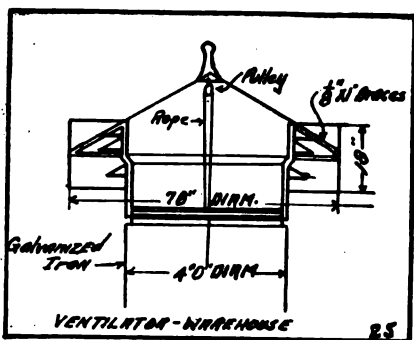


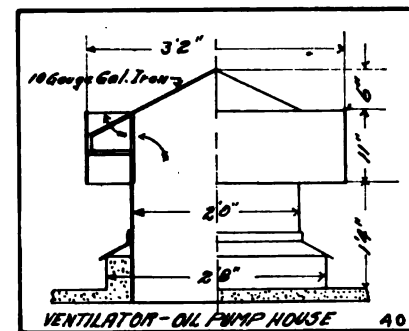
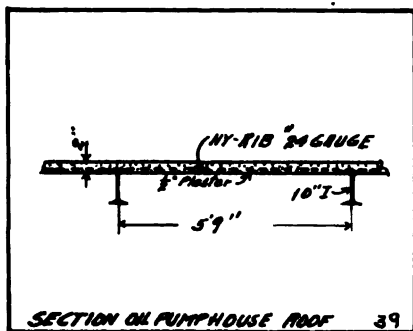
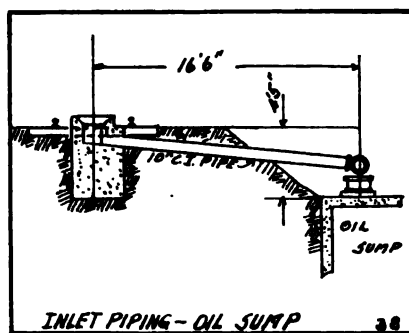
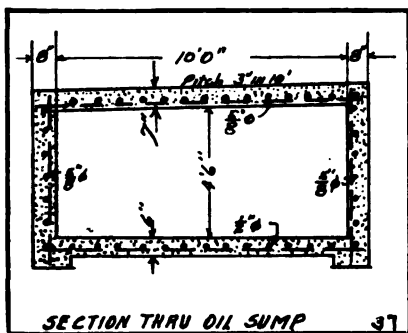
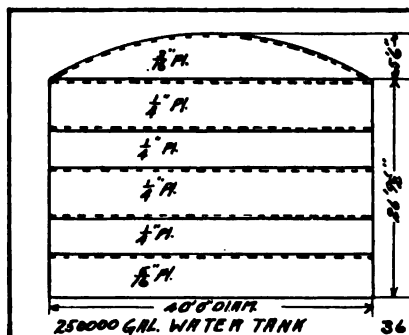
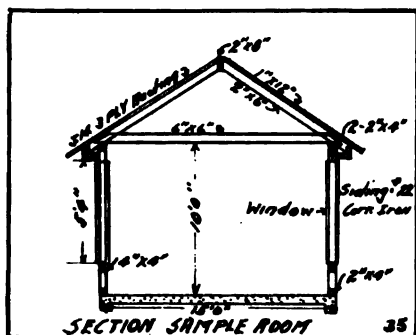
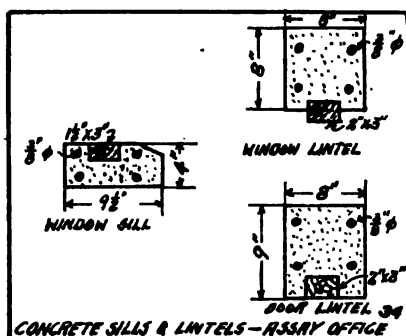
FLOW SHEET OF SMELTER. HEAVY LINES REPRESENT CONVEYORS.

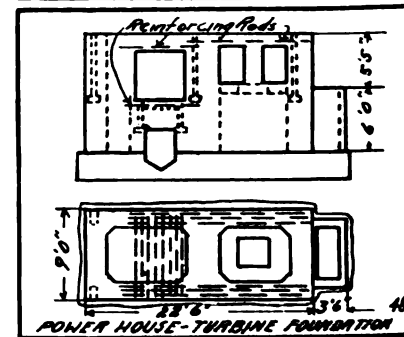
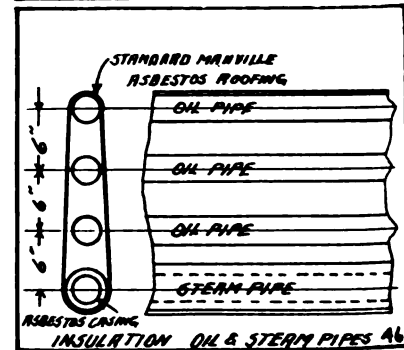
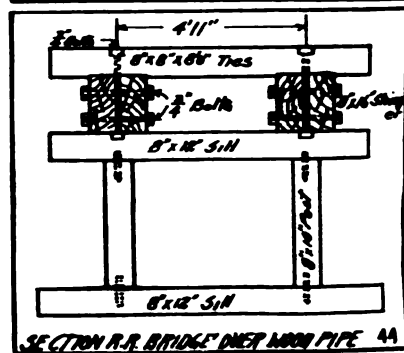
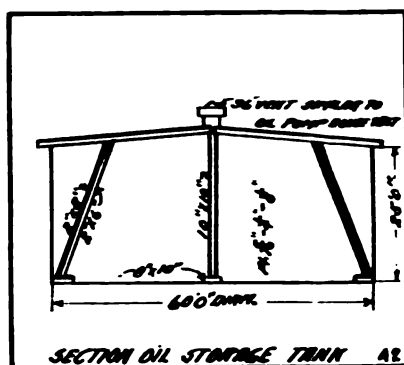
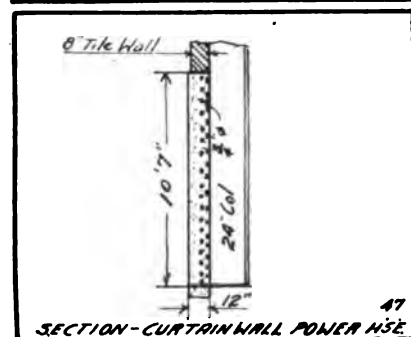
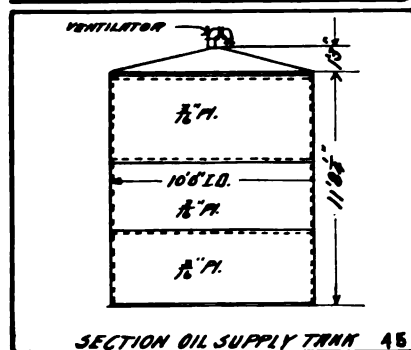
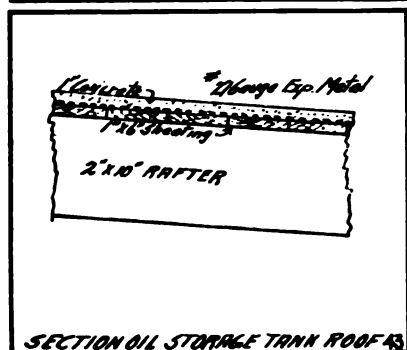
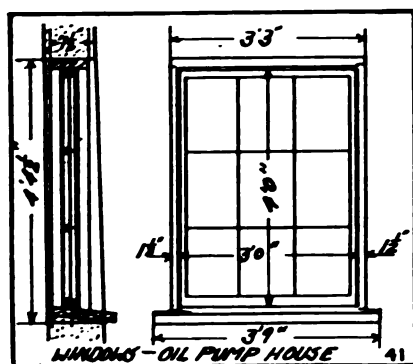


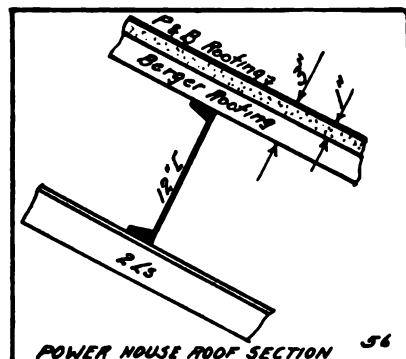
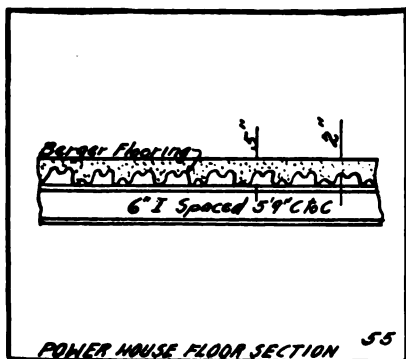
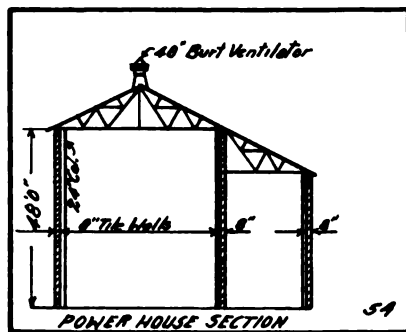
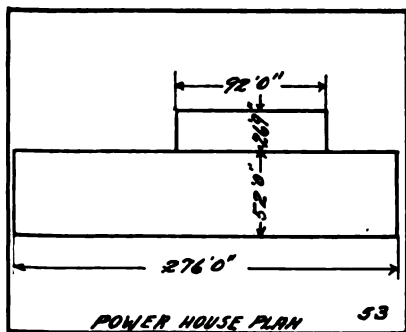
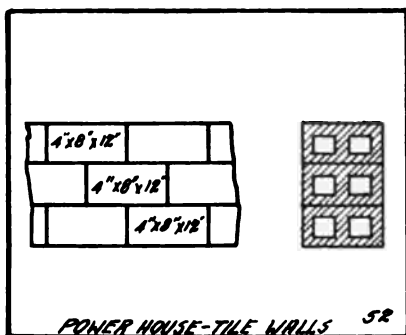
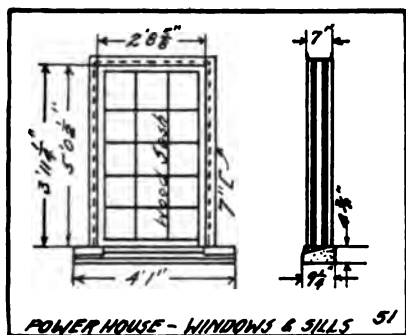
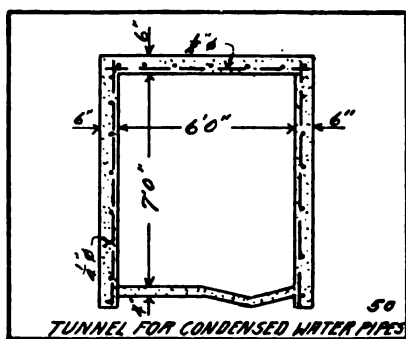
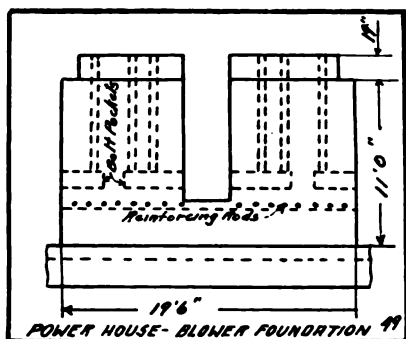


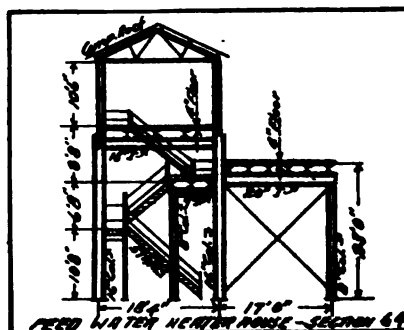
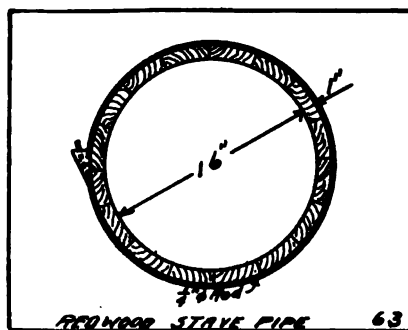
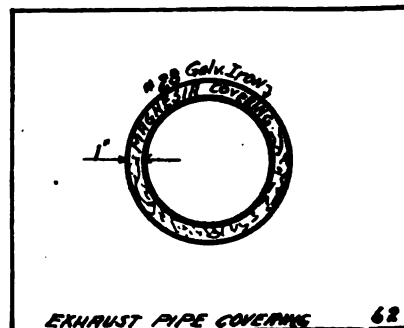
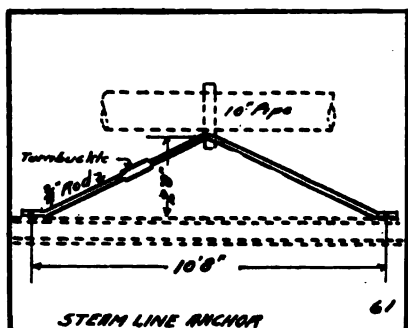
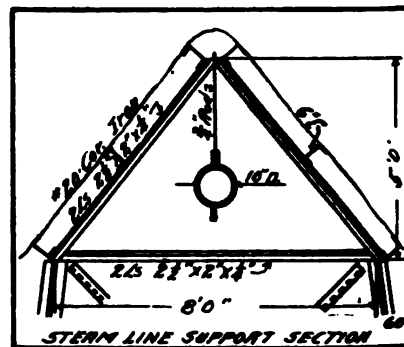
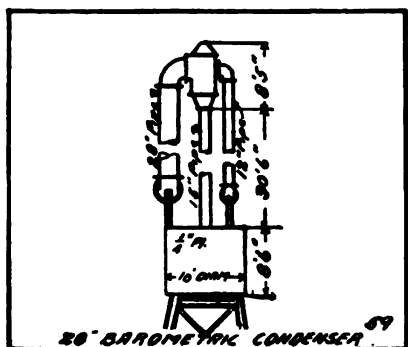
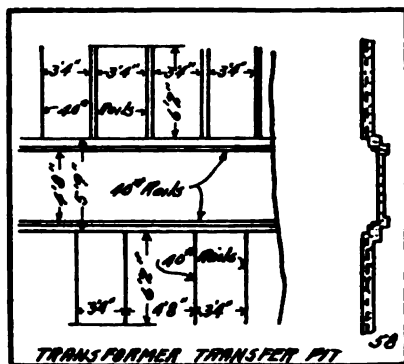
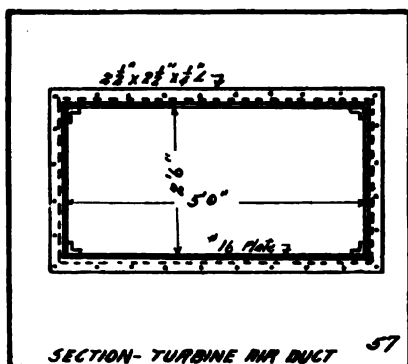


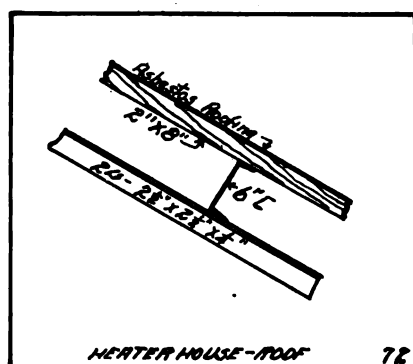
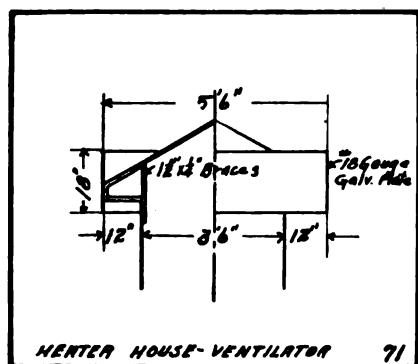
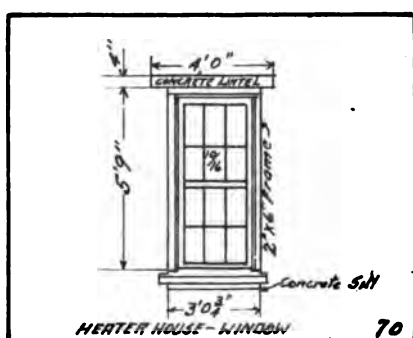
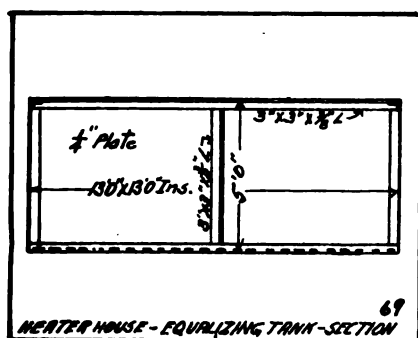
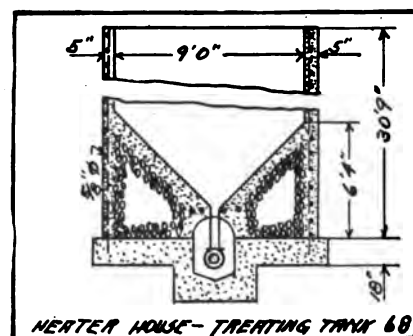
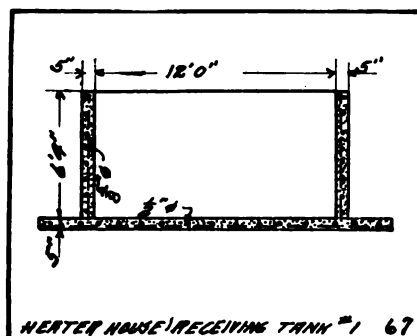
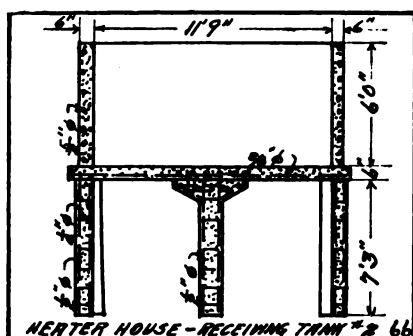
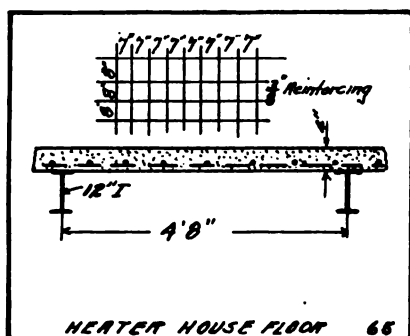


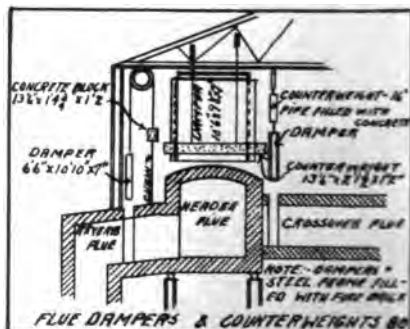
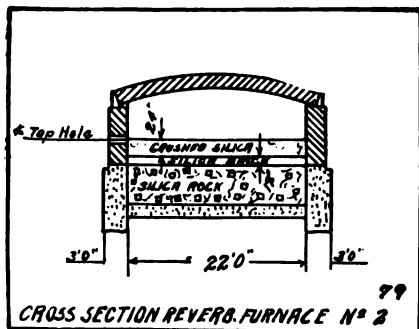
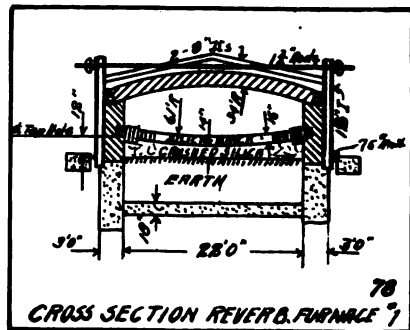
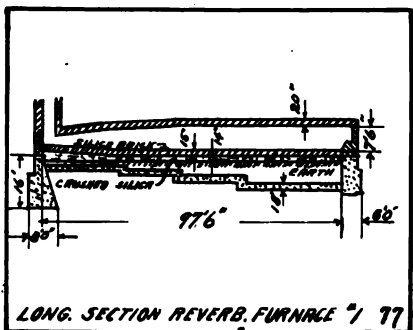
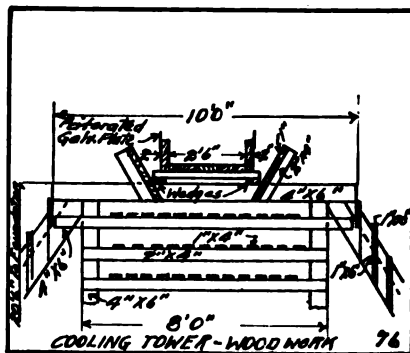
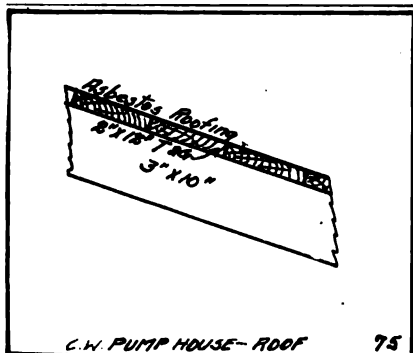
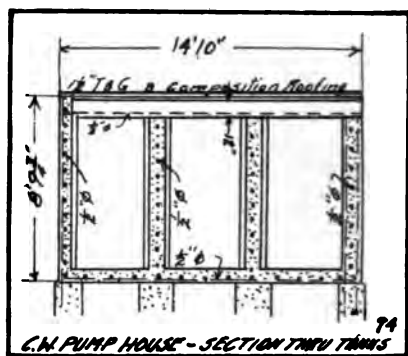
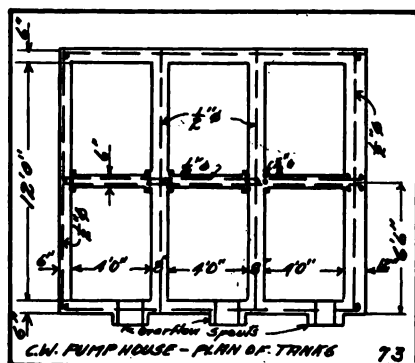


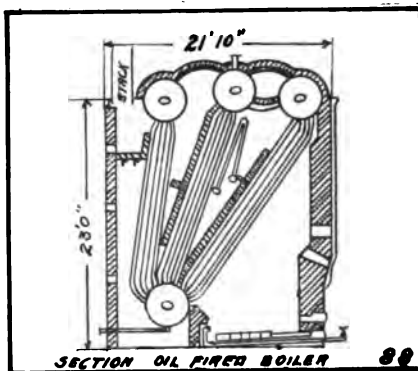
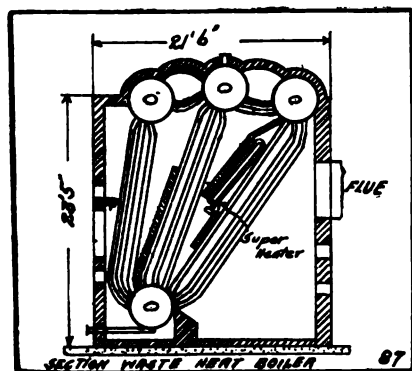
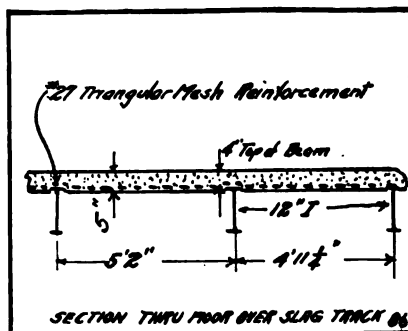
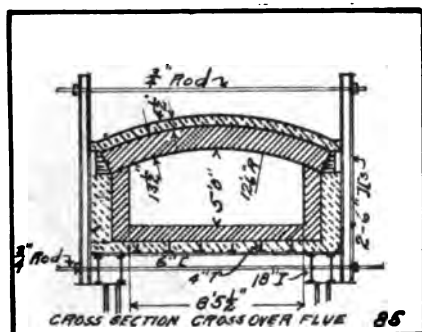
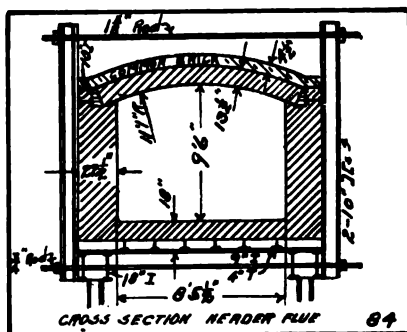
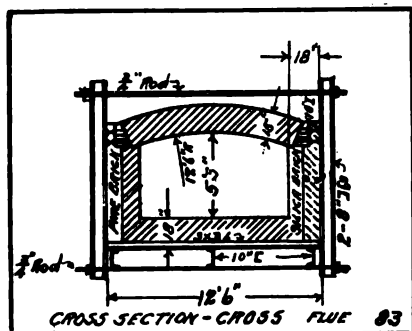
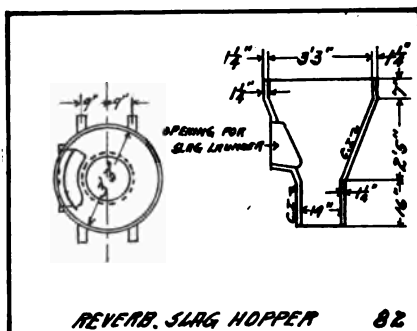
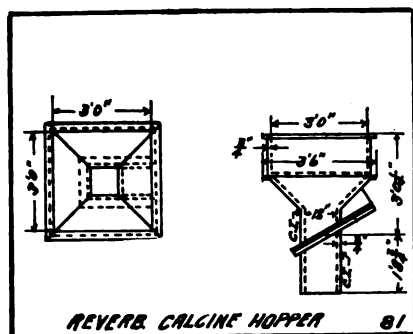


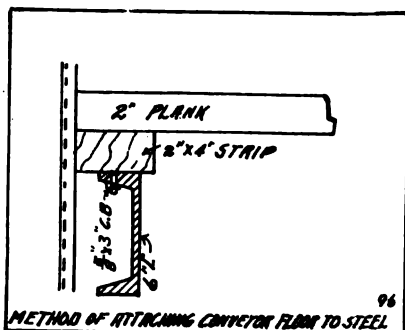
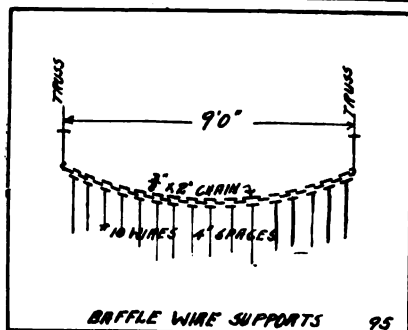
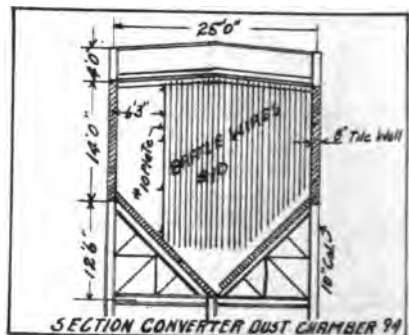
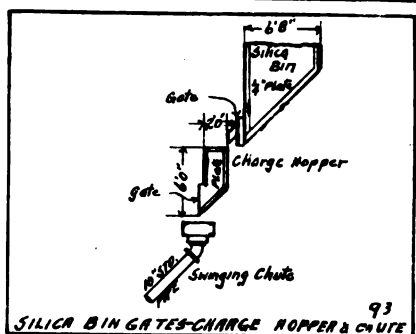
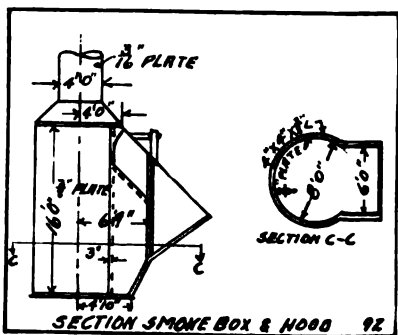
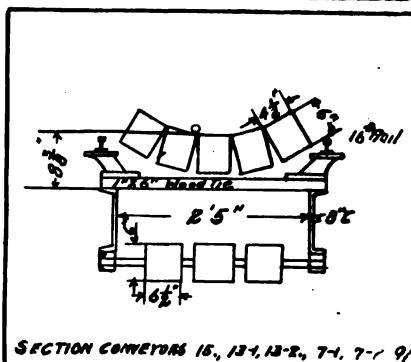
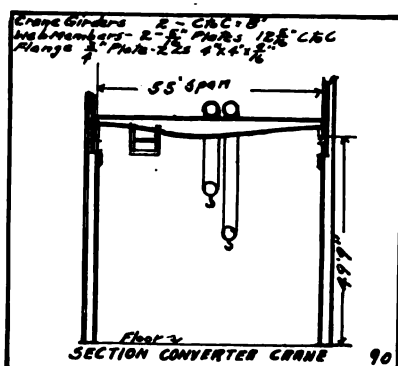
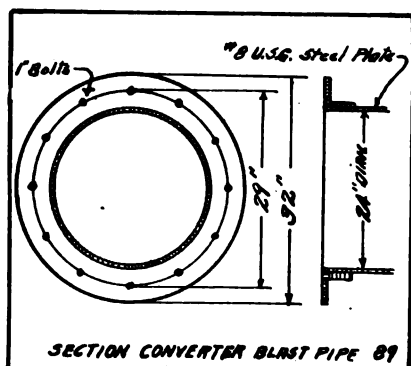


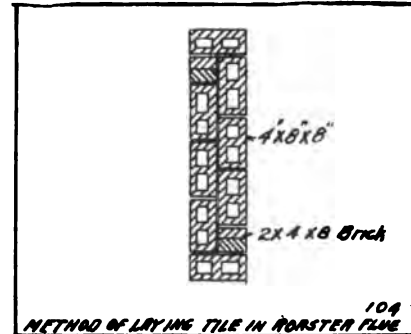
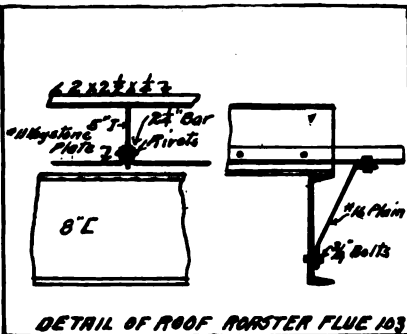
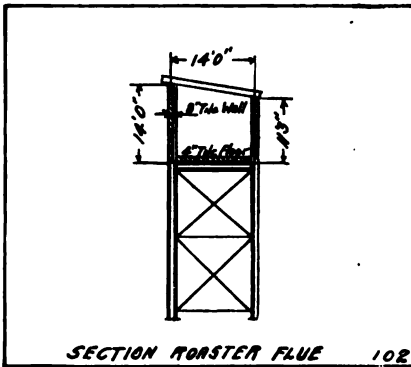
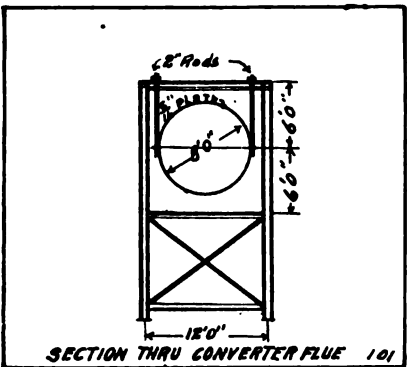
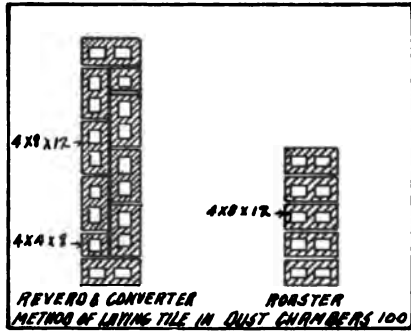
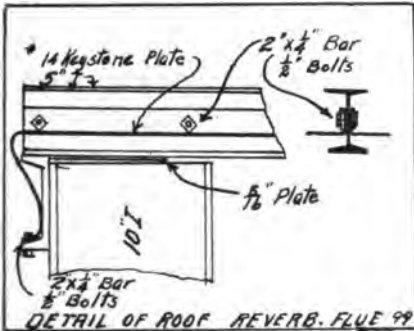
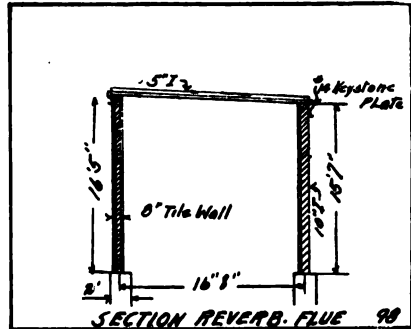
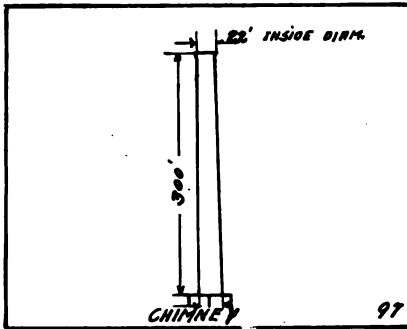


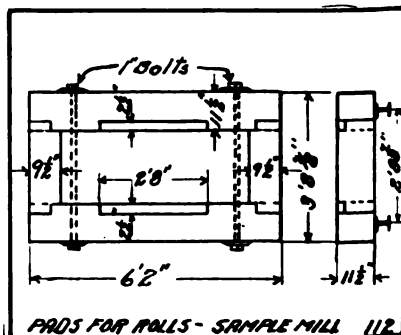
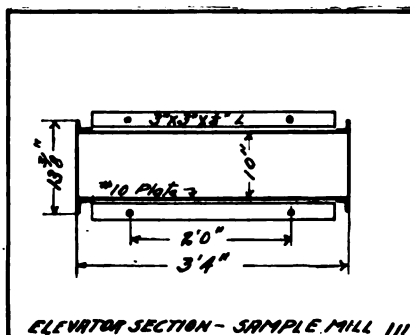
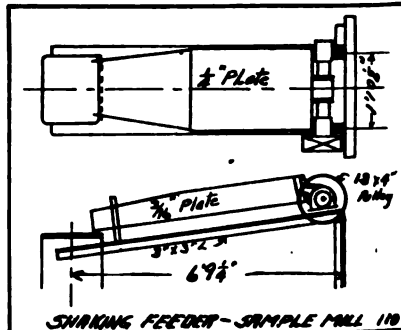
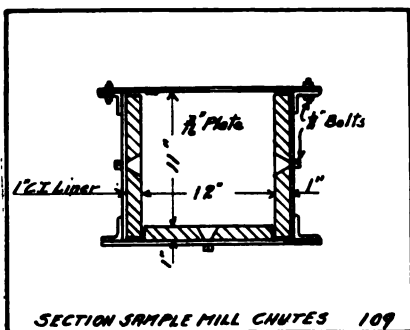
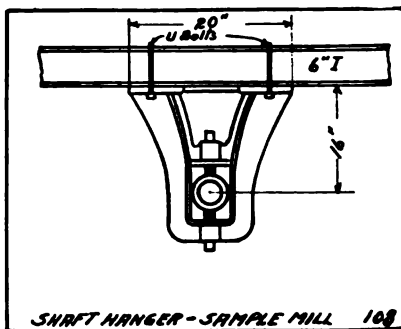
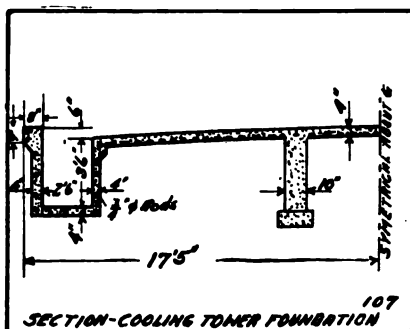
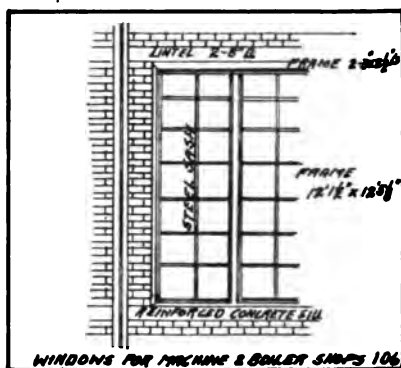
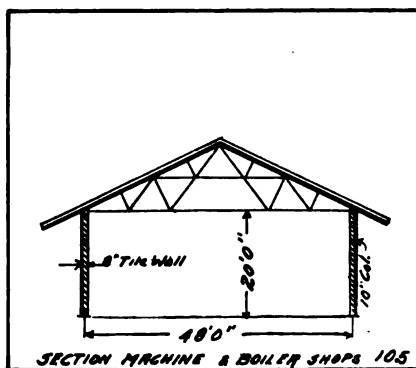


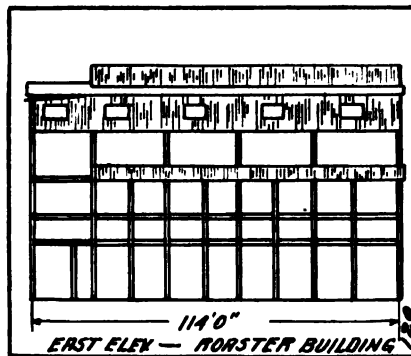
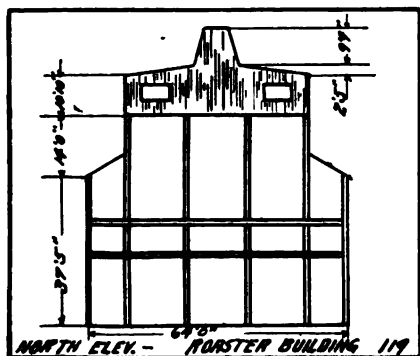
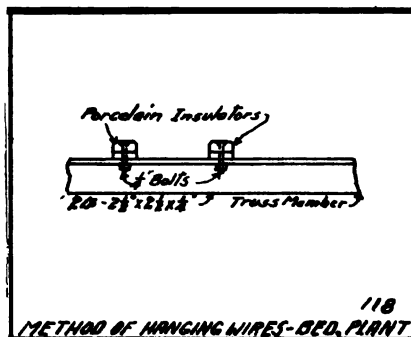
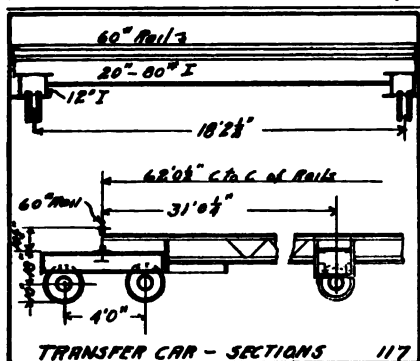
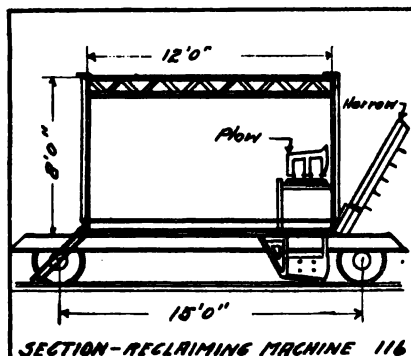
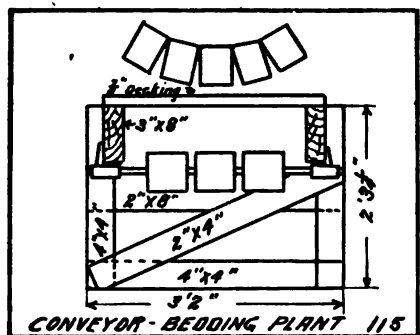
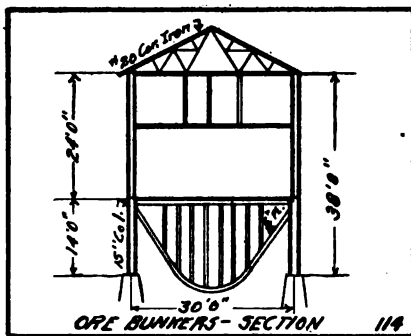
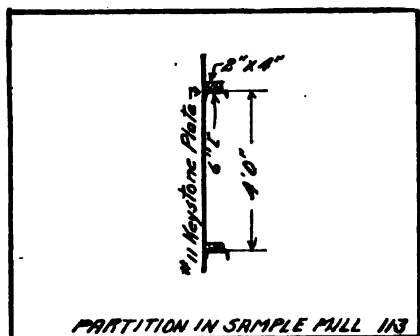


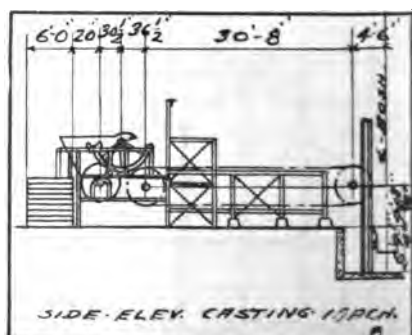
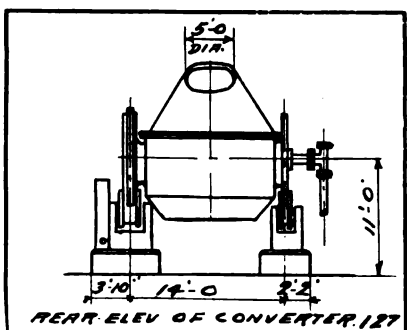
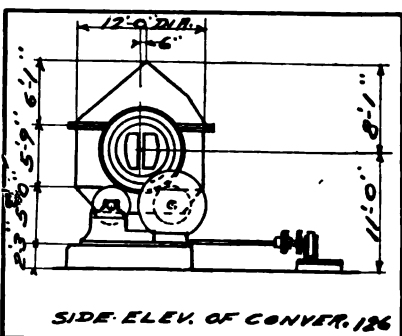
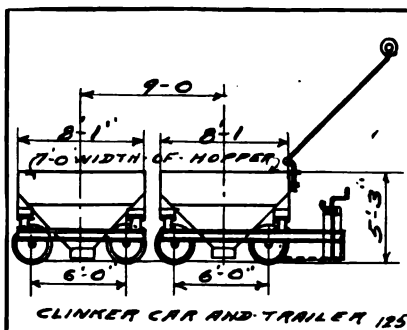
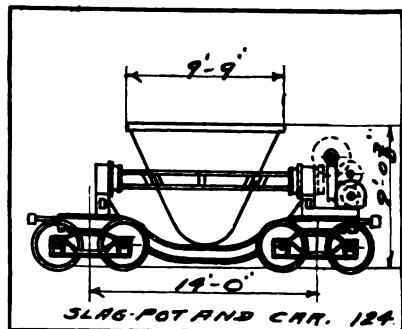
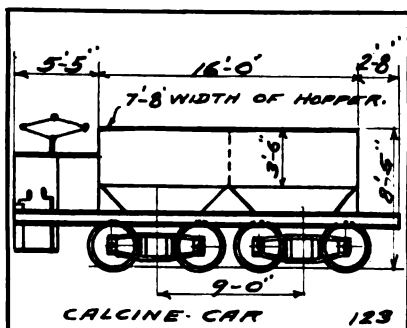
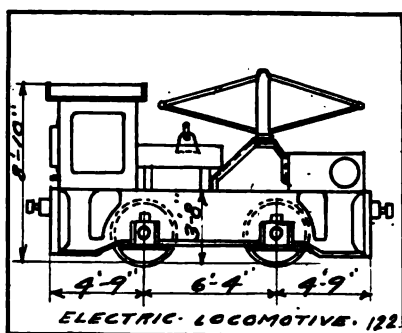
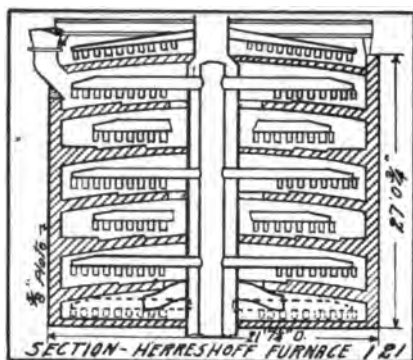


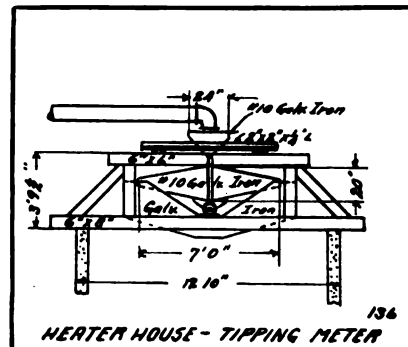
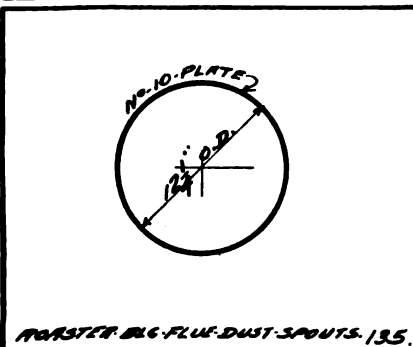
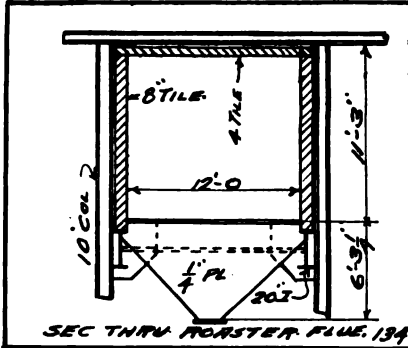
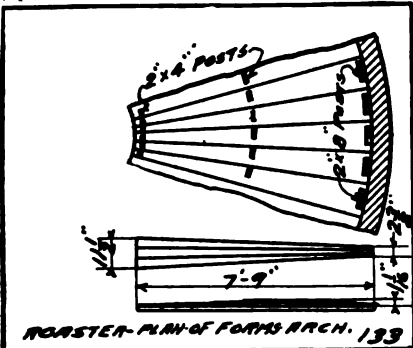
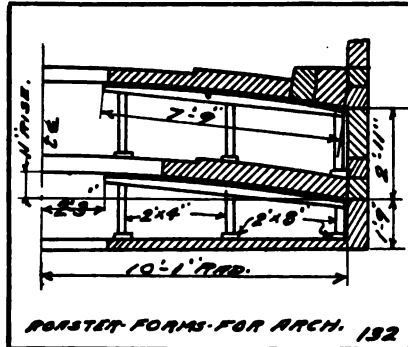
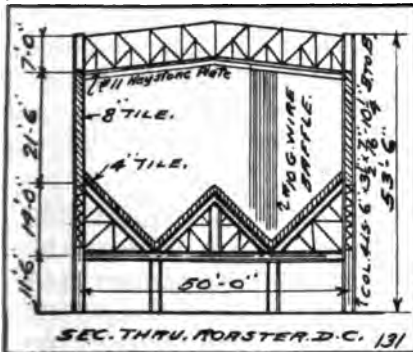
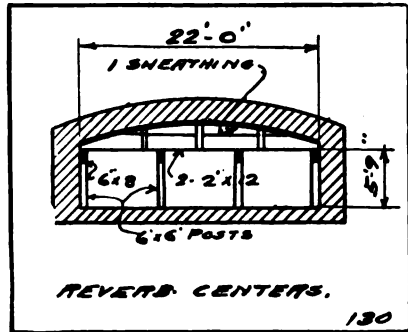
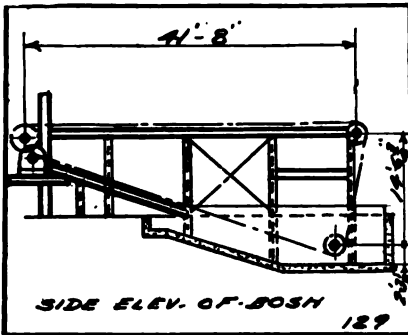












CHAPTER VI

DESCRIPTION OF COSTS

Engineering

Account 7100—Total Engineering Expense.

This account is a summation of accounts 7,101 to 7,206 inclusive, also of 9,000.1, power plant engineering expense. These accounts cover the engineering expenditures as their respective headings would indicate. As the total engineering expense they represent a percentage of the total cost of the smelter, less the engineering and indirect expenses, and have been so reported. In the making of any total estimate based on the unit costs derived from this sheet, it is assumed that of the total estimate so arrived at, 5.40 per cent. will be taken and added thereto to ascertain the item of engineering.

Account 7001—See Account 8999.

Account 7201—See Account 7100.

Account 7004—See Account 8999.

Account 7202—See Account 7100.

Account 7101—See Account 7100.

Account 7203—See Account 7100.

Account 7103—See Account 7100.

Account 7204—See Account 7100.

Account 7104—See Account 7100.

Account 7205—See Account 7100.

Account 7105—See Account 7100.

Account 7206—See Account 7100.

YARD TRACKS AND INDUSTRIAL SYSTEM

Tracks

Account 7301—Excavation.

This cost covers all of the excavating, barrow and grading incident to bringing the road beds of the New Smelter tracks to sub-grade. The materials worked varied from rock (Gila Conglomerate) through hard clayey soil filled with one-man stones, to light loam. The means employed to excavate covered the use of powder, plows, picks and shovels, slips and fresnos. The work covered by this cost was not carried on continuously, but as conditions about the plant permitted. The unit cost represents fairly the average cost of shallow excavating in large amounts about the smelter site.

Account 7302—Ties.

This account represents 700 steel ties, 7,656 white-oak ties laid in track, 524 white-oak ties in stock, 300 Oregon pine switch ties laid in track, and 1,082 white-oak ties used in temporary tracks and in handling machinery. There is also included here the labor of unloading, stocking, handling to points about the plant and placing upon the various grades. The steel ties are 7 ft. long, furnished with four No. 23 clips for 60-lb. A. S. C. E. rail, laid on 4 ft. 8½ in. gauge. They

were furnished by the Carnegie Steel Company and cost \$1.66 each. (See Fig. 10.) The white-oak ties are rough hewn, 6 in. by 8 in. by 8 ft. and cost \$1.05 apiece.

The switch ties were of Oregon pine, standard size and various lengths, according to their position in the track and the size of frog. They cost at the rate of \$30.00 a thousand, board measure. The account thus stands as follows:

Labor.....	\$425.13	
Wood ties.....	9,614.86	
Steel ties.....	1,162.93	
		\$11,202.92

The average cost of each wood tie in the track represents \$1.31. The ties were laid about 58 to the 100 ft. (See Fig. 10.)

Account 7303—Rails and Rail Fastenings.

The rails covered by this account were second hand, purchased from the Arizona and New Mexico Railway; 60 lb. A. S. C. E. standard. The track laid totaled 14,813 ft. long and was divided as follows:

On wood ties in dirt		On trestle	
Track No. 6.....	2,660 ft.	Track No. 7.....	294 ft.
7.....	1,023 ft.	15.....	403 ft.
8.....	1,940 ft.		
9.....	340 ft.	On steel ties	
10.....	645 ft.	Track No. 13.....	665 ft.
12.....	1,053 ft.		
13.....	1,396 ft.		
14.....	1,060 ft.		
15.....	1,685 ft.		
On wood ties in dirt		On steel ties	
Track No. 17.....	212 ft.	Track No. 14.....	283 ft.
18.....	279 ft.		
19.....	290 ft.		
24.....	585 ft.		

The account is divided up as follows:

29,626 ft., 60 lb. rail @ \$27.50 a ton.....	\$8,147.15
986 pr. angle bars @ \$0.50.....	493.00
4,000 bolts @ \$3.50 a hundred.....	140.00
4,000 nut locks \$1.07 a hundred.....	42.00
95 kegs spikes @ \$4.54.....	431.30
	\$9,254.25
Miscellaneous.....	585.54
	\$9,839.79
Labor.....	392.00

The miscellaneous item covers many second hand tie plates, short rail, material for extra angle bars made on the site, and a portion of temporary construction. The labor item represents handling, unloading and work on angle bars incident to different punching of rails.

Account 7304—Frogs and Switches

This item covers the cost laid in track of the following number of frogs purchased from the Cincinnati Frog and Switch Company:

9 No. 9 frogs with switches
7 No. 4 frogs with switches
2 No. 2½ frogs with switches.

It also covers labor incident to replacing 4 or 5 old frogs laid previous to the arrival of the new material. Included with the old installation labor is the cost of making several new switch points.

Account 7305—Laying, Surfacing and Ballasting.

This account covers the laying of all tracks in the industrial system about the smelter and the ballasting of such tracks where ballasting was required. The total amount of track laid was 17,130 ft., of which 697 ft. was upon steel trestles.

The ballast used varied according to the conditions prevailing at the time. A large amount of the track was ballasted with material borrowed near the site, while other portions were ballasted with red oxide tailings from the leacher at the cost of \$0.25 a yard, either dumped upon the track and spread with the use of the locomotive, or because of the excessive degree of track curvature for the locomotive, dumped and hauled to the place required.

The work of track laying and ballasting extended over many months, being prosecuted as the tracks were required for use, and in some cases as the steel arrived or ballast could be obtained. All tracks were standard gauge.

Trolley System

Account 7306—Poles and Setting.

This account covers the cost of stands, poles and ties to which the poles were attached as well as the labor incident to the erection of this work. There were 102 poles made of 4-in. pipe on an average of 16 ft. long set in a cast-iron stand 16 in. high. The accompanying sketches show four different conditions; double track with two poles used on the far sides of both tracks; double track one pole used between two tracks; single track and one pole and bracket on the under side of the slag track cut floor. (See Figs. 1, 2, 3 and 4.)

Account 7306.1—Brackets and Wiring.

This account covers all the material and labor incident to putting up the trolley line brackets, stringing the wire and insulating the work. The length of the trolley system is 7824 ft. of which 7,346 ft. is No. 000 H. D. grooved copper wire and 478 ft. of 25 lb. rail used through the slag track cut. The cost here has been figured on the lineal feet of system. Below is a list of the majority of items in the material account.

50 Insulator brackets similar to Ohio Brass Co. No. 1,254
200 Trolley brackets similar to Ohio Brass Co. No. 10,998
Six 15° frogs similar to Ohio Brass Co. No. 10,115
Six 15° frogs similar to Ohio Brass Co. No. 10,016
Three 15° frogs similar to Ohio Brass Co. No. 10,388

The messenger wire was $\frac{1}{4}$ -in. strand extra soft galvanized steel strand. Extension arms were made of $1\frac{1}{2}$ -in. pipe, 9 ft. long. One-quarter inch rod was used for arm brace. The voltage carried for the line is 230 D. C.

Account 7306.2—Rail Bonds.

This account covers cost of all material and labor incident to making the electrical bond throughout the trolley system. Each bond required the drilling of two $\frac{7}{8}$ -in. holes through the web of the rails. For the most part it was necessary to remove and replace the angle bars. Five hundred twenty-one No. 000 cable rail bonds with $\frac{7}{8}$ -in. hollow comp. studs, 1 ft. 11 in. center to center—see Fig. 5—type C. P. 2—A. S. & W. Co. furnished complete with drift pins. (See Fig. 5.)

Account 7306.30—Lighting.

This account covers the cost of all material and labor incident to lighting the trolley line. The electricity here used was taken direct from the trolley lines. Fifty-seven lamps—240 volt—120 watt carbon filament were installed. Four thousand feet of wire were used.

Rolling Stock**Account 7307—Cars, Electric Locomotives, etc.**

This account covers the cost of the following equipment together with the labor of unloading, trying out, removing the air brakes from the slag cars which came in on their own wheels, installing extra controllers on slag cars, and a large amount of repair work not carried by account 7307.1.

	Factory Cost	Freight	Total
Three 18-ton calcine cars, weight of each including electrical equipment 33,200 lb. from Kilbourne & Jacobs Mfg. Co.....	\$7,200.00	\$1,394.22	\$8,594.22
Four 225-cu. ft. capacity electrically operated slag cars, from M. H. Treadwell Co. Cars are side dumped by 15-h.p. motors; can also be dumped by hand. Weight of one car 57,000 lb.....	11,620.00	513.48	12,133.48
Two 18-ton electric locomotives from Westinghouse Mfg. Co. Draw-bar pull 8,000 lb. Each locomotive has two 84-h.p. motors. Weight of each locomotive is 43,950 lb.....	8,500.00	1,380.03	9,880.03
Weight of each locomotive is 43,950 lb...	8,500.00	1,380.03	9,880.03
One clinker larry car, 12½ tons capacity, 10-h.p. motor, weight 11,000 lb.....	1,450.00	176.00	1,626.00
Two trailer cars, 165 cu. ft. or 25,000 lb. capacity, weight of each car, 5,000 lb...	680.00	160.00	840.00
Two 15-h.p. 220-volt drum type controllers for electric locomotives to dump slag cars			165.64
One pair Schoen rolled steel wheels with axles for slag cars above.....			115.66
Miscellaneous material.....			662.60
			\$34,017.63

(See Figs. 122, 123, 124, and 125.)

Account 7307.1—Calcine Car Alteration.

The calcine cars with their pantographs on top were too high to operate successfully beneath the roasters. To overcome this difficulty the tracks beneath were lowered 1 ft. and the cab end of the car was cut down 18 in. in height. The pantograph was then placed on the cab end of the car. Much difficulty was also experienced with the pantographs themselves, especially when the cars were on short curves and taking switches. This account covers all the labor and material incident to overcoming these difficulties.

Trestle Approach to Reverberatory Building

Account 7308—Excavation.

The ground excavated was cemented sand and gravel, overlain with soil permeated with caliche. All the work was done by hand, using picks and shovels. The excavated material was cast to the side of the pier holes, and in some cases it was handled three times.

Account 7308.1—Foundation.

The kind of foundation here represented is the pier type. Sixty piers were cast 5 ft. by 5 ft. by 8 ft. One large abutment was 26 ft. high,

including its footing. The mixture of concrete used was 7 parts of gravel and sand to 1 part of cement, mixed by hand and transported in wheelbarrows, an average of 80 ft. There was no reinforcing, but two anchor bolts, $\frac{1}{2}$ in. by 2 ft. long, were placed in every footing. About 70 per cent. of the concrete was formed. The pier tops were finished to a perfect elevation. This is true of all other foundations where they support steel columns.

Account 7308.2—Steel Structure.

This steel trestle was a part of a contract between the Arizona Copper Co. and the Kansas City Structural Steel Co. who furnished erected the major portion of the structural steel about the plant, at a ton price of \$72.80 f.o.b. El Paso, and the corrugated iron at \$81 f.o.b. Pittsburgh. In every case where a steel structure occurs in this cost sheet, an additional amount of money has been expended for a variety of purposes. A new unit price is therefore obtained which varies with the structure in question. The extra expenses entailed are the freight from Pittsburgh and El Paso, the lumber for nailing strips, air lines and power for riveting and erecting, extra trackage to deliver steel within required distance of erection site according to contract, and many smaller items rightly debited here. The unit figure therefore arrived at gives the actual cost of the steel structure as it stands in every case. There were 163.97 tons of structural steel used here.

Account 7308.3—Woodwork.

This account covers the labor and material for the ties, walkways and railing upon the steel trestle. In the case of the ties they were laid for large part upon 173-ft. radius curve and required dapping to accommodate a $\frac{1}{2}$ -in. web projection of steel plate sticking up from each of the two girders which they spanned. The material used was as follows:

18,398 ft. b.m. 8 in. by 8 in. ties o.p.	\$652.98
9,248 ft. b.m. 2 in. by 4 in.; 4 in. by 4 in.; and 2 in. by 12 in. 872 lb. $\frac{1}{2}$ -in. round iron.....	21.37
Nails, bolts and miscellaneous.....	94.57
	\$768.92

150 Ton Track Scales—Receiving Yard

Account 7309—Excavation.

The excavation was in tight sand and gravel. It was done with pick and shovel, handled into cars and hauled 500 ft.

Account 7309.1—Foundation.

This was a job of plain concrete mixed 8 parts sand and gravel to 1 part cement, hauled 2,000 ft. by teams and wheeled in barrows 50 ft.

to place. The concrete was principally walls with a few piers. Eighty per cent. of the exposed surface, other than top and bottom, was formed. A great many $\frac{3}{4}$ -in. bolts were set in the concrete. (See Fig. 7.)

Account 7309.11—Cost and Erection.

This account covers the material and erection of one 50-ft., 150-ton suspension platform track scale with type-registering beam, graduated by 10 lb. The platform was arranged for two gauges of dead and live rails—4 ft. $8\frac{1}{2}$ in. and 3 ft. The scale was furnished by Fairbanks and Morse. The material portion of this account is divided up as follows: (See Fig. 7.)

	Cost	Freight	Total
One 50-ft. 150-ton suspension platform track scale.....	\$1,351.00	\$486.09	\$1,837.09
294 rail clips.....	45.00	77.36	362.91
75 stands and 72 rail blocks.....	211.85		
Patterns for blocks and stands.....	28.70		
Structural steel for track scales.....			1,089.62
Twenty-five 10 by 12 6 ft. ties and 2 by 12 planks covering.....			55.83
Bolts, washers, round iron, nails, etc.....			105.41
			\$3,450.86

Account 7309.30—Scale House.

This account covers the cost of material and erection of the scale house. The building in plan is 9 ft. 6 in. by 10 ft. with a shed roof. In front it is 10 ft. 6 in. high, and in the rear 8 ft. The studding is 2 in. by 4 in. and the rafters are 2 in. by 6 in. The siding is corrugated iron and on the roof is 1-in. sheathing and composition roofing paper. There are two windows in both front and rear, and a door in each end. (See Fig. 8.)

Bridge No. 1

Account 7310.1—Foundation.

The concrete covered by this account consisted of two footings about 6 ft. by 6 ft. by 55 ft. with two abutments about 3 ft. by 14 ft. by 55 ft. at base to 14 ft. at top. It was plain concrete, 7 parts sand and gravel and 1 part cement, mixed by hand and by machine and wheeled 45 ft. to place. Ninety per cent. of the vertical and inclined surfaces were formed.

Account 7310.10—Bridge No. 1 Steel Work.

This work was covered by the Kansas City Structural Steel Co.'s contract. (See account 7308.20.) The bridge consisted of two girders—each of three 18-in. by 50-lb. by 20-ft. I-beams, connected with bolts and separators and anchored to the foundations with four 1-in. bolts.

Culvert No. 1**Account 7311—Culvert No. 1 Masonry.**

This was a stone culvert 534 ft. long, with inside dimensions 4 ft. by 4 ft. The top was built of old 50-lb. rails at \$15 a ton, spaced 8-in. centers and rendered tight with stones set with cement mortar in between rails. There is 20 ft. or more fill over the culvert at various points. The stones laid in cement mortar for the sides and bottom were obtained on the site.

Retaining Walls**Account 7312—Excavation.**

This was a long narrow cut through fill, earth, and sand and gravel. It was taken out with picks and shovels and transported 200 ft. with slips.

Account 7312.1—Concrete.

The concrete covered by this account was a wall of gravity section, 8 in. at the top, of various heights and 80 ft. long. The mixture used was 7 parts sand and gravel to 1 part cement, wheeled 50 ft. Half of the yardage was hand mixed and half machine mixed.

Account 7312.20—Masonry.

This wall was built of stone which was gandy to the site and was laid in cement mortar. The wall was 124 ft. long, 12 in. to 18 in. thick and from 2 to 5 ft. high.

40-Ton Track Scales on Calcine Track**Account 7313—Excavation.**

The excavation here covered was a small rectangular cut through tight, red soil, filled with large stones. It was picked, shoveled and wheeled in barrows 50 ft.

Account 7313.1—Foundation.

The concrete under this account was small walls about 30 ft. by 5 ft. by 22 in. and a 10-in. slab. It was mixed 7 parts sand and gravel to 1 cement by hand and wheeled 25 ft. to place. About twenty-six $\frac{3}{4}$ -in. bolts were set in the concrete. Eighty-five per cent. of the vertical surface of this concrete was formed. (See Fig. 9.)

Account 7313.20—Cost and Erection.

This account covers the cost of the material and its erection of one 40-ton track scale. The scale platform is 24 ft. long with one 4-ft. 8 $\frac{1}{2}$ -in. track passing over it. It has a type registering beam. The scales were furnished by Fairbanks Morse Co. Itemized, the material accounts stand as follows:

	Cost	Freight	Total
1 40-ton track scale complete.....	\$400	\$116.86	\$516.86
6-in. by 6-in. ties; 8-in. by 12-in. stringers. 2-in by 12-in. decking.....			26.19
6-in. and 5-in. channels and 5-in. I-beams.....			102.85
Strap steel, bolts, pipe, hauling, etc.....			64.95
			\$710.85

(See Fig. 9.)

Account 7313.30—Scale House.

Same as 7309.50—practically.

Trestles to Receiving Bins

Account 7314—Excavating.

This excavation covered two large abutments 6 ft. in the ground and 10 piers going about 15 ft. into the ground. The material excavated was earth and adobe. It was handled in the pier footings by a windlass.

Account 7314.1—Foundation.

The concrete here covered was in two large abutments and in 10 piers. The abutments were about 8 ft. by 16 ft. by 24 ft. and the piers 7 ft. by 7 ft. by 23 ft. Forty-eight $\frac{3}{4}$ -in. bolts were set in the concrete. Twenty per cent. of the vertical surfaces were formed. The concrete was mixed in proportions of 7 parts sand and gravel to 1 part cement. A large amount of boulders were used in the piers. The concrete was both hand and machine mixed, and was wheeled in barrows an average of 200 ft.

Account 7314.2—Steel Structure.

There were 109.53 tons of structural steel used here.

Account 7314.30—Woodwork.

The woodwork here was practically the same in every respect as 7308.50, with additional walkways of 2 by 12 planks, nailed to strips bolted to the steel.

16,920 ft. b.m. 8 by 8 ties and 6 by 6 guard rails.

10,286 ft. b.m. 2 by 12, 2 by 4, 4 by 4 walk and 27,206 ft. total b.m. railings was used.

(See Fig. 6.)

Receiving Bins

Account 7401—Excavation.

This work covered the digging of a number of piers 7 ft. by 7 ft. to a depth ranging from 16 ft. to 25 ft. into gravel. The dirt was easily dug but had to be handled from the lower half of the holes with windlasses. It was carted away at the top 225 ft.

Account 7402—Foundation.

Only 5 per cent. of this concrete was formed. The lower part was machine mixed in proportions of 12 gravel and sand to 1 cement, while the upper part was 6 to 1. It was wheeled 200 ft. to place. The top surfaces were trowel finished to a perfect elevation for receiving the steel.

Account 7403—Steel Structure.

There were 11.35 tons of corrugated and 341.74 tons of structural steel used here. (See account 7308.2. See Fig. 11.)

Account 7404—Gates.

This account covers the cost of material, unloading hauling, fabrication, alteration, and erection of 30 gates beneath the receiving bins. All cast-iron parts together with operating wheel, shaft and gate itself were purchased outright.

The chutes attached to the gates were fabricated in the new smelter shops. The parts were assembled in the field and there erected. The holes in the steel structure to which the gates were attached had to be rebored in the field. The counterweights for the 12 coarse ore-bin gates were made on the job and erected. These 12 counterweighted gates are opened by rack and pinion, operated by a hand wheel and cut up through the stream. The chutes to guide the ore to the feeder are of $\frac{3}{8}$ -in. plate, while the gate is $\frac{5}{8}$ -in. The other 16 gates for the concentrate bin are similar to the above, but are not counterweighted and cut down through the stream. (See Fig. 12.)

Account 7405—Conveyor No. 1.

This account covers the entire labor and material connected with the installation of conveyor No. 1. It does not include the steel frame to which the idlers are attached, but does cover the cost and installation of the traveling feeder with the necessary ties, rails, wire and motor. This segregation of charges is true of all conveyor costs given in this Cost Sheet. All the conveyors were furnished by the Robins Conveying Belt Co. Conveyor No. 1 was a 30-in. belt, making a 97-ft. conveyor with a 3-ft. rise, operating at a speed of 150 ft. per minute, capable of handling 100 tons per hour. It is supplied with 12-in. material from the bins above it through a speeded feeder. (See Fig. 6.) The material account is segregated as follows:

	Cost	Freight	Total
202-ft. 30-in. belt.....	\$686.80	\$46.45	
Feeder belt.....	56.75		
Conveying idlers, etc.....	1,497.12	185.09	
Cent. switch.....	34.00	2.20	
Broken pulley.....	44.00		
Miscellaneous material.....	307.73		
One 5-h.p. motor.....	87.05		

\$2,713.45 \$233.74 \$2,947.19

Account 7406.01—Conveyor No. 22.

This conveyor is similar to No. 1. It has a 20-in. belt, making a conveyor 117 ft. long, with a 3-ft. rise, operating at a speed of 200 feet per minute with a capacity of 150 tons and taking $\frac{3}{4}$ -in. concentrates through a speeded feeder. The material account is segregated as follows:

	Cost	Freight	Total
241-ft. 6-in. to 20-in. belt.....	\$432.38	\$31.94	
Feeder belt.....	56.75		
Robins material.....	1,367.50	169.07	
Cent. switch.....	34.00	2.21	
Miscellaneous material.....	317.14		
5-h.p. motor.....	87.04		
	\$2,294.81	\$203.22	\$2,498.03

(See Fig. 13.)

Account 7407—Lighting.

The receiving bins were lighted with 22 drop lights using 100 volts A. C. current.

Crushing Plant**Account 7701—Excavation.**

This was a large rectangular cut for the Crusher Building made through cemented sand and gravel, with streaks of soil running through the cut hardened by caliche. The work was done with pick and shovel and handled by wheelbarrow into carts and hauled 225 ft.

Account 7702—Foundation.

This concrete was reinforced with $\frac{1}{2}$ -in. and $\frac{3}{4}$ -in. round, medium steel rods. It was cast in walls, 12 in. thick, about 12 ft. high and as a 12-in. floor slab. The concrete was machine mixed in the proportions of 5 sand and gravel to 1 cement and about 60 per cent. of the vertical surfaces were formed. It was wheeled in barrows 400 ft. up an 8 per cent. grade.

Account 7703—Steel Structure.

There were 5.17 tons of corrugated iron and 19.90 tons of structural steel used. (See account 7308.2.)

Account 7703.1—Doors, Windows and Frames.

The material here used for openings was as follows:

9 windows and frames 3 ft. 10 $\frac{1}{2}$ in. by 7 ft. 8 in., 24 lights. }	\$138.40
4 sash and frames, 2 ft. 11 $\frac{1}{2}$ in. by 3 ft. 11 $\frac{1}{2}$ in., 9 lights. }	
Balance, butts, catches, etc.....	25.38
Lumber for sills.....	6.93
	\$170.71

Account 7703.2—Painting Woodwork.

All the woodwork was painted with two coats of lead and linseed oil in cream color.

Account 7704—Crushing Machinery.

This account covers the material cost and labor of installing the following machinery:

One 36-in. by 18-in. Farrell Crusher, second hand, weight 50,000 lb.....	\$1,000.00
Miscellaneous lumber.....	93.61
	\$1,093.61

Account 7704.1—Chutes.

This account covers the cost of material noted below, the labor of fabricating the spouts, hoppers, and their erection:

- 1 Grizzly screen 3 ft. by 8 ft. made of 1-in. bars, having 2½-in. by 2½-in. openings, framework made of ⅝-in. plate and 3-in. by 3-in. angles.
- 1 Spout for grizzly, dumping on conveyor No. 4 made of ⅝-in. steel plate and 2-in. by 2-in. by ½-in. angles. ¾-in. C.I. liners used.
- 1 Hopper for crusher made of ⅝-in. steel plates 2½-in. by 2½-in. by ½-in. angles. ¾-in. C.I. liners used.

Account 7705—Shafting, Pulleys and Belting.

This account covers the material cost and erection of the following:

1 Pc. shafting 4 ⅞ in. by 6 ft. 6 in. with two collars.....	\$29.64
1 Pc. shafting 4 ⅞ by 5 ft. 3 in.....	16.51
Two 4 ⅞-in. rigid pillow blocks.....	58.55
One 48-in. by 11-in. split pulley.....	47.59
One 36-in. by 16-in. split pulley.....	43.71
Two 4 ⅞-in. safety collars	5.39
43 ft. 10-in. two-ply leather belt.....	60.80
42 ft. 14-in. two-ply leather belt.....	220.37
Miscellaneous	0.79
	\$483.35

Account 7706—Motor.

This account covers the cost of the following material and the installation:

One 50-h.p. Crocker-Wheeler squirrel-cage motor with starter	\$478.41
Lumber for housing motor.....	35.23
	\$513.64

Account 7707—Power Wiring.**Account 7707.1—Lighting.**

This account covers the cost and installation of the following material:

8 drops 16 candle power
30 ft. brewery cord
350 ft. No. 12 weatherproof wire
45 ft. 1-in. conduit.
Switches, bolts, etc.

Sampling Plant

Account 7801—Excavation.

This account covers the excavation of the sampling plant and the necessary backfill tamped in 5-in. layers in the low parts where the basement concrete floor was cast. It was done with picks, shovels and wheelbarrows, through earth, sand and gravel.

Account 7802—Foundation.

This concrete was cast in the walls and piers of the sampling plant. It was mixed by machine in the proportions of 7 parts sand and gravel to 1 part cement and wheeled in barrows 150 ft. Ninety per cent. of the vertical surfaces of the concrete was formed. The cost of all anchor bolts as well as the finish to exact level for building columns is included here.

Account 7802.1—Concrete Ground Floor.

This was plain concrete floor laid with sand joints in about 6-ft. square blocks 5 in. thick, in the proportions of 5 parts sand and gravel to 1 cement. The top finish was 1 in. thick, made 2 parts sand to 1 cement, troweled smooth. The concrete was mixed by machine and transported 175 ft. in barrows.

Account 7802.2—Reinforced Concrete Floors.

This concrete was cast over steel I-beams, using forms between the steel beams. The mix was the same as the above floor with the same top finish. The reinforcing used was Clinton welded fabric 2-in. by 12-in. mesh. The floor was $4\frac{1}{2}$ in. thick. The concrete was mixed by machine and transported 500 ft. to place by wagon, wheelbarrow and hoist.

Account 7803—Steel Structure.

There were 13.46 tons of corrugated iron and 97.39 tons of structural steel used. (See account 7308.2.)

Account 7803.1—Doors, Windows and Frames.

This account covers the purchase price and erection cost of material enumerated below. The doors are not given in the list, as they were made upon the job, but correspond to the frames noted. Necessary hardware is also included in the cost.

Fourteen 24-light windows 3 ft. $9\frac{1}{2}$ in. by 7 ft. 8 in. by $1\frac{1}{2}$ in.
with frames

Twenty-nine 40-light windows 7 ft. $5\frac{1}{2}$ in. by 3 ft. $10\frac{1}{2}$ in. by
 $1\frac{1}{2}$ -in. with frames

Two 9-light windows 2 ft. 11½ in. by 3 ft. 11½ in. by 1½ in.
with frames

One 4-ft. 8-in. by 7-ft. 4-in. door frame

One 9-ft. 2-in. by 8-ft. 10-in. door frame

Three 4-ft. 8-in. by 7-ft. 4-in. door frames

Five 3-ft. 8-in. by 7-ft. 4-in. door frames

One 3-ft. 8-in. by 7-ft. 2-in. door frame

One 9-ft. 2-in. by 9-ft. 2½-in. door frame

\$564.90

(See Fig. 14.)

Account 7803.11—Painting Doors and Windows.

This covers the material and labor of applying two coats of linseed oil and white lead.

Account 7804—Shafting, Pulleys and Belting.

This is not a good cost. The labor is unquestionably too low and has been absorbed by some of the following accounts up to account 7807.5. Eighty-five feet of shafting, varying in size from 1½ in. to 3⅞ in. with 28 pulleys of various diameters and face with the various hangers, collars, etc., and 1,325 ft. of 3-in. to 12-in. leather belting were to be taken care of here. The material is correct. (See Fig. 108.)

Account 7805—Motors.

The material covered by this account is as follows:

One 15-h.p. squirrel-cage motor.....	\$189.94
One 75-h.p. slip ring.....	644.00
Overload release.....	14.25
Miscellaneous.....	39.38
	\$887.57

Account 7806—Power Wiring.

Account 7806.1—Lighting.

36 drop lights.....	
No. 12 weatherproof wire used in conduit.....	\$140.57

Account 7807—Rolls and Samplers, Cost and Erection.

This cost covers the price and installation of the following material, together with the application of two coats of Dixon's Silica Graphite paint upon the Rolls, Samplers and Chutes:

	Cost	Freight	Total
2 sets 24 by 12 rolls, from Chalmers and Williams.....	\$1,330.00	\$344.16	\$1,674.16
1 set 42 by 16 rolls, from Chalmers and Williams.....	1,567.50	482.40	2,049.90
1 set 48 by 12 rolls, from Chalmers and Williams.....	1,710.00	716.43	2,426.43
One 27-in. Snyder sampler with 28- in. by 3½-in. pulley T. & L.....	38.00	5.04	43.04

	Cost	Freight	Total
Two 42-in. Snyder samplers with 40-in. by 3½-in. pulley T. & L.	133.00	29.95	162.95
1 No. 1 Vezin sampler with spout arranged for 5 per cent. cut, Allis-Chalmers.	162.00	20.51	182.51
1 No. 3 Vezin sampler with spout arranged for 10 per cent. cut, Allis-Chalmers.	220.00	45.64	265.64

	Cost	Freight	Total
Two 5-T steel plate crawls for 12-in. I's. . .	\$104.00	\$15.76	\$119.76
One 4-T steel plate crawls for 10-in. I's. . .	40.00	6.11	46.11
Two 3-T steel plate crawls for 9-in. I's. . .	64.00	8.61	72.61
One 5-T Triplex chain block.	112.00	14.49	126.49
One 4-T Triplex chain block.	88.00	9.10	97.10
Structural steel, machine parts.			632.44

Miscellaneous, etc. \$7,899.14
(See Figs. 15, 16, 110, 111 and 112.)

Account 7807.10—Cast Iron Liners and Drying Pan, and Erection of Chutes.

This account covers the erection of account 7807.5, and the cost and erection of the following:

1 Sample drying pan 8 ft. ½ in. by 5 ft. 2½ in.	\$136.11
¾-in. C.I. liners for chutes, castings c.	865.75
	\$1,001.86

Account 7807.20—Elevators.

Below is a description of the elevator with the material cost. The erection is too low and not usable.

33 ft. between head and tail shafts	
Head and tail pulley 34 in. by 9 in.	
Drive pulley 23 in. by 6 in., 40 h.p.m.	
All housed with No. 10 plate steel casing	
72 ft. 6 in. of 8-in. 6-ply rubber belt	
Forty-eight 6-in. by 4-in. malleable buckets, style A.A. Mfrs. std.	
Liners for casing made of white iron ¾ in. and 1 in. thick	
Cost.	\$458.22
(See Fig. 11.)	

Account 7807.5—Steel Chutes.

This is the average cost per pound of material and labor for fabricating all steel chutes used in the sample mill. In general they were made of ⅜-in. plate and light angles. The cost of erecting is in 7807.1 and in the comparative costs are found the individual chute costs. (See Fig. 109.)

Account 7809—Keystone Plate Partitions.

1,523 sq. ft. of No. 14 Keystone Plate partitions were erected by riveting the plates together and attaching them to the structural steel of the building. The plates were originally intended for a roof upon the Roaster Dust Chamber and had each long edge turned up $1\frac{1}{2}$ in. The cost of cutting these edges off is here included. (See Fig. 113.)

Account 7810—Alteration of Chutes and Machinery.**Bedding Plant and Bunker Bins****Account 7901—Excavation.**

This excavation involved making long, deep, oblong cuts through earth and sand and gravel bonded with caliche. It was necessary to use power to shake up the ground, followed in some cases with plows. A part of the work was handled with slips and fresnos; another part by picks, shovels and wagons. The average haul was 600 ft.

Account 7902—Foundation.

This concrete yardage was made up as follows:

550 ft. footing, 4 ft. wide by 2 ft. deep—plain	
550 ft. wall, 1 ft. 6 in. thick by 5 ft. high—plain	
1200 ft. footing, 3 ft. 6 in. wide by 10 in. deep—plain	
600 ft. wall, 1 ft. 6 in. thick by 6 ft. high—plain reinforced coping	
600 ft. wall, 1 ft. 6 in. thick by 11 ft. high—plain	
320 ft. footing, 7 ft. wide by 3 ft. deep, reinforced with $\frac{3}{4}$ -in. and $\frac{1}{2}$ -in. rods, 6 in. on centers	
320 ft. wall, 4 ft. 5 in. thick by 11 ft. high, reinforced with $\frac{3}{4}$ -in. rods, 12 in. on centers	
12,700 sq. ft. rough finished slab, 5 in. thick	

The concrete, of which 85 per cent. of the vertical surface was formed, was machine mixed, in the proportions of 7 parts sand and gravel to 1 part cement. It was wheeled to place in barrows, a distance on the average of 180 ft. About half of the yardage was reinforced.

Account 7903—Steel Structure. (See account 7308.2.)

Structural steel	510.41 tons
Corrugated iron	38.30 tons
	548.71 tons

(See Fig. 114.)

Account 7904—Conveyors 7¹, 7². (See account 7405, and Fig. 91.)

Conveyor 7¹ has a 20-in. belt, 180 ft. 3 in. from center line to center line of head and tail pulleys with an 8-ft. 9-in. rise, operating at a speed of 300 feet per minute, with a capacity of 150 tons per hour. It takes $\frac{3}{8}$ -in. material which it unloads through a 20 in. automatic tripper. It brings the fines from the Conveyor 5 to the bedding conveyors. Conveyor 7² has a 20 in. belt, 200 ft. long from center line to center line of head and tail pulleys with a 9-ft. 2-in. rise, operating at a speed of 400 feet per minute, capable of handling 100 tons per hour. It takes $2\frac{1}{2}$ -in. material from Conveyor 6 which it unloads through an automatic tripper into the Bunker Bins beneath. The original installation here called for a bedding tripper at \$950, which was superseded by an automatic tripper. Both charges are in this account. For a proper unit cost here, this charge of \$950 should be deducted.

The woodwork noted below in the segregated charges includes walkways of two 2 by 12's beside both conveyors, as well as decking.

	Cost	Freight	Total
Belts.....	\$1,433.79	\$102.35	\$1,536.14
Conveyor material.....	1,451.06		
2 automatic trippers and track (used).....	1,228.38	369.02	3,998.46
1 ore bedding tripper (discarded).....	950.00		
Lumber.....			253.74
7½-h.p. motor conveyor 7 ¹			138.75
10-h.p. motor conveyor 7 ²			156.80
Centrifugal switch.....			72.40
Drive Belts.....			54.80
Wire for motors.....			97.26
Painting material.....			20.88
Miscellaneous.....			44.44
Total.....			\$6,373.67

Account 7904.1—Conveyors 8¹, 8², 8³.

(See account 7405.)

Conveyors 8¹, 8², 8³ are practically identical, running out over the three beds. The exception is that conveyor 8³ has a 6-ft. in place of a 3-ft. rise at the start and is 5 ft. longer. In general the three belts are 20 in. wide, 186 ft. 9 in. long, from center line to center line of head and tail pulleys, with a 3-ft. rise, operating at a speed of 400 ft. per minute, with a capacity of 150 tons per hour, taking a $\frac{3}{8}$ -in. material and distributing it through a bedding tripper onto the beds below.

This account segregated shows as follows:

	Cost	Freight	Total
Belts.....	\$2,124.73	\$152.34	\$2,277.07
Conveyer material.....	1,793.55		
3 ore bedding trippers.....	2,850.00	569.53	5,213.08
Lumber, walkways, decking.....			237.13
Three 15-h.p. motors.....			569.82
Centrifugal switch.....			108.60
Drive belts.....			72.41
Electrical supplies.....			145.89
Painting material.....			31.32
Miscellaneous.....			63.66
Total.....			8,718.98

(See Fig. 91.)

Account 7904.2—Conveyors 9¹, 9², 9³, 10¹, 10².

(See account 7405.)

All these conveyors are on wooden supports, the material for which and the cost of walkways is here included.

Conveyors 9¹, 9², 9³ are identical. They take the reclaimed material from the beds to conveyor 10¹. They are 20-in. wide belts, 198 ft. 4 in. from center line of head pulley to center line of tail pulley perfectly flat, operating at a speed of 300 feet per minute, with a capacity of 100 tons per hour.

Conveyor 10¹ takes the material from 9¹, 9², 9³ to conveyor 11. It is 145 ft. 3 in. from center line of head pulley to center line of tail pulley, perfectly flat, operating at a speed of 300 feet per minute, with a capacity of 100 tons per hour.

Conveyor 10² takes the material from the Bunker Bin gates to Conveyor 14. It has a 20-in. belt, is 165 ft. long from center line of head pulley to center line of tail pulley, is perfectly flat, operating at a speed of 300 feet per minute, with a capacity of 100 tons per hour. It uses one feeder below the gates the same as on Conveyor 2.

The material segregated is as follows:

	Cost	Freight	Total
Belt.....	\$3,300.76	\$228.98	\$3,529.74
Feeder belt on 10 ²			56.76
Conveyor material.....	2,718.25		
Feeder conveyor 10 ²	925.00	586.43	4,229.68
Lumber for walkways, decking, framework, etc.....			755.61
5 overload releases.....			71.25
5 centrifugal switches.....			181.05
Five 5-h.p. motors.....			435.20
Drive belts.....			106.28
Electrical supplies.....			243.13
Painting material.....			52.19
Miscellaneous.....			95.30
Total.....			\$9,756.19

(See Fig. 115.)

Account 7904.3—Bunker Bin Gates.

These are cast-iron chutes about 22 in. square with an arc gate controlling the discharge through the bottom. All cast iron is $\frac{3}{4}$ -in. thick, save the $\frac{5}{8}$ -in. wearing plate upon the arc gate. The operating lever was furnished in the structural steel contract.

42 spouts $\frac{1}{2}$ -in. cast iron	
42 gates $\frac{1}{2}$ -in. cast iron	
42 cover plates $\frac{1}{2}$ -in. cast iron	\$1,021.64

Account 7904.4—Chutes for Conveyors 7¹ to 10² inc.

This account covers the fabrication, erecting and material in the chutes directing the ore from one belt to another at the Beds and Bunker Bins. Also included here are the cast-iron wearing plates for lining the Chutes.

The chutes are made of $\frac{3}{16}$ -in. plate and the necessary angles. The wearing plates are $\frac{3}{4}$ in. to 1 in. thick, hard, white cast iron.

Account 7905—Two Reclaimers.

(See Fig. 116.)

Account 7905.1—Two Reclaimers Wiring.

This account covers the material cost and labor of installing a double trolley wire on three beds together with the wiring of two reclaimers and switchboards. The material is wire, condulets, circuit breakers, and the like. (See Fig. 118.)

Account 7906—Lighting.

This account covers the labor and material used in lighting the Bedding Plant and Bunker Bins.

63 carbon lamps, 16 candle power
455 ft. brewery cord
1,395 ft. No. 8 weatherproof wire
1,835 ft. No. 12 weatherproof wire
205 ft. $1\frac{1}{2}$ -in. conduit
285 ft. 1-in. conduit.

Account 7907—Transfer Car.

This car was a structural steel frame about 4 ft. high and 62 ft. by 18 ft. in plan. It transfers the Reclaimers from one bed to another, or to the repair shed under its own power, using a $7\frac{1}{2}$ -h.p. motor, getting direct current through a trolley. It was furnished by the Robins Conveying Belt Co. (See Fig. 117.)

Account 7908—Signal System.

This account is of no value.

Roasting Plant

Account 8101—Excavation.

This covered a large surface grade made to the required elevation of the site, made by plows and fresnos and hauling the dirt to a railroad grade an average of 450 ft. This was followed by picks, shovels and carts, making deep cuts to gravel through red clay and boulders for the steel foundation.

Account 8102—Foundations.

This concrete was all cast as piers with at least $2\frac{3}{4}$ -in. bolts in each pier. Only 10 per cent. of the vertical surfaces was formed, though many of the piers were 6 ft. deep. One-half of the concrete was machine mixed and one-half was hand mixed in the proportions of 7 parts sand and gravel to 1 cement. The pier tops were finished to a perfect elevation to receive the steel columns. The concrete was wheeled about 60 ft.

Account 8103—Steel Structure.

(See account 7308.2.)

There were 23.16 tons of corrugated iron and 422.12 tons of structural steel used. (See Figs. 117 and 118.)

Account 8103.1—Elevator.

This account gives the entire labor and material incident to erecting a 52-ft. high 10-ft. 6-in. by 8-ft. 6-in. platform elevator in a self-supporting structural steel frame. It does not include other than the excavation and concrete pit. The account segregated stands thus:

Structural steel erected for elevator frame and tower.....	\$840.97
One 5-T electric hoist with 240-volt D.C. motor.....	1,261.60
3,792 b.f. lumber.....	11.56
Rope sheaves, counterweights, etc.....	29.23
Labor.....	37.79
Miscellaneous.....	8.47
	\$2,189.62

The labor installed the motor, hoist and wood platform.

Account 8104—Roasters, Cost and Erection.

This account covers the cost and erection of the roaster shells as furnished by the Kansas City Structural Steel Co. and the roaster equipment, namely, the central shaft, rabble arms, rabbles, driving mechanism, doors to roasters, cast-iron rings, etc. As segregated the material account shows below. The Herreshoff furnaces have 6 superimposed hearths and a top drying hearth. The arms are cooled by air furnished by two motor-driven fans. The diameter of the shell is 21 ft. $7\frac{1}{2}$ in. outside and 18 ft. 2 in. in height.

	Cost	Freight	Total
8 furnaces from Pacific Foundry Co. designed by Gen. Chemical Co. \$36,278.12	\$36,278.12	\$8,016.73	\$44,294.85
Eight 1-ton steel trolleys for 6-in. I's.....			155.30
(See Fig. 121.)			
One 1-ton duplex chain block.....			27.47
Steel shells erected—KCSSCo.....	10,192.73	840.06	11,032.79
Power for riveting.....			452.25
Miscellaneous.....			363.43
			\$56,326.09

Account 8104.1—Roaster Alteration.**Account 8105—Brickwork.**

This account covers the cost of the brick, mortar materials, etc., and labor of the mason with helpers of installing the brick in the roasters. The unloading of the brick from cars, the centers and carpenter labor are taken care of elsewhere. The brick here used were in the main special shapes and 95 per cent. of them were hard burned red brick. In all there were 15 different shapes. The hearths were laid dry and the shell brick laid with slimes from the copper company's concentrator. (See Fig. 133.)

Account 8105.01—Brick Unloading.

This covers the cost of leveling ground, checking up quantities, and unloading of brick from cars to roasters. This total cost includes the unloading of all brick for roaster use, the actual amount needed plus the extra not used.

Account 8105.02—Brickwork Centering.

This account covers the cost of making, installing, removing and the material for 16 sets of centers used for putting in 48 hearths. (See Fig. 133.)

Account 8106.01—Roaster Flue-spouts.

This covers the cost of material and erection of 10 spouts with gates from the roasters' common flue. The material is as follows:

10 cast-iron gates for 12½ in. diameter spouts.

10 spouts of No. 10 plate, 24 ft. long, 12½-in. outside diameter fastened to base of hopper by 2 in. by 2 in. angle collar.

10½ in. plate slides for gates.

(See Fig. 135.)

Account 8106.02—Tile Work.

This account covers the mason labor, carpenter labor, cost of tile and unloading, mortar materials and a lumber charge for scaffolds used in this flue. The flue is built of tile and is about 50 ft. from the ground. The tile for the roof was laid between T iron spanners from wall to wall. The mortar was lime. (See Fig. 132.)

Account 8106.03—Painting Flue.

The inside of the above flue was given one coat of silicate of soda. The account covers this material and labor cost.

Account 8107—Shafting, Pulleys and Belting.

Below is the material list erected under this account. The shafting was attached directly to the steel frame of the building.

47 ft. of 3 $\frac{3}{8}$ -in. shafting	}		\$1,500.50
103 ft. of 2 $\frac{1}{2}$ -in. shafting			
Two 20-in. by 12-in. pulleys			
One 36-in. by 10-in. with clutches for each Roaster			
150 ft. 10-in. 6-ply rubber belt	}		471.85
310 ft. 8-in. 7-ply rubber belt			
Miscellaneous			
			27.54
			\$1,999.89

Account 8108—Motor.

This covers the cost of material and labor of installing one 30-h.p. motor to drive the Roasters. It is located directly upon the first steel floor of the Roaster Building.

One 30-h.p. squirrel-cage motor	\$267.80
One overload release	14.25
Miscellaneous wire, insulators, belt, etc.	181.91
	\$463.96

Account 8109—Lighting.

The Roasters are furnished with light on all floors.

Account 8112—Motor-driven Fans.

This covers the price and cost of installing upon their foundations 2 motor-driven fans, which furnish the air to cool the roaster arms. They are 55-in. double width, full housing conoidal fans, direct connected, each with a 25-h.p. squirrel-cage induction motor.

Each fan has a capacity of 22,000 cu. ft. of air per minute against a pressure of 1 $\frac{1}{2}$ in. water.

	Cost	Freight	Total
2 fans and motors	\$1,203.00	\$199.49	\$1,402.49
Miscellaneous			3.42
			\$1,405.91

Account 8112.1—Blast Pipe.

This account covers the material price, cost of fabrication and installation of 240 ft. of blast pipe. The installation referred to is connecting up and riveting the pipe in place in the field only. The pipe is made of No. 10 and No. 12 plate and varied in diameter from 18 in. to 36 in. The inlet pipe to each roaster was 18-in. diameter.

Account 8113—Conveyor No. 12.

See account 7405. This conveyor takes the material of the beds from conveyor No. 11 and delivers it to conveyor 13¹ and 13². It is a 20-in. belt, 51 ft. 3 in. from center line of tail pulley to center line of head pulley, with an 8-ft. rise, operating at a speed of 300 ft. per minute, with a capacity of 100 tons per hour. The segregated material account is as follows:

	Cost	Freight	Total
Belt.....	\$209.43	\$14.43	\$223.86
Conveyor material.....	370.91	45.86	416.77
One 5-h.p.m.....			87.04
1 centrifugal switch.....	34.00	2.21	36.21
Lumber, decking and painting material.....			30.97
Spout conv. 11 to conv. 12.....			7.70
Miscellaneous.....			2.50
			\$305.05

Account 8113.1—Conveyors 13¹ and 13².

Conveyors 13¹ and 13² take the product from conveyor 12 running the length of the Roaster Building each delivers the material through a separate automatic tripper to the roaster bins. They are identical. Both are 20-in. belts, running perfectly flat, 109 ft. from center line of head pulley to center line of tail pulley, operating at a speed of 300 ft. per minute, with a capacity of 100 tons per hour. The account for material stands as follows:

	Cost	Freight	Total
Belt.....	\$337.72	\$59.26	\$396.98
Conveyor material.....	913.60	236.45	2,150.05
2 automatic trippers & track.....	1,000.00		
Two 5-h.p. motors.....			174.08
2 centrifugal switches.....	68.00	4.42	72.42
Drive belt.....			30.79
Lumber, decking, paint.....			123.90
Spouts from conveyor 12.....			11.54
Miscellaneous.....			12.57
			\$3,472.33

(See Fig. 91.)

Account 8113.2—Stile over Conveyors 13¹ and 13².

These stiles were made of structural steel, purchased from the Kansas City Structural Steel Co. and erected by the Arizona Copper Co.

Roaster Dust Chamber**Account 8121—Excavation.**

Same as 8101.

Account 8122—Foundation.

Same as 8102.

Account 8123—Steel Structure.

See account 7308.2.

There were 14.8 tons of corrugated iron, 376.4 tons of structural steel used here and 27.63 tons of No. 11 Keystone plate. (See Figs. 103 and 131.)

Account 8123.01—Wire Baffles.

This account covers the cost of material, labor, and repairs entailed in installing 60,480 wire baffles in the roaster dust chamber. The wires with hooks on one end like shepherds crooks were hung 4 in. on centers both ways from chains supported from the lower members of the roof trusses 4 in. apart. The segregation of material is as follows. The wires hung a few inches off the dust chamber bottom and were thus of various lengths. (See Fig. 131.)

1,008 $\frac{1}{2}$ -in. chains 20 ft. 6 in. long with two hooks.....	\$1,451.00
63,964 lb. No. 10 wire (black).....	2,877.82
6,557 lb. No. 10 wire (black).....	361.96
Miscellaneous.....	67.35
	\$4,758.23

(See Fig. 131.)

Account 8123.1—Tile Work.

The sides of the roaster dust chamber and inclined bottom were built of 4-in. hollow tile. This material, labor of masons and their helpers, lumber for scaffolds, carpenter labor, mortar, material and power for hoisting are here included in the cost.

Account 8123.11—Tile Unloading.

This account covers the cost of unloading, wheeling, checking quantities and leveling up ground to receive tile.

Account 8123.2—Painting Outside.

The outside of the tile portion of the chamber, namely, sides and bottom, were given one coat of mineral red and linseed oil. The mortar was scraped from the tile before applying. This account covers the labor and material.

Account 8123.3—Painting Inside.

The tile work on the inside of the dust chamber was given one coat of silicate of soda used as a paint. This account covers the labor and material of this operation.

Reverberatory Plant**Account 8301—Excavation.**

This was the making of a deep surface cut for the building. The material was principally red clay and boulders. In many cases power was used. In general the ground was plowed, scraped with fresnos through a trap into narrow-gauge side-dump cars and hauled, 2,000 ft. by steam locomotive to make a railroad fill.

Account 8301.01—Backfilling.

This covers the cost of backfilling in the reverberatory bottoms and between the reverberatories. The dirt was red clay soil. It was plowed, hauled in wagons, dumped, shoveled into a derrick box and lifted by a locomotive crane over the reverberatory sites and dumped. It was then distributed with wheelbarrows and tamped in 4-in. layers.

Account 8302—Foundation.

This work consisted of long walls averaging 230 ft. long 4 ft. at top and 6 ft. at bottom, and of beams to withstand the reverberatory buck stay pressure, averaging 660 ft. long by 3 ft. by 3 ft. The walls were reinforced with $\frac{3}{4}$ -in. rods, spaces about 4 to 6 in. on centers, while the beams were reinforced with $\frac{3}{4}$ -in. and 1-in. rods about 4 in. center lines one way.

The mixture used was 1 part cement and 5 parts sand and gravel, machine mixed, transported 100 ft. average, with wagons, cars, wheelbarrows and concrete carts as the situation demanded. 100 per cent. of the vertical surface was formed.

Account 8302.1—Concrete Counterweights.

These are used for the cross and header flues. Some are 1 ft. square from 6 ft. to 10 ft. long, cast in wooden forms. Others are circular, cast in steel cylinders. The concrete was mixed in a machine, wheeled 150 ft. and made plain with 1 part cement to 5 sand and gravel.

Account 8303—Steel Structure.

There is in this building 55.31 tons of corrugated iron, and 405.78 tons of structural steel.

Account 8304.—Reverberatories—Brickwork.

This account covers all the brick, mortar material and mason labor used in laying the brick of three reverberatories. The overall dimensions of the furnaces are 104 ft. long by 27 ft. wide and about 10 ft. 6 in. to the crown of the arch. The side walls are 2 ft. 6 in. thick and the arch is 20 in. deep. In the three reverberatories there was used 106,350 red brick, laid in lime mortar and 257,288 various shaped silica brick dipped in silica slimes. (See Figs. 77, 78 and 79).

Account 8304.01—Unloading Brick.

This account covers the cost of preparing the unloading site, building three brick sheds of 84,000 cu. ft. capacity, the unloading of the brick, checking the quantities, and piling separately 31 different shapes.

Account 8304.02—Centering.

This account covers the cost of material, fabrication of one center together with the labor and erecting and tearing it down three times. The arch was made of 2 by 12 centers with 1-in. sheathing tacked on top. The 2 by 12 centers were spaced 18 in. centers, supported on 6 by 8 stringers held up by 4 by 8 posts suitably braced. (See Fig. 130.)

Account 8304.05—Rehandling Brick.

This account covers the transporting at many different times of the silica brick from the sheds to the reverberatories a distance of 500 ft. by one-mule carts.

Account 8304.1—Steel Work.

This account covers the cost of the material and labor of installing the steel buck stays for the reverberatories. Below is a list of material:

Buck stays	276 12-in. 31.5-lb. beams on sides
Buck stays	84 12-in. 31.5-lb. beams on ends
Rails	1,212-ft. 60-lb. rails
Rails	594-ft. 75-lb. rails
Cross stay rds.	78 1½ in. diam. 31 ft. 8 in. long
Longitudinal rds.	27 1½ in. diam. 110 ft.
6 steel supports for longitudinal rods made of 2 to 8-in. angles 11½ lb.	
(See Fig. 78.)	

Account 8304.2—Silica Fill.

This cost is for the silica purchased, crushed in a variety of ways, transported to the furnaces and tamped in place there in layers. It came from the Calumet and Arizona Mining Co., at Douglas. The segregation of the account per ton is as follows:

First cost	Freight	Crushing and placing	Total
\$2.757	\$1.749	\$3.134	\$7.64

Account 8304.3—Hoppers and Chutes.

This account covers the cost of material and the installation of feed hoppers and chutes with their gates and levers to the reverberatories.

Cast-iron hoppers, chutes, weights, levers, bars.....	\$1,216.20
300 ft. ¼-in. steel sash cord, 100 clips, 50 thimbles.....	23.00
102 lb. ½ by 3 flat iron.....	2.45
200 lb. 1½-in. shafting.....	12.09
Miscellaneous.....	9.85

\$1,263.59

(See Figs. 81 and 82.)

Account 8305—Cross and Header Flues Brickwork.

This covers the cost of the tile, brick, mortar, lumber for scaffolds, mason and carpenter labor incident to building the cross and header flues from the reverberatories to the boilers. They are 11 ft. from the ground and 8 ft. by 8 ft. 6 in. in section. The roof is a brick arch held by buck stays. There were red brick, fire brick and 4-in. bottom tile used in the construction. (See Figs. 80, 83, 84 and 85.)

Account 8305.1—Cross and Header Flues Unloading Brick.

This covers the unloading, checking quantities and preparing site for the brick used in the flues.

Account 8305.02—Cross and Header Flues—Centering.

This covers the cost of material and labor for making, installing and wrecking the arch center for the flues of 8305.01. Centers were made for about one-half the length and then moved to the other half.

Account 8305.2—Cross and Header Flues—Painting Brick.

This covers the cost of painting the outside of the flues with one coat of mineral red and linseed oil when the flues were in service. The steel work of the buck stays was given at the same time one coat of graphite paint.

Account 8306—Flues, Boilers to Reverb. Flue—Excavation.

This work covers small pier excavation in red clay. It was picked, shoveled into barrows and transported about 15 ft.

Account 8306.1—Flues, Boilers to Reverb. Flue—Foundation.

This foundation is some small piers of plain concrete mixed by machine, 1 part cement to 6 parts sand and gravel, transported by wheelbarrows 125 ft. About 40 per cent. of the vertical surfaces was formed. Every pier has 2½-in. anchor bolts.

Account 8306.2—Flues, Boilers to Reverb. Flue—Steel Structure.

See account 7308.2.

This covers seven 6-ft. diameter flues of ¼-in. steel with their supports. There were 34.78 tons.

Account 8307—Boiler Building—Excavation.

This account covers the digging of two long deep cuts for retaining walls. Two feet of clay were encountered, followed by sand and gravel and boulders with caliche. The ground was partly blasted, all picked, shoveled into wagons and hauled a distance of 600 ft.

Account 8307.01 Waste Heat Boilers—Excavation.

This work was digging shallow trenches for small foundations, through red clay and small boulders. The ground was picked, shoveled and hauled 600 ft.

Account 8307.02—Oil-fired Boilers—Excavation.

Same as 8307.01.

Account 8307.05—Boiler Feed Pumps—Excavation.

This was a deep square cut involving 659 cu. yd. through red clay and boulders, into sand, gravel and boulders tightened with caliche. It was partly loosened with powder, picked, shoveled and hauled by wagons 400 ft. The lower half was handled twice, once onto scaffolds and the second time out of the pit.

Account 8307.04—Backfill, Back of Boiler Wall.

This was filling behind a long retaining wall. This dirt was adobe, wetted and tamped in 5-in. layers. The dirt was wheeled 60 ft. to place.

Account 8307.1—Boiler Building—Foundations.

This work covered a reinforced wall 240 ft. long, 11 ft. high, 1 ft. at top, 2 ft. at bottom, with pilasters connected at top with horizontal reinforced concrete beams forming the support for waste heat and oil-fired boilers. Three-fourth inch and $\frac{7}{8}$ -in. rods spaced about 6 in. to 8 in. were used together with many foundation bolts. The mixture was 5 sand and gravel to 1 cement, made in a machine and hauled 175 ft. in wagons, thence by wheelbarrow 10 to 15 ft. into place. One hundred per cent. of the vertical surfaces was formed.

Account 8307.11—Waste Heat Boilers—Foundations.

This work covered the reinforced concrete beams noted in 8307.1 required for the waste heat boilers. The other conditions were the same, save 50 per cent. only of the vertical surfaces was formed.

Account 8307.12—Oil-fired Boilers—Foundations.

See account 8307.11.

Account 8307.13—Feed Pumps—Foundation.

This account covered the reinforced cantilever walls for a pit 26 ft. by 26 ft. in plan, 14 ft. high and 1 ft. thick. The rods were $\frac{1}{2}$ in. and $\frac{3}{4}$ in. The mixture was machine mixed 5 sand and gravel to 1 cement, hauled 175 ft. in wagons to place. Seventy-five per cent. of the vertical surfaces was formed.

Account 8307.2—Floor over Slag Track Cut—Floor.

This covered the laying of a 6-in. reinforced concrete floor 30 ft. by 240 ft. over steel I-beams with a mortar finish troweled smooth. The mix was machine made 5 sand and gravel to 1 cement, with 2 to 1 top finish. A $\frac{3}{16}$ -in. woven wire triangular mesh was used and 50 per cent. of the surface was formed. The concrete was wheeled in barrows an average of 175 ft. (See Fig. 86.)

Account 8307.3—Floor around Boilers.

This is a 4,000-sq. ft. plain concrete floor of 4 in. laid in blocks with sand joints and given a 2 to 1 top finish. The concrete was machine mixed, 7 sand and gravel to 1 cement and wheeled in barrows about 175 ft. on the average.

Account 8308—Boiler Building—Steel Structure.

See account 7308.2.

There is in this building 35.03 tons of corrugated iron and 257 tons of structural steel.

Account 8308.5—Platforms and Brackets.

These were structural steel walkways installed after the boilers had been piped and bricked, furnished by the Kansas City Structural Steel Co. and erected by the Arizona Copper Co. construction force. There were 29.5 tons of material. The installation necessitated boring for connections and much removing steam piping.

Account 8309—Waste Heat Boilers, Including all steel.

This account covers the cost of the 7 waste heat boilers with the steel framework of the settings erected. These boilers are class M No. 26 Stirling waste heat boilers, for 180 lb. pressure. They have 7,460 sq. ft. of total heating surface and occupy each a space of 16 ft. by 20 ft. 4 in. by 26 ft. 4½ in. The waste heat from the flue, common to all the reverberatories, enters the front of the boilers at the top. (See Fig. 87.)

Account 8309.01—Waste Heat Boilers—Brickwork.

This account covers all red and fire brick and tile with mortar and lumber for scaffolds, as well as mason and carpenter labor entering into the bricking of the waste heat boilers. The unloading of the brick and handling from the pile to the boiler site are taken care of elsewhere. (See Fig. 87.)

Account 8309.02—Waste Heat Boilers—Unloading Brick.

This covers the cost of preparing the site, unloading and checking all brick used under this account.

Account 8309.03—Waste Heat Boilers—Painting.

When the boiler settings were warm, they were given one coat of mineral red in oil. This account covers the labor and material incident to this operation.

Account 8309.05—Waste Heat Boilers—Rehandling Brick.

This covers the cost of handling brick a distance of 250 ft. from piles to site of waste heat boilers in wheelbarrows.

Account 8309.10—Oil-fired Boilers, Including all Steel.

Same as account 8309. (See Fig. 88.)

Boilers were 3, Class M, No. 14 Stirling for oil firing, having each 4.017 sq. ft. of heating surface. Each boiler occupied a space of 10 ft. by 20 ft. 4 in. by 26 ft. 4½ in. The oil burners are not here included, but 3 stacks are of 48-in. diameter each and 60 ft. high above damper frame, made of No. 10 and No. 8 steel. (See Fig. 88.)

Account 8309.11—Oil-fired Boilers—Brickwork.

Same as account 8309.01. (See Fig. 88.)

Account 8309.12—Oil-fired Boilers—Unloading Brick.

Same as 8309.02.

Account 8309.13—Oil-fired Boilers—Painting.

Same as 8309.03.

Account 8309.15—Oil-fired Boilers—Rehandling Brick.

Same as 8309.05, save distance was about 300 ft.

Account 8310—Superheaters—Waste Heat Boilers.

This account covers the cost of the material and the labor of installing 7 Foster superheaters for class M, No. 26 waste heat Stirling boilers.

7 Foster superheaters.....	\$8,256.40
Miscellaneous.....	32.31
	\$8,288.71

(See Fig. 88.)

Account 8310.10—Superheaters—Oil-fired Boilers.

This account covers the cost of the material and the labor of installing 3 Foster superheaters for class M, No. 14 oil-fired boilers.

3 Foster superheaters.....	\$2,675.03
Miscellaneous.....	81.38
	\$2,756.41

Account 8312—Miscellaneous Piping. Boilers and Reverb. Building.

This account covers the cost of material and installation of miscellaneous piping in the boiler and reverberatory buildings. The sizes are various. It is not valuable for unit costs.

Account 8312.1—Excavation.

This account covers excavation and backfill for a long deep trench. The material met with was red clay filled with boulders and sand and gravel. It was done with picks and shovels, and handled 300 ft. with wheelbarrows and slips. Two hundred feet of the trench were cribbed and lagged 20 ft. high. Much of the dirt had to be handled three times in removing it from the trench.

Account 8312.11—Feed Piping from Heating Plant to Feed Pumps.

This account covers the cost of the pipe, pipe conduit, insulating material and the labor incident to installing them in a trench running from the hot water heating plant to the boiler feed pump house back of the boilers. The conduit was ordinary vitrified 15-in. sewer pipe split in halves. The first half was laid in the trench, the joints cemented, followed by the laying of the 8-in. standard wrought iron pipe. About this the asbestos filler was packed and after each section of the conduit top was laid, the filler was stuffed in over the top of the 8-in. pipe to thoroughly cover it. The material account is segregated as follows:

557 ft. 15-in. J. M. sectional conduit.....	\$2,273.47
577 ft. 8-in. wrought-iron pipe.....	374.49
Asbestos filler and miscellaneous.....	109.83
	\$2,757.79

Account 8312.20—Feed Piping from Pumps to Boilers.

This cost is not valuable for unit purposes. It represents pipe fittings, pipe covering, paint, and the labor of erecting pipe and fittings, covering some of the pipe with insulation, and painting all pipe. The piping was about one steam and two electrical feed pumps at the boilers. It also covers a hot water line the length of the boiler building, a cold water line the same length with connections from each line to each boiler. The two main lines are 6 in. The connections to the boilers are 3 in. The hot water lines are covered throughout. The pipes are of standard strength and the fittings are extra heavy. The labor costs include also the manufacture of all pipe hangers. A further segregation of the material is as follows:

Pipe.....	\$416.39
Fittings.....	2,408.89
Pipe covering.....	137.26
Hangers and miscellaneous.....	78.46
	\$3,041.00

Account 8312.5—Blow-off Piping and Drum.

This cost is not valuable for unit purposes. It represents the cost of the material below and the labor of installing it. The blow-off piping runs about 10 ft. beyond the end of the boiler building and discharges there into a steel drum 4 ft. diameter by 4 ft. high, with an 8-in. diameter pipe riser. From the base of the drum it discharges into a sewer pipe nearby. The drum foundation is included in the concrete cost. A segregated material cost is as follows:

225 ft. 2½-in. standard wrought-iron pipe	}	\$447.93
25 ft. 8-in. standard wrought-iron pipe		
20 ft. 2½-in. extra heavy non-rising stem gate valves		
20 ft. 2½-in. asbestos packed cocks	}	79.84
1 blow-off drum, 4 ft. diam. by 4 ft. high		
		\$527.77

Account 8313—Wiring Electrical Feed Pumps.

This covers the wiring of the two 40-h.p. motors of the feed pumps to the mains. The material was as follows:

2 circuit breakers	\$31.70
Conduit and covering	85.20
Wiring and miscellaneous	60.99
\$177.89	

Account 8313.1—Lighting for Reverb. and Boiler Building.

This represents the material and labor of hanging 104 drop lights in the boiler and reverberatory building.

8 tungsten lamps, 40 watt	}	\$473.19
96 carbon lamps, 16 c.p.		
145 ft. brewery cord		
2,710 ft. 1-in. conduit		
Wire, switches, etc.		

Account 8314—Slag Launderers.

All the material cost and labor installation of these reverberatory slag launderers and two converter slag launderers are here included. The reverberatory slag launderers at the slag end of the furnace consist of a settler and spout. The settler is about 6 ft. by 3 ft. by 2 ft., made of ¾-in. steel plate and 3-in. by 3-in. angles. The spout leading from the settler to the slag cars is of cast iron 6 ft. 6 in. long and 1 in. thick. The converter slag launderers are built of cast iron about 1½-in. thick on the average, in 4-ft. 6-in. sections and are 24 ft. 6 in. long. They are set up aloft in the converter building on a structural steel frame made of 10-in. 15-lb. I's 2½ by 2½ by 1½ angles attached to the framework of the building, and lead into the top of the reverberatory furnaces.

Account 8315—Matte Launderers.

These launderers lead from small settler boxes at the tap holes of the reverberatories along the dirt floors in which they are set to a height of 10 to 12 ft. above the converter building floor where they discharge into properly located matte pots. The small boxes are of ¾-in. plate with 3 by 3 and 2½-in. by 2½-in. angles. The launderers are of cast iron average 2½-in. in thickness. Here is included the cost of the above material together with the labor of installing them.

Account 8316—Six No. 14 Wilgus Oil Systems.

This account covers the cost of 6 Wilgus oil pumps, asbestos covering for portions of these pumps, the labor of installing the pumps, the labor of thoroughly overhauling them, required because of the unsatisfactory condition existing in the leaking steam heating coils and the labor of applying the asbestos covering. The 5½-in. by 3½-in. by 5-in. duplex oil pumps were set directly on the concrete floor in front of the oil-fired boilers.

Account 8317—Two Electrical Feed Pumps.

These pumps located back of the boilers were lowered into the 13-ft. pit onto their foundations and set ready for piping connections. They are two vertical triplex, 8-in. by 10-in. Aldrich, electrical driven pumps each attached with flexible couplings to a 40-h.p. motor. The cost covers the material segregated below and the labor of installing the same:

	Factory	Freight	Clifton
Two 40-h.p. motors.....	\$1,700.00	\$24.44	\$1,724.44
Two vertical triplex pumps.....	2,794.00	547.07	3,859.07
Spare parts for pumps.....	518.00		
Miscellaneous.....			50.46
			\$5,633.97

Account 8317.1—One Steam Feed Pump.

Here is given the labor of installing and the material cost of one 10-in. by 6-in. by 12-in. duplex boiler steam feed pump. This pump is located next to the two electrically driven Aldrich pumps of 8317.

Account 8317.2—Crawls and Chain Blocks in Feed Pump House.

This gives the cost of delivering and hanging in place in the feed water pump house two 2-ton steel plate crawls for lower flange of 8-in. I-beam, one 2-ton duplex chain block for 16-ft. lift together with the material cost as segregated below.

Two 2-ton crawls.....	\$53.74
One 2-ton chain block.....	58.31
	\$112.05

Account 8318—Fettling System.

Here is given the cost of installing fettling tracks and cars over the three reverberatories. A structural steel frame to support an 18-in. gauge car and walkway was erected along each side of each reverberatory, being attached to the steel frame of the building. The cost of this material as given below and the labor of installing the same are here covered:

Three 18-in. gauge bottom dump cars with Hyatt roller bearings.....	\$196.83
3 steel turn plates $\frac{1}{2}$ in. by 72 in. by 138 in.....	93.24
18.25 tons fabricated structural steel and rail.....	1,492.66
2 by 12 lumber for walkways.....	18.48
Power for riveting and miscellaneous.....	141.87
	\$1,943.08

Converter Plant

Account 8401—Excavation.

This was a large slice, similar to side hill work, through red clay and boulders into sand and gravel tightened with caliche. It was shaken up with powder, plowed, fresnoed through a trap into narrow-gauge side dump cars and conveyed 1,000 to 2,000 ft. by a narrow-gauge locomotive.

Account 8402—Foundation.

This was a big wall same as under 8301, with about 30 piers 5 ft. by 6 ft. by 8 ft. deep. The concrete was machine mixed, 7 parts sand and gravel to 1 cement, transported 75 ft. in concrete buggies and wheelbarrows. About 50 per cent. of the vertical surfaces was formed. Each pier had four 2-in. anchor bolts 6 ft. long.

Account 8403—Converter Building—Steel Structure.

(See account 7308.2). There is in this building 94.01 tons of corrugated iron and 689.85 tons of structural steel.

Account 8404—Converter Stands—Excavation.

This excavation was small rectangular cuts in sand and gravel, made with picks and shovels and handled in wagons 900 ft.

Account 8404.1—Converter Stands—Foundation.

This concrete was the same mix as 8402, but 100 per cent. of its vertical surfaces was formed. It was hauled 300 ft. to place in dump wagon and cars.

Account 8405—Converter Stands and Shells.

This account covers the cost of the material noted below, together with the labor required to install the stands, put together the parts of the converters, erect motors, controllers, solenoid brakes and attach blast connections. The unloading of this material from the cars is in account 8411. The shells were 12 ft. in diameter, of the Great Falls type, having each 28 tuyères of $1\frac{1}{2}$ in. diameter, extra heavy pipe.

Three cast-iron converter stands, with 50-h.p. motors, brakes, controller, etc.....	\$9,801.01
Four 12-ft. converter shells.....	12,115.46
Blast connections, valves, etc.....	321.81

\$22,238.28

(See Figs. 126 and 127.)

Account 8405.01—Repairs to No. 2 Stand.

When No. 2 converter shell was let into place by the crane, it was allowed to fall a short distance and cracked the cast-iron stand. The stand was then taken off its foundation and patched with a steel plate.

Account 8405.10—Converter Shells—Brick Lining.

This account covers both the labor and material incident to lining four converter shells and tops with magnesite brick. In bottom of each shell there is an average of 9 in. of silicate of soda and burnt magnesite beneath the brick. Around the sides there are $4\frac{1}{2}$ in. of this material laid in behind the brick. The top is laid with brick only. The material required for one converter is as follows:

28 sacks (286 lb. each) of magnesite cement.

50 sacks (234 lb. each) of burnt magnesite.

5 bbl. (635 lb. each) of silicate of soda.

4,385 magnesite brick of various shapes.

Account 8405.11—Converter Shells—Unloading Brick.

This covers the cost of the shed together with the checking, unloading and piling all magnesite brick, cement, magnesia and sodium silicate.

Account 8406—Cranes.

This covers the cost of two 40-ton Morgan cranes and the labor of installing them on the craneway, and putting together the equipment ready for operation. It does not include the wiring. They were hoisted place on the craneway by the use of two erecting engines. These cranes are of 40-ton capacity, have four motors, span 55 ft. from rail to rail, and are rigged for a 50-ft. lift. Each crane has a 15-ton auxiliary hoist. (See Fig. 90.)

Account 8406.1—Wiring Cranes.

This cost is not valuable as it represents 30 per cent. more labor than should have been spent. The cranes were wired twice because the first time was done improperly. The wiring is largely in conduits. Here too is the cost for the trolley lines from which the cranes take their power.

Account 8407—Clinkering Machines.

These two machines are set 24 ft. above the floor of the converter building on structural steel supports. The steel supports are a part of the converter building and have been costed in that account. The main body of the machine, the mixer, is the frustrum of a cone 13 ft. 6 in. long, whose head end is 5 ft. diameter and whose discharge end is 9 ft. 6 in. diameter. It is made of $\frac{3}{4}$ -in. steel plate, lined with 1-in. cast-iron liners. The whole is mounted on trunnions operated by a 50-h.p. motor. The ladle which feeds the converter slag into the head

end is 60 cu. ft. capacity and is tilted by a screw operated by a 15-h.p. motor.

The feeder which lets siliceous ore into the head end to agglomerate with the slag extends from the silica bins to a pipe discharging into the dropping stream of slag. It is a screw conveyor 4 ft. 9½ in. long. Each machine has a hood connected to a steel flue 2 ft. 6 in. diameter by 36 ft. 8 in. long, leading into the converter dust chamber.

The machinery for two machines enumerated above cost	\$11,872.82
Two 50-h.p. motors as above.....	828.61
Two 15-h.p. motors as above.....	820.16
2 brakes for ladle tipping motor.....	176.51
2 traveling switches for brakes.....	136.44
2 circuit breakers.....	102.80
Miscellaneous.....	44.60
	\$13,981.94

This cost includes the price of the machines and the cost of installing them.

Account 8407.01 Clinkering Machines—Alteration No. 1.

The teeth on the drive gears had to be chipped off and trued up so as to mesh properly.

Account 8407.02—Clinkering Machines—Alteration No. 2.

Account 8407.03—Clinkering Machines—Electrical Alterations.

Account 8407.1—Wiring Clinkering Machines.

This covers the labor and material of wiring the two 50-h.p. motors and two 15-h.p. motors operating the clinkering machines.

Account 8409—Wiring for Converter Control.

Account 8409.1—Lighting.

This covers the labor of installing the material incident to lighting the converter building, as well as the material itself. There were used thirty-four 16-c.p. carbon lamps, twenty-six 250 watt tungstens, 440 ft. of No. 8 and No. 12 weatherproof wire exposed, 880 ft. of No. 12 wire in ¾-in. conduit, 684 ft. of No. 12 wire in 1-in. conduit.

Account 8410—Air Pipe from Power House—Excavation.

This covers the cost of digging a trench through sand, gravel and big boulders for a 24-in. pipe, with pick and shovel and backfilling the same.

Account 8410.1—Air Pipe from Power House—Laying.

This covers the cost of the material segregated below and the labor of installing it. The pipe was placed underground and ran from the power house to connect with all of the converters. It was built to carry air

under 12 lb. pressure of No. 8 U. S. gauge plate riveted, tested for 25 lb. pressure and painted with asphaltum paint. It was made in 30-ft. sections and fastened together with forged steel flanges.

400 ft. 24-in. pipe, 10 in. cast-iron nozzles, tees and ells....	\$1,332.70
22 ft. 10-in. pipe and two 10-in. flanges.....	27.54
Two 24-in. cast-iron gate valves.....	415.25
Three 10-in. cast-iron gate valves.....	138.55
Miscellaneous.....	127.85
	\$2,041.89

(See Fig. 89.)

Account 8411—Ladles, Boats, Bails, Tools, Etc.

This account covers the cost of the material segregated below, the labor of unloading it, the labor of unloading material in account 8405. and the cost of material and manufacture of several converter collar pullers, as well as alterations upon the slag ladles and scrap boats.

	Factory	Freight	Total
One 7-ft. by 7-ft. slag boat complete with chain.....	\$362.00	\$38.00	\$400.00
2 converter scrap boats 9 ft. by 2 ft. 3 $\frac{1}{2}$ in. high by 2 ft. 5 in. wide.....	512.60	28.02	540.62
3 cast-steel slag ladles.....	745.90	283.04	1,028.94
2 cast-steel matte ladles (20 tons capacity).....	1,152.45	519.11	2,371.56
1 bail.....	400.00		
1 bail.....	200.00		
1 pattern.....	100.00		
4 chains and converter lifting devices.....			340.63
1 cast-iron skull breaker.....			124.99
Miscellaneous material for collar, etc.....			125.65
			\$4,932.39

Account 8413—Casting Machines—Excavation.

This covers 2 deep rectangular cuts in sand, gravel and big boulders with pick and shovels. It was loaded into carts and hauled 600 ft.

Account 8414—Casting Machine—Foundation.

The foundation for each machine consisted of a rectangular sump with plain concrete floor enclosed by reinforced concrete retaining walls. The walls were about 6 in. thick, 8 ft. high, reinforced with $\frac{3}{4}$ -in. and $\frac{3}{8}$ -in. rods. The concrete was machine mixed, 5 parts sand and gravel to 1 cement, hauled in cars 150 ft. dumped and handled to site in wheelbarrows 150 ft. 100 per cent. of the vertical concrete surfaces was formed.

Account 8415—Casting Machine—Cost and Erection.

This account covers the cost of all the material composing 2 casting

machines, and all the labor required to erect on their foundations ready to operate. Each machine has a steel cradle to receive a ladle of molten copper. This cradle is controlled from a pulpit and is tipped by the power from a 20-h.p. motor. It is set high enough to pour into a casting spoon of 1½-in. cast iron whose approximate dimensions are 2 ft. wide by 3 ft. 6½-in. long, and from 7 in. to 1 ft. 5½-in. deep. This casting spoon pours into the moulds which are attached to a heavy steel conveyor. The moulds are 39 in number, made of 2½-in. cast iron reinforced with ⅝-in. perforated plate. Their inside dimensions are 2 ft. 4 in. by 1 ft. 6¼-in. by 3¼-in. deep. From the pulpit, by use of power from a 20-h.p. motor, the conveyor with the moulds moves along under a spray of water from needle holes in pipes placed above them until they reach the end of the conveyor where a device in the bottom of the moulds loosens the ingots, allowing them to drop into a tank of water. This bosh is made of ⅝-in. plate, 3 by 3 and 4 by 3 angles. It is 7 ft. wide, 23 ft. 5½-in. long, and varies in depth from 7 ft. 10 in. to 2 ft. 10 in. The copper bars are removed from here by a steel drag conveyor operated by a 11-h.p. motor, controlled from the pulpit. When the bars leave the bosh and fall onto the striking plate they are handled by a radial crane whose moving end travels on a 40-ft. curved I-beam. Along the radial crane beam travels a small air hoist capable of picking up 1 ton. It operates under an air pressure of 16 lb. A jib crane is so located, attached to a building column, that it can handle the moulds for removing and replacing. It has a 3,000 lb. capacity triplex block and 8-in. I-beam trolley. Below is a segregated material list:

2 casting machines.....	\$18,657.89
Two 11-h.p. and four 20-h.p. motors.....	2,933.88
2 jib cranes.....	327.22
2 radial cranes.....	1,167.91
2 traveling switches.....	135.75
2 brakes for ladle tipping motors.....	176.51
4 circuit breakers.....	103.50
Moulds, etc.....	708.55
	\$24,211.21

(See Figs. 128 and 129.)

Account 8415.1—Casting Machine—Repairs.

Account 8416—Loading Platform—Excavation.

Same as 8413, except that it was not hauled away.

Account 8416.1—Loading Platform—Foundation.

This was a low retaining wall of gravity section 300 ft. long, machine mixed, 7 sand and gravel to 1 cement, transported in cars 150 ft. by wagon 350 ft. and by wheelbarrow 70 ft. One hundred per cent. of its vertical surface was formed.

Account 8416.11—Loading Platform—Floor.

This was a plain concrete floor mixed and handled as above, with a $\frac{1}{2}$ -in. finish of 2 sand to 1 cement. There were no joints in the concrete. The finish was troweled smooth.

Account 8416.2—Loading Platform—Backfill.

Behind the 300-ft. wall—8416.1 sand and gravel was backfilled. The material lay 8 to 10 ft. from the wall.

Account 8416.3 Loading Platform—Striking Plates.

Two striking plates, one at each casting machine are placed so that the copper ingots discharged from the casting machine elevator fall directly upon them. They were made by setting 4 by 4 by $4\frac{1}{2}$ -in. wood blocks dipped in hot tar and placed on end upon a concrete base. Over the blocks a steel plate 6 ft. $\frac{1}{2}$ in. by 9 ft. 10 in. by $\frac{1}{2}$ in. was laid and secured by 16 $\frac{3}{4}$ by $2\frac{1}{2}$ -in. bolts, grasped by cast-iron fasteners set in concrete below.

283 ft. b.m. lumber.....	\$8.59
2 steel plates.....	99.19
32 cast-iron fasteners and bolts.....	18.91
	\$126.69

Account 8417—Hoods and Smoke Boxes.

This account covers all the material of the converter hoods, smoke boxes, flues leading to converter dust chamber, together with the labor of erecting them. It likewise includes removing the stacks 4 ft. in diameter, making new ones 5 ft. in diameter and erecting them together with change required to put large doors in the back of the boxes. The smoke boxes, of which there are three, are made of $\frac{3}{8}$ -in. plate, and 4 by 4 by $\frac{1}{2}$ angles. They are 16 ft. high and about 9 ft. in diameter. The hoods, of which there are three, are made from $\frac{1}{2}$ -in. plate and 4 by 4 by $\frac{3}{8}$ angles. They hang on the front of the smoke boxes and direct the gases into the flues. The original stacks, of which there were three connecting the smoke boxes and the converter dust chamber, were 4 ft. diameter and 26 ft. long, made of $1\frac{3}{8}$ -in. plate. They were replaced by similar ones 5 ft. in diameter. (See Fig. 92.)

Account 8417.1—Hood to Protect Converter Operator.

Only one of these was made. Three-sixteenth inch plate was used. The dimensions are 7 ft. 2 in. by 7 ft. 2 in. by 7 ft. 8 in. high, one end is open. The account covers material used, fabrication and erection.

Account 8418—Spouts, Gates and Hoppers at Silica Ore Bins.

This account covers the material cost of the gates with operating devices, the 10-in. pipe chutes and the labor of erecting same, together

with the labor of erecting the hoppers. The hoppers were furnished by the Kansas City Structural Steel Co., and are costed with the building. The hoppers are situated below the silica bins, above the converters and by a spring device and pointer indicate to an operator on the ground when they have been filled to the desired amount. The gates allow the material to flow through a 10 in. pipe chute directly into each converter. These chutes can be turned aside from the converter mouth by a chain, wheel and gear so as not to interfere when out of use. (See Fig. 93.)

Account 8419.1—10-ton Bullion Scales—Excavation.

The excavation consisted of small cuts made in sand and gravel with pick and shovel and cast to one side.

Account 8419.2—10-ton Bullion Scales—Foundations.

This concrete was cast plain in low 8-in. thick walls about a pit 4 ft. by 6 ft. in plan. The mix was machine made, 6 sand and gravel to 1 cement, and transported a distance of 1,900 ft. in wagons. Seventy-five per cent. of the walls' vertical surface was formed.

Account 8419.3—10-ton Bullion Scales—Cost and Erection.

This represents the cost of the scales and the labor of installing them. The scales were pit pattern, 10-ton copper bullion class, with type registering beam weighing to 1 lb. They came complete with all necessary structural steel framework and cast-iron platform plate.

Account 8419.4—10-ton Bullion Scales—Scale House.

This is a shed roof building without sides about 16 ft. by 20 ft. The roof is of 1-in. sheathing, covered with composition roofing. It was painted 2 coats of oil and lead.

Account 8425—Conveyor No. 15.

(See account 7405.)

Conveyor No. 15 is a 20-in. belt, making a conveyor 165 ft. long, running perfectly flat, operating at a speed of 300 ft. per minute, capable of handling 100 tons per hour. It receives material from conveyor 14 and delivers it to the silica bins of the conveyor building through an automatic tripper. The account is segregated as follows:

Belt.....	\$664.19
Robins' Material.....	1,231.91
Centrifugal switch.....	36.20
Lumber (decking, etc.).....	67.05
Spout from No. 14 to No. 15.....	31.21
29 ft. 6 in. of 5-in. d.l. drive belt.....	20.82
7½-h.p. motor.....	138.75
Miscellaneous.....	61.34

\$2,251.47

(See Fig. 91.)

Account 8426.1—Wet Pan—Excavation.**Account 8426.2—Wet Pan—Foundation.**

This concrete was hand mixed. Owing to some conditions not satisfactorily ascertained the concrete did not set. This necessitated its being put in twice. The yardage is that of one installation and the cost two. The mix is 7 to 1.

Account 8426.3—Wet Pan—Cost and Erection.

This mill was installed to furnish "mud" for the converters and reverberatories. The account covers the material segregated below and the labor of installing the same.

One 5-ft. wet pan; size of mullers 36 in. by 4½-in.; pulley 34 in. by 10-in.; 4-arm type friction clutch	\$634.34
One 15-h.p. motor, 860 r.p.m. squirrel-cage with starting compensator	248.45
One 18 by 10 solid hub cast-iron pulley	15.47
One 38 by 7 solid hub cast-iron pulley	24.71
Two 2 ½ by 24-in. drop hangers	23.48
11 ft. by 2 in. 2 ½ shafting, collars, etc.	13.10
30 ft. 6-in. double leather belting	25.39
35 ft. 9-in. double leather belting	44.42
Miscellaneous	20.74

\$1,050.10

Account 8426.4—Wet Pan—Bins and Spout.

This bin with spout was made in the smelter shops, using ½-in. steel plate. It has a capacity of 260 cu. ft. The account covers the material used, labor of fabrication and erection.

Converter Dust Chamber**Account 8421—Excavation.**

This account covers the making with pick and shovel of small cut for a retaining wall, and digging a number of small pier holes. The material was red clay and stones, running into sand and gravel which was loaded into carts and hauled 600 ft.

Account 8422—Foundation.

This concrete was cast as piers about 4 ft. by 4 ft. by 5 ft. about 45 per cent. of whose vertical surface was formed. It was mixed in a machine, in the proportions of 7 sand and gravel to 1 cement, transported by cars and wheelbarrows 200 ft. The pier tops were finished to a perfect elevation to receive structural steel columns.

Account 8423—Steel Structure.

(See account 7308.2.) This structure contained 228.18 tons of structural steel and 10.12 tons of Keystone plate roofing. (See Fig. 94.)

Account 8423.01—Wire Baffles.

(See account 8132.01, for description.) This account covers the cost of the material below and the labor incident to its erection.

14,365 lb. No. 10 steel wire baffles.....	\$670.34
8,500 lb. $\frac{3}{4}$ -in. steel chains, 2-in. links.....	430.91
Miscellaneous.....	0.70
	\$1,101.95

(See Figs. 94 and 95)

Account 8423.1—Tile Work.

This is identical with 8123.1.

Account 8423.11—Unloading Tile.

This is identical with 8123.11.

Account 8424—Iron Doors and Frames.

This covers the cost of the cast-iron doors, etc., set in the tile work of the converter dust chamber. The labor represents hauling the same to the site. The labor of setting is included with the tile work.

5 cast-iron peep doors and frames, doors 4 ft. 6 in. by 2 ft. 6 in..... \$158.93

Account 8428—Smoke Box Track.

This is a track back of the smoke boxes for the converters. The material is second hand, picked up from construction equipment. The account is of no value.

Conveying System**Account 8501—Excavation.**

This covers excavation made at various times for piers and trenches for walls to support the conveying system structures. The ground was mostly red clay and boulders, sometimes sand and gravel. The excavating was done with pick and shovel and the material cast to the side of the cuts.

Account 8502—Foundation.

This account covers plain concrete cast in a great many piers, and reinforced concrete cast in a shape to make two long tunnels through which conveyors 11 and 14 rise from below conveyors 10¹ and 10² located under the bunker bins. The tunnels are 6 ft. by 6 ft. with 12-in. walls, reinforced with $\frac{1}{2}$ -in. and $\frac{3}{4}$ -in. rods, spaced 6 in. About 80 per cent. of the vertical surfaces was formed. All concrete was machine mixed in different proportions and transported variously to the many different situations.

Account 8503—Steel Structure.

(See account 7308.2.) There were here used 30.94 tons of corrugated iron, and 180.79 tons of structural steel. These structures are elevated steel conveyor ways.

Account 8504—Woodwork.

This account represents the labor and material of flooring the steel conveyor ways for conveyors 3, 4, 5, 6, 11 and 14. The lumber used was 2 by 12 S1S2E No. 1 merchantable Oregon pine. On No. 14 the 2 by 12's were rabbeted. Considerable cutting was done to frame about conveyor steel framesupports. This cost includes also attaching nailing strips to the steel work to which the flooring was nailed. (See Fig. 96.)

Account 8504.1—Floor Battens.

This account covers labor and material incident to nailing battens beneath the floor boards of conveyors 3, 4, 5, 6 and 11. The lumber here used was not rabbeted.

Account 8505—Conveyors No. 3, 4, 5, 6, 11 and 14.

(See account 7405.) Conveyor No. 3 has a 20-in. belt, making a conveyor 182 ft. 5½ in. long, rising 46 ft., operating at a speed of 250 ft. per minute, with a capacity of 150 tons per hour. It conveys concentrates from No. 2 belt into the sampling mill.

Conveyor No. 4 has a 20-in. belt, making a conveyor 220 ft. 9 in. long, rising 64 ft., operating at a speed of 250 ft. per minute, having a capacity of 100 tons per hour. It takes crushed ore from the crushing plant to the top of the sampling mill.

Conveyor No. 5 has a 20-in. belt, making a conveyor 127 ft. long, having a rise of 26 ft. 4 in., operating at a speed of 250 ft. per minute, with a capacity of 150 tons per hour. It carries the fines from the sample mill on their way to the beds.

Conveyor No. 6 has a 20-in. belt, making a conveyor 113 ft. 8 in. long, having a rise of 25 ft. 6 in., operating at a speed of 250 ft. per minute with a capacity of 100 tons of ore per hour. It carries material from the sample mill on its way to the bunker bins.

Conveyor No. 11 has a 20-in. belt, making a conveyor 369 ft. 8½ in. long, having an 87-ft. rise, operating at a speed of 300 ft. per minute with a capacity of 100 tons per hour. It conveys the product from conveyor 10¹ to conveyor 12 at the Roasters.

Conveyor No. 14 has a 20-in. belt, making a conveyor 271 ft. 5 in. long, having a 71-ft. rise, operating at a speed of 300 ft. per minute with a capacity of 100 tons per hour. It takes ore from conveyor 10² beneath the bins to conveyor 15.

The material for these conveyors somewhat segregated is as follows:

	Factory	Freight	Clifton
Belt.....	\$5,351.03	\$365.88	\$5,716.91
Robins material.....	4,290.86	530.49	4,821.35
Extra pulley.....			38.00
4 centrifugal switches.....	136.00	8.81	144.81
One 10, one 15, and four-20-h.p. motors.....			1,309.68
5 overload releases.....	65.00	6.25	71.25
Miscellaneous, decking, etc.....			403.62
			\$12,505.62

Account 8505.1—Chutes.

This cost is of no value.

Account 8505.2—Guides.

The belts in the conveying system could not be made to run true on the troughing idlers. To overcome their riding out of position long boards were fixed at the sides of the belts to guide and keep them in position. These boards were picked up about the plant and the cost represents only the labor of installing them.

Account 8505.3—Weightometer.

This account covers the cost and labor of installing a Merrick weightometer on conveyor 11. The weightometer is installed on a 20-in. inclined conveyor belt with a speed of 300 ft. per minute, whose angle of inclination is 13 degrees 28 minutes and whose troughing idlers are 4 ft. on center lines. The belt has a normal capacity of 100 tons per hour.

Account 8506—Lighting.

This represents installing the following lights:

33 drops
1,285 ft., No. 12 weatherproof wire
120 ft. of conduit

Chimney**Account 8601—Excavation.**

This was a deep hexagonal cut made through clay, caliche and well into sand and gravel containing big boulders. The material was loosened with picks, slipped out with fresnos, dumped through a trap into carts and hauled 2,700 ft.

Account 8602—Foundation.

This was a very large block of concrete cast in a hexagonal shape 20 ft. deep and 50 ft. inside least diameter. In the bottom of the block 3 layers of 1-in. rods laid 1 ft. on centers were placed. The mixture was machine made, 8 parts sand and gravel to 1 cement, using lots

of large rock. About 40 per cent. of the vertical surface was formed. The concrete was transported in cars 100 ft.

Account 8603—Brickwork.

The stack was contracted erected by the Alphons Custodis Chimney Construction Co. It is 300 ft. high, 26 ft. 8 in. inside diameter at the base and 22 ft. at the top. The average thickness of the walls is about 24½ in. Every 25 ft. inside the stack is corbelled out to hold the lining of radial perforated fire brick, laid in acid-proof mortar. The base of the stack is of red brick and the round portion is of perforated radial blocks. The outside upper 75 ft. of the stack were pointed with acid-proof mortar. There was used in the construction:

138,000 lb. lime	652 tons wire cut brick
290 lb. cement	56 tons wedge brick
1,638 tons radial brick	100 bbl. acid-proof mortar

The cost here given includes constant inspection by the Arizona Copper Co. organization. (See Fig. 97.)

Reverberatory Flue

Account 8611—Excavation.

This covers the excavating of some long deep trenches for footings and a large amount of back filling. It was done in red clay and gravel with picks and shovels. The back filling was wheeled 25 ft. to place and tamped in 5-in. layers.

Account 8612—Foundation.

This concrete was cast in 2 long reinforced concrete cantilever type retaining walls. The walls were 12 in. at the top, 14 in. at bottom, and 5 ft. high. One-half-inch and ¾-in. rods, spaced 6 in. centers, were used. The mixture was machine mixed in the proportion of 5 sand and gravel to 1 cement, transported in wagons, wheelbarrows and concrete carts 250 ft. to place. Ninety-five per cent. of the vertical surface of the concrete was formed.

Account 8613—Brickwork.

This account is similar to others of the same nature, including cost of tile, mortar, scaffolds, and the labor of masons, their helpers, and carpenters. (See Figs. 98 and 100.)

Account 8613.01—Unloading Brick.

This covers the cost of preparing site, unloading tile, and the checking of same.

Account 8614—Steel Structure.

(See account 7308.2.) There were 32 tons of structural steel used here and 9.61 tons of Keystone plate roofing. (See Fig. 99.)

Account 8614.1—Clean Out Doors.

This covers the cost of labor of altering and material in the clean out doors and frames for this flue

18 cast-iron frames and steel plate doors, $\frac{1}{4}$ in. by $16\frac{1}{2}$ in. by 2 ft. 2 $\frac{1}{2}$ in.	\$128.27
$\frac{1}{4}$ -in. sheet steel and miscellaneous.....	25.34
	\$153.61

Account 8614.2—Caulking Roof.

This account covers the labor and material of making as nearly air-tight as was possible the roof to this flue. Asbestos wicking was caulked into all the bad joints.

Converter Flue**Account 8621—Excavation.**

This was a small amount of excavation for a number of piers through red clay with boulders and sand and gravel. It was done with pick and shovel, the dirt being cast to the sides of the holes.

Account 8622—Foundation.

These foundations were 14 plain concrete piers about 4 ft. 6 in. by 4 ft. 6 in. by 5 ft. The concrete was machine mixed, about 7 parts sand and gravel to 1 cement, and transported 200 ft. in wheelbarrows and concrete carts to place. Seventy-five per cent. of the vertical surfaces was formed.

Account 8624—Steel Structure.

(See account 7308.2.) 81.99 tons of structural steel were used here.
(See Fig. 101.)

Roaster Dust Chamber Flue**Account 8626—Excavation.**

This excavation covers the cuts for a number of piers through red clay containing boulders, made with pick and shovel and thrown to one side of the excavation.

Account 8627—Foundation.

This concrete was cast in 12 piers about 4 ft. 6 in. by 4 ft. 6 in. by 5 ft. It was plain concrete, machine mixed in proportions of 7 sand and gravel to 1 cement, and was transported to place 200 ft. in cement cars and wheelbarrows. Seventy-five per cent. of the vertical surface was formed.

Account 8628—Brickwork.

This is the same as 8123.10. (See Figs. 99, 100, 102, and 104.)

Account 8628.01—Unloading Tile.

This is the same as 8123.11.

Account 8629—Steel Structure.

There were 85.21 tons of structural steel used here and 9.25 tons of Keystone plate roofing. The flue is 6 ft. 6 in. by 12 ft. in cross section and connects the roaster dust chamber with the stacks. (See Fig. 102.)

Boiler and Blacksmith Shop**Account 8701—Excavation.**

This excavation involved making a 6-ft. slice to get the proper grade for the building site, together with piers and small wall excavation. It was plowed and slipped away in fresnos 400 ft.

Account 8702—Foundations.

These foundations were the small walls and piers for the brick and steel column supports. The concrete was plain, hand mixed in the proportions of 6 sand and gravel to 1 cement, and handled 100 ft. in wheelbarrows to the forms. Fifty per cent. of the vertical surface was formed. This was the first concrete cast at the smelter.

Account 8703—Steel Structure.

There were 32.72 tons of structural steel used in the framework of the building. (See Fig. 105.)

Account 8703.1—Doors, Windows and Frames.

This account covers the purchase price of all doors, windows, their frames, lintels and glass. It also covers the labor of installing the steel lintels which run from building column to building column; the erection of the steel door and window frames; the erection of the steel sash and doors; and the glazing of these doors and windows. After the lintels had been framed in, the tile work brought up to sill base and the sill set, the frames were put in place, bolted to the lintels and tied by rods back to the building columns. When the frames had been entirely bricked in, the steel sash were bolted in place and later glazed. A segregated material list is as follows:

Thirteen 11 ft. 7 in. by 12 ft. $\frac{1}{4}$ in. steel sash 63 lights, 2 mullions, with 3 to 6 light ventilators, not glazed.

One 10 ft. 3 in. by 12 ft. $\frac{1}{4}$ in. steel sash 56 lights, 1 mullion, no ventilators not glazed.

Two 10 ft. 3 in. by 12 ft. $\frac{1}{4}$ in. steel sash, 48 lights, 1 mullion, no ventilators, not glazed.

One 4 ft. by 9 ft. steel sliding door, with six 14 in. by 20 in. lights, not glazed, lower panels steel plate.

One 8 ft. by 9 ft. steel sliding door, with eighteen 14 in. by 20 in. lights, not glazed, lower panels steel plate.

One 14 ft. by 20 ft. Kinner steel rolling door.

One 10 ft. by 10 ft. Kinner steel rolling door.

Eleven 14-ft. 10-in. lintels built up of 8-in. channels.

Two 13-ft. 4-in. lintels built up of 8-in. channels.

Two 11-ft. 6-in. lintels built up of 8-in. channels.

One 10-ft. 4-in. lintels built up of 8-in. channels.

One 11-ft. 6-in. lintels built up of 8-in. channels.

850 lights 14 in. by 20 in., $\frac{1}{4}$ in. factory ribbed glass,

164 lights 13 $\frac{1}{2}$ in. by 19 $\frac{1}{2}$ in. factory ribbed glass.

82 lights 14 in. by 19 $\frac{1}{2}$ in. factory ribbed glass.

44 lights 13 $\frac{1}{2}$ in. by 20 in. factory ribbed glass.

Steel windows and door frames for above made of two 3 $\frac{1}{2}$ by 2 $\frac{1}{2}$ by $\frac{1}{4}$ angles.

(See Fig. 106.)

Account 8703.11—Concrete Sills.

This account covers the labor and material used to make the following list of concrete sills. The sills were made 3 parts sand and gravel to 1 cement, cast in collapsible moulds and later finished. Three $\frac{5}{8}$ -in. rods are used in each sill.

11 sills, 8 $\frac{1}{2}$ in. by 10 in., 14 ft. 10 in. long.

1 sill, 8 $\frac{1}{2}$ in. by 10 in., 11 ft. 6 in. long.

1 sill, 8 $\frac{1}{2}$ in. by 10 in., 8 ft. 6 in. long.

2 sills, 8 $\frac{1}{2}$ in. by 10 in., 12 ft. 2 in. long.

(See Fig. 21.)

Account 8703.2—Tile Walls.

This cost includes the cost of tile, mortar and scaffolds, together with the mason and carpenter labor used to build the walls. The walls were non-bearing 8 in. thick, built of hollow tile, laid in between the steel building columns. The mortar used was 1 cement, 1 lime and 1 sand.

Account 8703.21—Unloading Tile.

This covers the cost of preparing site, unloading, and checking quantity of tile.

Account 8703.22—Coping.

This covers the cost of labor and material incident to coping the walls at the top, beneath the roof. A two by four was bolted to the top course of tile and another to the underside of the roof. These were lathed across with metal lath and plastered with cement mortar. (See Fig. 17.)

Account 8703.30—Roof.

This account covers the cost of the material and labor incident to roofing the boiler and blacksmith shop. Oregon pine sheathing, 2 by 8, surfaced, tongued and grooved, was nailed to strips bolted to the purlins. Over this 3-ply asbestos roofing paper was laid. (See Fig. 17.)

Account 8703.31—Ventilators.

This covers the cost of labor and material incident to installing three 48-in. Burt ventilators on the peak of the boiler and blacksmith shop roof.

The ventilators were skidded up onto the roof with hand tackle along a runway, bolted to the purlins and flashed. (See Fig. 18.)

Account 8703.4—Dirt Floor.

This account covers the labor incident to bringing the dirt floor of this building to the required grade. The dirt was wheeled in and tamped in 3-in. layers.

Account 8703.5—Benches.

This account covers the labor and material of making from time to time benches, racks and the like used in this shop.

Account 8703.6—Painting.

This covers the cost of painting all the steel sash one coat of "turkey red." and the woodwork, namely, the under side of the roof two coats of white lead and linseed oil, cream color.

Account 8704—Crane.

This covers the purchase of the crane listed below, the labor of overhauling and erecting it.

One 3-ton hand power traveling crane, chain block transfer type 18-ft. span, complete with roller bushed geared trolley and provided with 3-ton triplex chain block for 13 ft. lift.	\$378.35
Miscellaneous.	60.06
	\$438.41

Account 8705—Tools.

This account covers the purchase price of the tools enumerated below and the labor required to install them.

	Factory	Freight	Clifton
1 No. 2 punch and shear, Hilles & Jones.	\$1,530.00	\$435.00	\$1,965.00
1 No. 0 bending rolls.	580.00	75.00	655.00
One 1,100-lb. steam hammer, Niles-Bement-Pond Co.			
1 blower, size 5, type D, American Blower Co. . .	1,015.00	408.00	1,423.00
One 5-h.p. 440-volt, 3-phase, 60-cycle 1,720 r.p.m. motor.	160.00	19.90	179.90
1 No. 5 swage block.			35.08
1 Peter Wright anvil, weight 497 lb.			70.57
10 in. galv. iron pipe and connections.			106.63
3 sheets steel, $\frac{1}{4}$ in. by 48 in. by 120 in.			16.02
One 2-in. heading, upsetting and forging machine, Acme Machinery Co.	2,790.00	440.70	3,230.70
1 sisco anvil, 407 lb.			46.60
1 Hay Budden anvil, 420 lb.			48.10
40 ft. of 6-in. I-beam.			12.62
Castings.			41.00
Miscellaneous.			29.14
			\$7,859.36

Account 8706—Shafting, Pulleys, Belting.

This account covers the purchase price of the list of material below and the labor of installing the same, and the necessary wooden bridge trees.

33 ft. of 2 $\frac{1}{4}$ -in. and 18 ft. of 2 $\frac{1}{2}$ -in. shafting.

5 pulleys, varying from 26 in. to 52 in. with bearings and hangers.

1 length of 8-in. double leather belt, 104 ft. long.

1 length of 6-in. double leather belt, 140 ft. long.

(See Fig. 19.)

Account 8707—Motor.

This account covers the purchase price of the material below and the labor of installing it. This motor furnished the power for the boiler and blacksmith shops.

One 20-h.p. 440-volt, 3-phase, 60-cycle, 850-r.p.m. motor.

Account 8708—Lighting.

This account covers the cost of the material below and the labor of its installation.

14 carbon lamps, 16 c.p.

260 ft. brewery cord

3 tungstens, 250 watt

300 ft. No. 12 wire

100 ft. conduit.

Machine and Carpenter Shop**Account 8715—Excavation.**

Same as account 8701.

Account 8716—Foundation.

Same as account 8702.

Account 8717—Steel Structure.

This building is the same as 8703 account. There were used here 38.23 tons of structural steel. (See Fig. 105.)

Account 8717.1—Doors, Windows and Frames.

This account is the same as 8703.10, with the following list of material:

Thirteen 11 ft. 7 in. by 12 ft. $\frac{1}{4}$ in. steel sash, 63 lights, 2 mullions, with 3 to 6 light ventilators, unglazed.

Two 10 ft. 3 in. by 12 ft. $\frac{1}{4}$ in. steel sash, 56 lights, 1 mullion, no ventilators, unglazed.

Two 10 ft. 3 in. by 12 ft. $\frac{1}{4}$ in. steel sash, 48 lights, 1 mullion, no ventilators, unglazed.

Two 4 ft. by 9 ft. steel sliding doors, with six 14 in. by 20 in. lights, unglazed, lower panels steel.

One 11 ft. by 12 ft. steel sliding door, with forty 14 in. by 20 in. lights, unglazed, lower panels steel.

Two 14 ft. by 20 ft. Kinner steel rolling doors.

Fourteen 14-ft. 10-in. lintels, made of 8-in. channels.

Six 11-ft. 6-in. lintels, made of 8-in. channels.

Seven hundred forty 14 in. by 20 in., $\frac{1}{2}$ in. thick factory ribbed glass window panes.

One hundred sixty-four 13 $\frac{1}{2}$ in. by 19 $\frac{1}{2}$ in., $\frac{1}{2}$ in. thick factory ribbed glass window panes.

Eighty-two 14 in. by 19 $\frac{1}{2}$ in., $\frac{1}{2}$ in. thick factory ribbed glass window panes.

Twenty 15 in. by 20 in., $\frac{1}{2}$ in. thick factory ribbed glass window panes.

Steel window and door frames for the above list. (See Fig. 106.)

Account 8717.11—Concrete Sills.

Same as account 8703.11, but the following product:

Thirteen 8 $\frac{1}{2}$ in. by 10 in., 14 ft. long sills.

Two 8 $\frac{1}{2}$ in. by 10 in., 11 ft. 6 in. long sills.

Two 8 $\frac{1}{2}$ in. by 10 in., 6 ft. 6 in. long sills.

(See Fig. 21.)

Account 8717.20—Tile Walls.

Same as account 8703.20.

Account 8717.21—Unloading Tile.

Same as account 8703.21.

Account 8717.22—Wall Coping.

Same as account 8703.22.

Account 8717.30—Roof.

Same as account 8703.30. This roof contains 77.21 squares, equal to 14,543 b.m. 2 by 8 lumber.

Account 8717.31—Ventilators.

Same as account 8703.31. Three 48-in. Burt ventilators used here.

Account 8717.40—Floor.

This account covers the cost of the material and labor required to lay this floor. Six inch by eight inch stringers were laid 2 ft. 6 in. on centers with earth tamped in between them. On the stringers No. 3 grade, 3 in. by 12 in. white cedar planking of various lengths was spiked down. (See Fig. 24.)

Account 8717.50—Benches.

Same as account 8703.50.

Account 8717.60—Painting.

Same as account 8703.60.

Account 8718—Crane.

This is the same as account 8704, with the exception that the crane here used is of 5 ton capacity.

Account 8719—Tools.

This account covers the purchase price of all the material listed below and the labor cost of installing it:

	Factory	Freight	Clifton
1 Prentiss machine bench vise, No. 2..			\$20.15
1 machine bench vise, No. 21.....			20.16
1 machine bench vise, No. 22.....			28.85
1 machine pipe vise, No. 2A.....			2.38
1 machine pipe vise, No. 4A.....			7.77
1 stationary bench vise, No. 56.....			20.72
40 ft. of 1½-in. pipe.....			2.97
1 No. 48 power grindstone.....			56.62
2 emery wheels.....			8.90
1 emery wheel grinder.....			17.00
1 No. 40 special turning machine.....			36.22
1 set faces for wiring machine.....			5.56
1 gauge.....			2.35
1 burr machine and stand.....			9.92
1 No. 17 S. P. crimper and stand.....			10.77
1 No. 3 beading machine.....			26.79
1 No. 0236 squaring shears.....			180.86
1 stake-holder and stakes.....			42.15
1 rivet set.....			2.65
1 No. 101 tinner's rule.....			2.73
1 power hack saw No. 3.....			29.63
1 radial drill press, 42 in.....			752.20
Miscellaneous.....			21.92
1 50-in. cornice brake.....			155.96
1 16-in. rip saw.....			4.30
Castings.....			10.10
1 No. 1 drill chuck.....			5.61
1 No. 2½ drill chuck.....			7.02
72 hack saw blades.....			5.55
1 surfer, 20 in. by 6 in.....	\$180.00	\$26.70	206.70
1 No. 50 hand saw.....	175.00	27.45	202.45
1 lathe, 14 in. by 8 ft.....	563.75	81.40	645.15
1 lathe, McCabe patented double spindle.	2,111.00	277.15	2,388.15
1 Crescent saw table.....	168.75	51.34	220.09
One 20-in. Rockford shaper.....	425.00	175.07	600.07
One 2-in. bolt cutter.....	355.00	47.10	402.10
1 Crane pipe machine 2 in.....	192.00	16.56	208.56
1 Crane pipe machine 4 in.....	480.00	44.10	524.10
1 Crane pipe machine 12 in.....	1,500.00	163.59	1,663.59
Small tools, miscellaneous equipment.....			394.36
			\$8,953.13

Account 8720—Shafting, Pulleys and Belting.

This account covers the purchase price of the material below and its cost of installation:

- 1 pc. 30 ft., 2 $\frac{1}{4}$ in. diameter shafting.
- 1 pc. 60 ft., 2 $\frac{1}{4}$ in. diameter shafting.
- 1 pc. 18 ft., 2 $\frac{1}{4}$ in. diameter shafting.
- 1 pc. 18 ft., 2 $\frac{1}{8}$ in. diameter shafting.
- 1 pc. 22 ft., 2 $\frac{1}{8}$ in. diameter shafting.
- 1 pc. 10 ft., 2 $\frac{1}{8}$ in. diameter shafting.
- 1 pc. 4 ft., 2 $\frac{1}{8}$ in. diameter shafting.

Many pulleys ranging from 10 in. to 68 in. diameter, with necessary hangers, collars, boxes, etc. (See Fig. 19.)

Account 8721—Motor.

This account covers the purchase price and cost of installing the following motor.

One 40-h.p., 440-volt, 3-phase, 60-cycle, 850-r.p.m. motor.

(See Fig. 20.)

Account 8722—Lighting.

This covers the cost of the following material and the labor of installing it.

17 carbon lamps	16 c.p.
3 tungstens	250 watt
240 ft. brewery cord	
100 ft. conduit	
360 ft. No. 12 weatherproof wire.	

(See Fig. 23.)

General Office

Account 8804—Furniture and Fixtures.

This account covers the furniture and fixtures purchased for the smelter office, which to date has not been built. The furniture is in use in the temporary offices.

Warehouse

Account 8810—Excavation.

This was the same as 8701.

Account 8811—Foundation.

This was the same as 8702, save that the walls were higher.

Account 8812—Steel Structure.

This building is the same type as the boiler and machine shops, save a corrugated iron roof was used in place of a wood and paper covering. There were 26.5 tons of structural steel used and 13.26 tons of corrugated iron. (See Fig. 26.)

Account 8812.1—Doors, Windows and Frames.

The doors for the warehouse were similar to the boiler and machine shops. The lintels over the windows and doors were the same as in

the shops. The small doors, all windows and frames were wood. This account covers the cost of the door and window material listed below and the labor of installing the same.

51 windows, 3 ft. 9 $\frac{1}{2}$ in. by 7 ft. 8 $\frac{1}{2}$ in. by 1 $\frac{1}{2}$ in. These were grouped 16 in triple frames, all glazed.....	\$305.99
16 wood frames for 48 of above windows	170.32
Lumber for 3 window frames, all door frames and all hardware....	83.78
1 O. G. 1 light glazed door, 3 ft. 6 in. by 7 ft. 1 3/8 in.....	7.91
Two 9 ft. 10 in. by 7 ft. 6 $\frac{1}{2}$ in. Kinner rolling doors	157.30
Steel lintels.....	331.01
(See Fig. 27.)	
	\$1,056.31

Account 8812.11—Concrete Sills.

See account 8703.11. Sills were made here for frames of account 8812.10. (See Fig. 22.)

Account 8812.2—Tile Walls.

Same as for account 8703.11.

Account 8812.21—Unloading Tile.

Same as for account 8703.21. (See Fig. 26.)

Account 8812.22—Coping.

Same as for account 8703.22. (See Fig. 28.)

Account 8812.3—Painting Roof.

This covers the labor and material of painting underside of corrugated iron roof 2 coats of lead and linseed oil, cream color.

Account 8812.31—Ventilators.

Same as for account 8703.31. These three ventilators were 48 in. diameter with round base. (See Fig. 25.)

Account 8812.40—Floor Excavation.

This entailed cutting down the front in the warehouse 6 to 8 in. and backfilling in places.

Account 8812.41—Floor Concrete.

This concrete floor was cast in large 6 ft. to 8 ft. blocks, 4 in. thick, with sand joints between blocks. The concrete was hand mixed in the proportions of 6 sand and gravel to 1 cement. It was transported in wheelbarrows 100 ft. The top finish, $\frac{3}{4}$ in. thick, was 2 sand to 1 cement. This top was troweled smooth.

Account 8812.50—Lighting.

This account covers the cost of the following material and the labor of installation.

26 carbon lamps, 16 c.p. 365 ft. brewery cord
170 ft. No. 12 weatherproof wire.

Account 8813—Fixtures.

This account covers the purchase price of the steel bins, shelving, counter scales, office partition and furniture, as listed below; also the erection cost.

197 ft. Bergers sectional steel bins and shelving. See sketch, lineal feet refers to half of bins shown by sectional elevation. Bins received knocked down, gauge of material 16 to 20.....	\$1,116.82
1 No. 1046 dormant warehouse scales, weighing 5,000 lb. to $\frac{1}{2}$ lb.....	141.42
Furniture, material for office partition, etc.....	282.88

(See Fig. 30.)

\$1,541.12

Account 8813.10—Painting.

This account covers the cost of material and labor of a lot of miscellaneous painting at the warehouse. The steel doors were given one coat of turkey red. The iron lintels were given one coat of lamp black in linseed oil. The counter was stained and oiled.

Account 8813.11—Painting Sash.

This account covers the cost of material and labor used to paint all the warehouse sash. They were given two coats of white lead and linseed oil, cream color.

Laboratory**Account 8820—Excavation.**

This covers the excavating for the laboratory walls and basement in red clay with boulders and gravel. It was done with pick and shovel, and wheeled 75 ft. in barrows. Some backfilling for the floors in 3-in. layers is also here included.

Account 8821—Foundation.

This covers the concrete building walls which were machine mixed, in the proportions 8 sand and gravel to 1 cement. The concrete was handled in wagons 250 yd. The walls were 12 in. at top to 18 in. at bottom. One hundred per cent. of the vertical surface was formed.

Account 8821.1—Plain Concrete Floors.

These floors were mixed, 5 sand and gravel to 1 cement, in a machine, transported 1,000 ft. in wagon and laid 4 in. thick with a smooth finish. Sand joints were used. The top finish was $\frac{3}{4}$ in. thick, 2 parts sand to cement, and was troweled smooth.

Account 8821.2—Reinforced Floors.

These floors were formed, two way reinforced with $\frac{1}{2}$ -in. and $\frac{3}{8}$ -in. rods. In other respects they were the same as 8812.1.

Account 8821.3—Sills and Lintels.

The sills and lintels used at the laboratory were separately moulded reinforced concrete. The concrete was made 3 parts sand and gravel to 1 cement. Three $\frac{5}{8}$ -in. rods ran the entire length of both sills and lintels. The lintels were 8 in. by 8 in. by 5 ft. The sills were $4\frac{1}{2}$ in. by 9 in. by 4 ft. 1 in. (See Fig. 34.)

Account 8822—Tile Walls.

This account covers the tile, mortar, scaffolds, mason labor, carpenter labor, and hauling incident to building the tile walls of the laboratory.

Account 8822.2—Carpenter Work.

This account covers the material and carpenter labor incident to the installation of the partitions, ceilings, and roof structure. (See Fig. 31.)

Account 8822.5—Doors, Windows and Frames.

This account covers the cost of the following material and the labor of installing the same, together with the necessary frames.

- 16 windows 6 ft. 8 in. by 3 ft. $1\frac{1}{8}$ in. by $1\frac{1}{8}$ in. glazed.
- 5 sash with four, 12 in. by 24 in. lights, glazed.
- 1 sky light 6 ft. $3\frac{1}{8}$ in. by 6 ft. $9\frac{1}{8}$ in. with 36, 12 in. by 14 in. double strength glass
- 1 sky light 6 ft. $3\frac{1}{8}$ in. by 4 ft. $6\frac{1}{2}$ in. with 24, 12 in. by 14 in. double strength glass.
- 1 sky light 4 ft. $1\frac{1}{8}$ in. by 4 ft. $6\frac{1}{2}$ in. with sixteen 12 in. by 14 in. double strength glass.
- 4 doors 2 ft. 8 in. by 6 ft. 8 in. by $1\frac{1}{8}$ in., glazed.
- 4 doors 2 ft. 8 in. by 6 ft. 8 in. by $1\frac{1}{8}$ in. glazed.

(See Figs. 32 and 33.)

Account 8824—Wood Fixtures.

The account covers the cost of the laboratory hoods, stacks, benches, cabinets and the like.

Account 8825—Lighting.

This account covers the wiring in the laboratory for lights, hot plates, furnaces, etc.

Account 8826—Plumbing.

This account covers the purchase price of the material below and the labor of installing it.

- | | |
|------------------------------|------------------------|
| 1 flush closet | 1 distilling apparatus |
| 2 sinks and drains | 1 water tap |
| Piping, fittings, lead, etc. | |

Account 8828—Painting.

This covers the cost of the material and labor incident to painting at the laboratory. It is of no value for unit costs.

Account 8829—Plastering.

This covers the cost of plastering material, scaffolds plasterers and carpenter labor used in plastering the inside tile walls of the laboratory. The plastering was contracted at 18 cents a square yard, while the company furnished all material and carpenter labor.

Account 8830—Apparatus.

This account covers the purchase price of the material segregated below, and the labor of setting up the same.

Five 12 in. by 18 in. by 6 in. 110-volt hot plates.....	\$143.08
1 Thompsons analytical balance style 28.....	91.65
1 distilling apparatus.....	62.70
1 Hoskins electric furnace.....	225.53
1 electric drying oven.....	41.79
Brushes, tubing, funnels, etc.....	54.23

\$618.98

Account 8831—Oil Centrifuge.

This apparatus was located near the oil tanks in a corrugated iron shed, 4 ft. by 4 ft. by 8 ft. The labor in this account was for the building as well as setting up the centrifuge and delivering the material.

1 Braun oil centrifuge, vertical, direct compound type, 110 volt, alternating.....	\$139.25
Material for shed.....	18.34

\$157.59

Sample Room**Account 8841—Excavation.**

This excavation covered the making of a thin top slice and shaping the ground for a plain concrete floor. The dirt was red clay. It was done with pick and shovel and cast to one side.

Account 8842—Foundation.

This covers the making of a few small concrete walls, machine mixed, 8 sand and gravel to 1 cement. Seventy-five per cent. of the vertical surface was formed. The concrete was handled about 450 ft.

Account 8842.1—Concrete Floors.

These floors were of smooth troweled concrete, 5 in. thick, machine mixed, 5 sand and gravel to 1 cement, the top finish 1 in. thick, 2 sand and 1 cement. All of the material was hauled in wagons 500 ft. The concrete was laid in blocks with sand joints.

Account 8843—Walls and Roof Structure.

This account covers the purchase price of the following material and the cost of erection.

Lumber for sides and roof rafters.....	\$148.12
Corrugated iron.....	47.68
Nails, etc.....	6.01
	\$201.81

(See Fig. 35.)

Account 8843.4—Roof.

This account covers the roof sheathing, composition paper, and the labor incident to installing it.

1,008 b.f., 1 in. by 12 Oregon pine.....	\$31.50
10 squares 3-ply asbestos roofing.....	45.07
	\$76.57

Account 8843.5—Doors and Windows.

This account covers the cost of the following material, and the cost of installing the same:

Two 3 ft. by 7 ft. by 1 $\frac{3}{8}$ in. doors.....	} \$115.24
Twelve 2 ft. 11 $\frac{1}{2}$ in. by 5 ft. 3 $\frac{3}{8}$ in. by 1 $\frac{3}{8}$ in. windows, glazed ..	
Frames for doors and windows.....	
Miscellaneous.....	3.61
	\$118.85

Account 8844—Oven.

This account covers the making of a drying oven, together with the cost of material and installation. The oven was made in the shops of $\frac{3}{8}$ -in. sheet plate, 4 ft. by 2 ft. by 6 ft. high, and lined with $\frac{3}{8}$ -in. asbestos mill board. It had suitable shelves of pipe coils.

One oven.....	\$46.77
Steam piping, etc.....	11.79
	\$58.56

Account 8845—Motor Platforms and Fixtures.

This covered the cost of the following material and its installation:

4 bucking boards, 3 ft. diameter by 2 $\frac{1}{4}$ in. thick	\$61.80
21 sheets galvanized iron (No. 18) for pans, etc	53.06
Lumber for one bench top, 3 ft. by 12 ft. for pulverizers, and one bench top 2 ft. by 12 ft. for cutting samples.....	14.74
	\$129.60

Account 8846—Lighting.

This covers the cost of the lighting wiring for seven lights.

Account 8848—Painting.

This covers the cost of material and labor incident to painting the sash of this building, two coats of white lead and linseed oil.

Account 8849—Machinery.

This covers the purchase price of the list of material given below, together with the labor of installing it.

	Factory	Freight	Clifton
Two 2 by 6 roll jaw crushers. (Sturtevant Milling Co.).....	\$250.00	\$51.80	\$301.80
4 jaw plates.....	10.00	0.80	10.80
4 shields.....	3.00	0.22	3.22
8 toggles.....	3.00	0.14	3.14
4 toggle seats.....	4.00	0.11	4.11
2 Jones samplers 10 by 18 in.....	40.00	2.25	42.25
4 Braun pulverizers, 9 in. pul.....			363.48
2 Braun coal grinders, light and loose pulleys, 12 in. diameter.....			147.62
Moisture scales.....			20.11
Sieves and miscellaneous.....			15.47
			\$912.00

Account 8849.1—Motor.

This covers the purchase price and cost of installing the following motor.

One 5-h.p. Westinghouse motor..... \$116.77

Account 8849.2—Shafting, Pulleys and Belting.

This account covers the cost of the material segregated below and the labor of installing it:

One 24 in. by 6 in. by 1½ in. bore wood split pulley.....	}	\$138.13
Four 26 in. by 6 in. by 1½ in. bore wood split pulley.....		
Two 22 in. by 2 in. Princeton clutch pulley.....		
26 ft. 1 in. of 1½ in. shafting.....		
4 rigid ring oiling pillow blocks.....		
Two 1½ in. safety set collars.....	}	32.04
Leather belting for motor 2 in., 3 in. and 5 in.....		
		\$170.17

MISCELLANEOUS ACCOUNTS**Account 8901—Derricks and Construction Equipment.**

This account is indirect expense, see 8999. The charge covers the cost of all derricks, concrete mixers, carts, wagons, picks, shovels, in fact every tool used during construction.

Account 8902—Sewer System—Cost of Pipe and Laying.

This account covers the cost of the material and labor of laying the same used in the sewer system. The sewer lines aggregated 2,967 ft. of vitrified sewer pipe, ranging from 6 in. to 15 in. in diameter. It was all laid an average depth of 4 ft. below the surface. Concrete manholes for the system are in account 8902.2.

Account 8902.1—Sewer System—Excavation.

This excavation covers the entire trenching or tunneling as was in some cases done for the sewer system. All kinds of soil were run through. The trenches varied from 18 in. to 60 in. wide, and from 2 ft. to 20 ft. in depth.

Account 8902.2—Sewer System—Concrete.

This concrete covered a large number of small jobs including manholes and the like along the sewer lines. In general the mix was 7 sand and gravel to 1 cement.

Account 8903—Permanent Outside Closets.

This covers 3 latrines built of 2 by 4's, 4 by 4's and corrugated iron. In size they were 8 ft. by 19 ft. 6 in. by 8 ft. high. The closet was built of concrete and is cleaned by a periodical flushing of water which is siphoned into a sewer.

Account 8904—Telephone System.

This account is indirect expense, see 8999. It covers the cost of a telephone line from Clifton to the smelter, together with all phones and connections about the plant during construction.

Account 8905—Permanent Outside Lighting.

The material account here is as follows:

Four 110 volt flaming arc lamps	\$142.40
Conduit, wire, etc.....	35.59
	\$177.99

The poles here used were old ones previously charged out to construction equipment. The lights were placed at various needed points about the plant and then connected to the nearest 110-volt circuit. Labor costs cover the setting of poles, the connecting of lamps, running a conduit line down each pole to a switch box, and installing the switch box.

Account 8905.1—Temporary Outside Lighting.

This account is indirect expense, see 8999.

Account 8906—Water Pipe Lines—Excavation.

This excavation covers the trenching for all the water lines. It represents all sorts of material excavated from 8 ft. to 15 ft. in depth.

Account 8906.01—Water Pipe Lines—Concrete.

This was a small amount of concrete used to anchor the 6-in. line at the foot of the hill, as it comes down from the tank.

Account 8906.02—Water Pipe Lines—Cost and Laying.

This account covers the cost of the material and labor of laying all the water lines about the smelter.

There was 2,052 ft. of 6-in. pipe
 1,058 ft. of 4-in. pipe
 200 ft. of 2½-in. pipe
 268 ft. of 2-in. pipe
 115 ft. of 1½-in. pipe
 50 ft. of 1-in. pipe

4,253 ft.

Total of 4,253 ft. with all necessary fittings, valves, and fire hydrants.

Account 8906.1—Six-Inch Pipe Line from Clifton.

This account covers the cost of the material and labor incident to laying a 6-in. water pipe line, 8,988 ft. long, from Clifton to smelter. It includes excavating, painting, and backfill.

Account 8906.2—Water Supply Tank—Excavation.

This covers the making of a 3-ft. slice for a water tank foundation. It involved the use of powder, and was handled with picks, shovels, and wheelbarrows.

Account 8906.4—Water Supply Tank—Cost and Erection.

This tank was erected on a hill at an elevation of 200 ft. above the tracks where the material was received. It is 40 ft. in diameter, 26 ft. 9½ in. high, and has a steel cover. Its capacity is 250,000 gallons. The tank steel was erected by contract. The account stands thus:

Road work.....	\$72.80
Hauling.....	121.31
33.67 tons, tank erected.....	3,550.00
Freight on tank.....	235.69
Power for riveting.....	92.31
Miscellaneous.....	64.92

\$4,137.03

Account 8907—Watchman.

This account is indirect expense, see 8999. It covers the pay of watchman during the construction period.

Account 8908—Power Distribution.

This account covers the cost of material and labor incident to the following work. Two- and three-inch conduits were run underground

from the power house to the heater house and to the converter building. Along these lines four concrete manholes or pull boxes were installed. At the converter building the wire from the conduits run up a steel column and from thence on steel brackets through the reverberatory and boiler building to a point of distribution by poles. From this point the wires go to the roasters, sample mill, crushing plant, and bedding plant. The wire ranged from 1,000,000 c.m. to No. 12 weatherproof.

Account 8908.1—Temporary Oil Tanks.

This account is indirect expense, see 8999. It covers the cost and erection of an oil tank with pipe lines used during the construction period.

Account 8909—Permanent Air Line—Excavation.

This covers the cost of trenching and backfilling for the air lines. The trenches were in every kind of soil, 18 in. to 6 ft. deep and 1 ft. to 3 ft. wide.

Account 8909.1—Permanent Air Line—Laying.

This account covers the cost of material used and the labor of installing the same. The lines together are 2,316 ft. long and composed of the following quantities of different sized pipe.

526 ft., 1-in. pipe	656 ft., 2-in. pipe
36 ft., 1½-in. pipe	838 ft., 3-in. pipe
80 ft., 1½-in. pipe	180 ft., 4-in. pipe

Account 8910—Transmission of Power to Various Departments.

This is a suspense account which has been charged away.

Account 8911—Watchman House.

This account is indirect expense, see 8999. It covers the cost of a temporary building.

Account 8912—Tool Shed.

This account is indirect expense, see 8999. It covers the cost of a temporary building.

Account 8913—Barn and Corral.

This account is indirect expense, see 8999. It covers the cost of the corral and barn.

Account 8914—Temporary Blacksmith Shop.

This account is indirect expense, see 8999. This is a temporary building used during construction.

Account 8916—Temporary Power Plant.

This account is indirect expense, see 8999. It covers a large compressor, Maune type boiler and temporary building, together with the labor of installation.

Account 8917—Temporary Crushing Plant.

This account is indirect expense, see 8999. It covers the cost and installation of a 10-in. by 20-in. crusher, 65-ft. bucket, elevator, rock and sand bins, shafting, belts, pulleys and motors.

Account 8918—Temporary Water Tanks.

This account is indirect expense, see account 8999. It covers several small tanks erected for use at the temporary camp site, as well as a large general supply, wooden tank.

Account 8919—Temporary Electrical Shop Equipment.

This account is indirect expense, see 8999. It covers a temporary shop equipment.

Account 8920—Wagon Roads.

This account is indirect expense. It covers the building of many roads and trails required during the construction period.

Account 8921—Temporary Pumping Plant.

This account is indirect expense. It covers a timbered shaft, pump and pipe line.

Account 8922—Temporary Pipe Lines.

This account is indirect expense. It covers the cost of all temporary water and air lines laid during the construction period.

Account 8923—Temporary Warehouse.

This account is indirect expense. It covers the cost of the temporary warehouse and equipment.

Account 8924—Temporary Cement Sheds.

This account is indirect expense, see 8999. It covers the cost of corrugated iron sheds for storing cement.

Account 8925—Horses, Harness and Carts.

This is indirect expense, see 8999. This covers the cost of some carts, work harness, and the purchase price of an animal killed during construction.

Account 8926—Temporary Office.

This account is indirect expense, see 8999. This covers the cost of the construction office.

Account 8927—Temporary Lavatories.

This account is indirect expense, see 8999.

Account 8928—Temporary Machine Shop.

This account is indirect expense, see 8999. This covers the cost of temporary machine shop.

Account 8929—Employee's Railroad Transportation.

This account is indirect expense, see 8999. During the construction period 300 to 500 men were employed, over three-fourths of whom lived in and around Clifton. The cost of transporting these men back and forth each day was borne by the construction.

Account 8930—Clearing Land.

This account is indirect expense, see account 8999.

Account 8931—Test Holes.

This account is indirect expense, see 8999.

Account 8933—Furniture and Fixtures.

This account is indirect expense, see 8999. It covers the temporary office fixtures.

Account 8934—Miscellaneous Supplies.

This account is indirect expense, see 8999.

Account 8935—Shop Equipment.

This account is indirect expense, see 8999. It covers material used in various temporary shops.

Account 8936—Overhead Shop Expense.

This account is indirect expense, see 8999. This account covered all labor and supplies used in the shops not directly charged to the work in hand.

Account 8937—Stock Lumber.

This is a suspense account.

Account 8938—Powder Magazine.

This account is indirect expense, see 8999.

Account 8939—Miscellaneous Labor.

This is indirect expense, see 8999.

Account 8941—Temporary Railway Receiving Bins.

This account is indirect expense, see 8999.

Account 8942—Water Supply.

This account is indirect expense, see 8999. It covers the cost of water from Clifton used during construction.

Account 8943—Corral Expense.

This was suspense account distributed at the close of each month, on the basis of the cost per animal-day.

Account 8944—Switching and Freight from Clifton.

This account is indirect expense, see 8999. It was impossible to place these charges to the material freighted, owing to incomplete records.

Account 8945—Office Stationery and Supplies.

This account is indirect expense, see 8999. .

Account 8946—Warehouse Operating Expense.

This account is indirect expense, see 8999. It covers the expense of running the warehouse during the construction period.

Account 8927—Timekeeping Expense.

This account is indirect expense, see 8999. It covers the expense of the timekeeping and distribution during the construction period.

Account 8948—Form Lumber.

This is a suspense account. Its money was apportioned to all concrete costs on the basis of board-feet of lumber used on each job.

Account 8949—Cement.

This was a suspense account. It was apportioned to all concrete accounts on the basis of sacks used.

Account 8951—Sand and Gravel.

This is a suspense account. Its money was apportioned to the different concrete accounts on the basis of cubic yards of concrete cast.

Account 8952—Employee's Quarters.

This account is indirect expense, see 8999. It represents the cost of a boarding house, bath house, and 42 tent houses, less the rent they paid on during the construction period.

Account 8953—Crushing Plant Operating Expense.

This is a suspense account. Its money was apportioned to the different concrete jobs on the basis of cubic yards of concrete benefiting.

Account 8954—Concrete Power and Repairs.

This is a suspense account, see 8953.

Account 8955—Mortar Sand.

This is a suspense account. It was apportioned to the brick-work on the basis of the amount used by the various jobs.

Account 8955.1—Mortar Lime.

This is a suspense account, see 8953.

Account 8955.2—Mortar Cement.

This is a suspense account, see 8955.

Account 8955.4—Fire Brick Mortar.

This is a suspense account, see 8955.

Account 8955.5—Silica Brick Mortar.

This is a suspense account, see 8955.

Account 8956—Operating Temporary Power House.

This is a suspense account. It was apportioned to the various jobs on the basis of horse power days.

Account 8957—Maintenance of Track in Yards.

This account is indirect expense, see 8999. It represents the upkeep of tracks during construction, and the cost of many temporary construction tracks.

Account 8958—Ditch at Tunnel No. 2.

This account is indirect expense, see 8999. It represents a ditch built by the Arizona & New Mexico Railway, to protect their main line which has been endangered by surface water as a result of the site chosen for the smelter.

Account 8961—Steam Heating System—Excavation.

This covers the making of a long shallow trench and backfilling it. The ground was red clay.

Account 8961.1—Steam Heating System—Cost and Installation.

This covers the cost of laying 260 ft. of 2-in. steam pipe and 236 ft. of 2½-in. steam pipe in a 2-in. lumber box. The pipe was covered with double standard magnesia covering.

Account 8975—Cleaning Up.

This account is an indirect charge, see 8999. It represents cleaning up about the plant after construction.

Account 8976—Rehandling Brick and Tile.

This account is an indirect charge, see 8999. It represents handling tile not directly chargeable to the jobs where the tile was used, but as a result of other considerations.

Account 8998—Direct Charges.

This account is indirect expense, see 8999. It represents direct charges to indirect expense.

Account 8999—Indirect Expense.

This account is a summation of the accounts, as listed under charges to indirect expense. As the total of indirect expense, they represent a percentage of the total cost of the smelter, less the engineering and indirect expenses, and have been so reported. In the making of any total estimate based on the unit costs derived from this sheet, it is assumed that of the total estimate, 7.53 per cent. will be taken to ascertain the item of indirect expense.

POWER PLANT

Power House

Account 9001—Excavation.

This excavation was a large cut, about 55 ft. by 280 ft. by 10 ft., for the basement of the power house, the machine foundation and the building piers. The material encountered was red clay and boulders on top, with sand and gravel beneath which was saved for concrete material. Powder was used, followed by plowing, picks, shovels, fresnos, and carts. The material was hauled 450 ft. (See Fig. 53.)

Account 9002—Building Foundation Piers.

This concrete was cast in piers which supported the steel columns. The piers were about 3 ft. by 4 ft. by 3 ft. plain concrete, hand mixed, in the proportions of 7 sand and gravel to 1 cement, and transported 150 ft. in wheelbarrows to place. A great deal of difficulty was experienced here in keeping out the sand and gravel which constantly *sluffed* in from the sides. Fifty per cent. of the vertical surface was formed.

Account 9002.1—Building Foundation Walls.

This concrete was cast as a long reinforced wall running around outside and bearing on the columns of the building. It is 12 in. at top, 20 in. at bottom, 11 ft. high, designed as a slab to withstand earth pressure on a 20-ft. span. The concrete was machine mixed in the proportions of 5 sand and gravel to 1 cement, wheeled 150 ft. to place and reinforced with $\frac{1}{4}$ - and $\frac{3}{4}$ -in. rods variously spaced. Ninety-five per cent. of the vertical surface was formed. (See Fig. 47.)

Account 9002.2—North Tunnel.

This concrete was cast as a box culvert in an open cut. It is about 6 ft. by 7 ft. by 300 ft. long. It was hand mixed in the proportions of 5 sand and gravel to 1 cement, wheeled 100 ft. to place and reinforced with $\frac{1}{4}$ in. and $\frac{1}{2}$ -in. rods. Fifty-five per cent. of the vertical surface was formed, as well as the roof of the culvert. (See Fig. 50.)

Account 9002.3—Concrete Drain.

This concrete forms a drain through the middle of the power-house basement. There are two walls, 10 in. thick, 18 in. high, 3 ft. apart, with a 9-in. bottom in the drain laid on grade. The concrete was one-half machine and one-half hand mixed, in the proportions of 7 sand and gravel to 1 cement, wheeled in barrows 150 ft. to place. Ninety per cent. of the vertical surface was formed.

Account 9002.4—Basement Floor—Concrete.

This concrete was cast as a floor on grade in the basement of the power house. It was laid in 6-ft. blocks with sand joints. The mixture.

machine made, was 7 parts sand and gravel to 1 cement. The top finish was $\frac{3}{4}$ in. thick and made 2 parts sand to 1 cement. The total thickness of the floor is 4 in. The concrete was transported in barrows on an average of 125 ft.

Account 9002.45—Basement Floor Painting.

This account covers material cost and labor of cleaning floor and applying two coats of Toch Brothers cement filler.

Account 9002.6—Preparing of Concrete for Painting.

This covers the cost of preparing basement concrete walls for painting. The air holes were filled, rough surfaces smoothed, and one coat of cement grout applied.

Account 9002.7—Painting Concrete.

This account covers the cost of the material and labor of applying it to the power house basement walls. One coat of Wadsworth Howland Bay State cement coating was put on. This cost \$1.88 a gallon f.o.b. Clifton. The material account is as follows:

150 gallons of paint.....	\$281.40
Brushes.....	8.42
Turpentine.....	5.71
Miscellaneous.....	6.08
	\$301.61

Account 9003—Steel Structure.

This account covers all the structural steel including railings and the like used in the power house building. There were 254.29 tons. (See Fig. 54.)

Account 9003.1—Tile Walls.

This account covers the cost of all the tile, mortar material, scaffolds, mason labor, carpenter labor, and hoisting power required to lay up the walls. The walls were 40 ft. high on the average from the ground level. The walls were 8 in. thick, laid directly upon the concrete walls. See account 9002.1. Two mortars were used in laying up the tile, one with lamp black in to give the outside joint a pleasing appearance. The joints were struck. The mortar mixture was 1 cement, 3 sand, and 1 lime. (See Figs. 52 and 54.)

Account 9003.11—Unloading Tile.

This account covers the preparing of the site for unloading, checking quantities and unloading the tile used in the power plant.

Account 9003.12—Wall Coping.

Similar to account 8703.22, which see. The depth of this coping was, however, 18 in. against 12 in. at the warehouse and shops.

Account 9003.2—Doors, Windows and Frames.

This account covers the doors, sash, hardware and frames listed below, together with the labor cost of installing the same. The steel frames were made by the Kansas City Structural Steel Co.

	Factory	Freight	Clifton
105 single sash, 2 ft. 8½ in. by 5 ft. 1½ in. by 1½ in. glazed with 15 ribbed glass lights, fixed sash.....	\$325.50		
1 single sash, 2 ft. 8½ in. by 4 ft. ½ in. by 1½ in. glazed with 12 ribbed glass lights, fixed sash.....	26.00		
73 windows, 2 ft. 8½ in. by 6 ft. 3 in. by ½ in. glazed with 18 ribbed glass lights, double hung.....	284.70		
40 sash, 3 ft. ½ in. by 2 ft. 6½ in. by 1½ in. glazed with 6 ribbed glass lights, hinged. . .	56.00		
		<u>\$93.05</u>	<u>\$785.25</u>
Steel frames for all but the 40 6-light sash and 6 door frames, 3 ft. 1 in. by 7 ft. 6 in. . .	1,815.20	292.77	2,107.97
300 Caldwell sash balances, No. 18.....			111.95
3,884 ft. b.m. lumber for frames for 40 single sash.			91.44
Locks, hinges, nails, etc.....			223.32
			<u>\$3,319.93</u>

(See Fig. 51.)

Account 9003.21—Concrete Sills.

This account covers the material used and the labor expended in making the power house concrete sills. About one-half of the sills were cast in place and the remainder as separately moulded members. They were 8 in. by 10 in. for the windows and 3 in. by 10 in. for the fixed sash at the top of the building. The material was hand mixed, 3 sand to 1 cement, and 3½-in. rods ran the length of the sills. (See Fig. 51.)

Account 9003.3—Ventilators.

This covers the cost of 6 48-in. Burt ventilators with square base and the labor of installing them on the roof of the power house. They were lifted onto the roof with a locomotive crane.

6 Burt ventilators.....	\$280.17	\$152.57	\$432.74
Miscellaneous.....			7.02
			<u>\$439.76</u>

Account 9003.4—Main Floor Columns.

This account covers the cost of the material, fabrication, and labor of erecting 57 pipe columns and 11 structural steel columns to support the steel beams of the power house floor. The 57 pipe columns were made of 4-in. pipe, with a cast-iron base and capital, and were so placed

that they might be easily shifted a foot or two along the low flange of the I beams they support, should they interfere with future piping schemes. The structural columns were permanently located. The material segregated is as follows:

Caps and bases, cast iron.....	\$227.80
668 ft. of 4-in. pipe.....	186.40
Structural steel and rivets.....	136.99
Miscellaneous.....	75.25
	\$626.44

Account 9003.41—Main Floor Slab—Concrete.

This account covers the cost of the Berger multiplex plate laid on the steel I beams of the power-house floor, and covered with concrete reinforced with $\frac{1}{2}$ -in. and $\frac{1}{4}$ -in. rods. The Berger plate was laid upon steel beams and not wired; $\frac{1}{2}$ -in. and $\frac{1}{4}$ -in. rods, 1 ft. on concrete, were laid in the concrete at right angles with the grooves of the plate. The concrete mixture was 7 parts sand and gravel to 1 cement, machine made. The top finish was proportioned 2 sand to 1 cement. It was laid $\frac{3}{4}$ in. thick, smooth troweled, and marked off in 6-ft. squares. The material account stands thus:

102.1 square, 3-in. Berger plate	\$2,101.88
$\frac{1}{2}$ -in. and $\frac{1}{4}$ -in. reinforced steel.....	62.06
Concrete materials.....	1,177.67
	\$3,341.61

(See Fig. 55.)

Account 9003.42—Painting under Side Main Floor.

The Berger plate exposed beneath the floor of 9003.41 was cleaned of rust and painted two coats of linseed oil and white lead, cream color. This necessitated low scaffolds. The square yards in which the unit cost is reported are the yard measurements derived from developing the plate. This cost covers the material and the labor.

Account 9003.43—Painting Top Main Floor.

This account covers the labor and material used in painting the upper surface of the power house Berger plate concrete floor. It was given two coats of Toch Brothers cement filler and one coat Toch Brothers warm gray cement paint, after the floor had been well cleaned and dried out.

Account 9003.5—Berger Multiplex Plate.

This account covers the material and labor used in putting Berger plate upon the power-house roof. The eaves of the roof are 41 ft. from the ground and the floor is $\frac{1}{4}$ pitch. The Berger plate was hoisted

to place by use of a single pulley and hand rope. When laid in position it was wired to the purlins with No. 10 wire. The segregated account stands as follows:

Berger plate.....	\$2,962.25
Wire.....	20.41
Tools, etc.....	80.52
	\$3,063.18

(See Fig. 56.)

Account 9003.51—Roof Concrete.

This account covers the concrete placed on the roof. It was hoisted to the eaves at various places and transported to position in hand buckets. The concrete was hand mixed, 5 sand and gravel to 1 cement. The top finish was not used, but the concrete was straight-edged to proper level and troweled as smooth as possible. (See Fig. 56.)

Account 9003.52—Roof Tar.

This account covers the cost of the material below, and the labor used in applying it to the roof. A composition of tar, cement and coal oil was made and painted directly upon the concrete roof in an effort to make it waterproof.

11 barrels of tar.....	\$113.74
7 sacks cement.....	4.95
54 gallons coal oil.....	6.48
Miscellaneous.....	2.56
	\$127.73

Account 9003.53—Roof Down Spouts and Tile Drain.

This covers the cost of the material used in 10 down spouts and drains, together with the labor employed in erecting them. The down spouts are 4-in. galvanized iron spouts which, after leaving the gutters, pass directly through a hole prepared in the building wall to the inside of the building, and thence to the basement floor. At this point they enter a 4-in. vitrified sewer pipe which is laid beneath the floor discharging into the drain down the center of the building. The material account stands as follows:

355 ft. 4-in. vitrified sewer pipe.....	\$117.90
550 ft. 4-in. galvanized iron drain pipe.....	90.87
Miscellaneous.....	31.67
	\$240.44

Account 9003.54—Roof Painting Underside.

This account covers the cost of the paint material, brushes, scaffolds, etc., together with the labor required to paint the underside of the

power-house roof. High swinging scaffolds from the roof truss purlins were used to work from. They were slow and difficult to move from place to place. The rust was cleaned off and it was given two coats of white lead and linseed oil, cream color.

Account 9003.55—Roof—P and B Roofing.

This account covers cost of material and labor incident to applying a paper roof. Wood strips were imbedded in the concrete around the base of each ventilator, and across the roof at the juncture of the lean-to roof with the main building roof. A cement coating was then applied to the concrete followed by application of hot maltha, with sheets of felt imbedded in the maltha. The felt was so lapped one piece upon another as to give three thicknesses over the entire roof. Another application of hot maltha was swabbed over this ground work, followed by one course of three-ply P and B roofing paper.

Account 9003.60—Painting Sash.

This account covers the material and labor incident to painting all the power-house sash three coats. Linseed oil and white lead was used.

Account 9003.61—Painting Woodwork.

This account covers the painting of the power-house doors and miscellaneous woodwork.

Account 9004—Crane.

This crane has a capacity of 20 tons. It is operated by hand and spans the power-house floor, a distance of 50 ft. It has a 40-ft. lift and runs on 50-lb. rails. The account covers the material as shown below, and the labor of erecting the crane:

One 20-ton hand traveling crane.....	\$1,278.00
Freight on same.....	408.00
Miscellaneous.....	37.27
	\$1,723.27

Account 9005—Well Grading.

This covers the cost of grading off a point of conglomerate rock in preparing a site for a well. Large blasts of dynamite were used.

Account 9005.1—Shaft Sinking.

This covers the cost of sinking a shaft in conglomerate at the waters edge upon the site prepared by account 9005. The shaft was 6 ft. by 8 ft. and went to a depth of 45 ft. It was necessary to install and run a No. 7 Cameron pump to handle the water.

Account 9005.2—Timbering.

The timbers used were 8 in. by 8 in., Oregon pine square-shaft sets, making two compartments in the shaft. The shaft was lagged.

Account 9005.31—Aldrich Pump Installation.

The money expended under this account was for unloading two pumps which were not installed.

Account 9006.01—Nordberg Blowers—Foundation.

These two concrete foundations were each about 30 ft. by 15 ft. by 20 ft. The concrete was mixed 6 parts sand and gravel to 1 cement by machine, and transported 50 ft. to place. 100 per cent. of the vertical surface was formed. Each foundation had over thirty bolts set exact with template and piped. The pipes were in short pieces pulled up as the foundation raised and out at the completion. The cost of these pipes and the bolts are in every case given in the concrete cost. (See Fig. 49.)

Account 9006.1—Nordberg Blowers—Cost and Installation.

This account covers the cost of the material as listed below, together with the labor of erecting the same. These engines are two Nordberg cross-compound blowing engines, designed to compress 10,000 cu. ft. of free air at an altitude of 3,500 ft. to 12 lb. pressure, while 15 lb. may be carried if desired. The high-pressure steam cylinder is 20 in., the low-pressure 42 in., while the air cylinders are 44 in., all having the common stroke of 42 in. The engines are furnished 160 lb. steam pressure, superheated 75° F. The speed is 71 r.p.m. The labor of grouting, and the labor of testing out and starting up are included here.

2 Nordberg blowing engines, with receivers.....	\$30,967.34
2 No. 34 crane tilt traps.....	107.78
Grout, etc.....	1,438.90
	\$32,514.02

Account 9006.2—Nordberg Blowers—Painting.

This account covers the cost of material and labor of painting the two Nordberg blowing engines. All of the power-house machinery was painted by contract for the sum of \$820. This sum covered the labor and all tools, such as brushes, putty knives, light ladders, etc. The paint, oils, colors, dryers, and scaffolds where necessary were furnished by the company. The money covered by the contract and material used was apportioned to the painting account of the different pieces of machinery on the basis of the time spent on each piece of machinery. Every machine was given one coat of paint, one coat of filler, and two coats of olive-green enamel.

Account 9007.01—Turbines—Foundation.

This concrete is identical with 9006.01.

Account 9007.1 Turbines—Cost and Installation.

This account covers the purchase price of three Curtis turbines and material as listed below, together with the labor of erection, grouting, wiring from generator to switchboard, testing and starting up. The turbines are 2,000-k.w. Curtis-type horizontal shaft engines and direct connected to 2,500-k.v.a., 6,600-volt, 60-cycle, 3-phase, 1,900-r.p.m. generators. The approximate size of each unit is 23 ft. 8 in. long by 10 ft. 6 in. wide by 9 ft. 7 in. high, with a net weight of 108,300 lb.

3 turbines.....	\$77,828.10
486 gallons of gargoyle turbine oil.....	233.04
Grout, electrical material.....	1,525.35
	\$79,586.49

Account 9007.2—Turbines—Painting.

See account 9006.2.

Account 9007.3 Turbines—Air Pipe Making.

(See Fig. 57.) This account covers the making of the air ducts for the three turbines. They were fabricated in the smelter shops of No. 16 steel with $2\frac{1}{2}$ by $2\frac{1}{2}$ by $\frac{1}{4}$ angles. The total length for the three was 103 feet.

Account 9007.4—Turbines—Air Pipe Erection.

This account covers the labor of erecting the air ducts in account 9007.3. The material used was cloth insertion packing, rivets, hangers, anchors, etc.

Account 9007.51—Transformer Trucks and Transfer Table.

This account covers the placing of 325 ft. of 40-lb. rail for installing 15 transformers with trucks, and the cost of those trucks. (See Fig. 58.)

Account 9007.52—Auto Transformers.

This account covers the cost of the 10 transformers oil and wire as listed below, together with placing wiring, trying out and testing the same.

10 oil-cooled auto transformers for raising voltage from 6,600 to 13,200 volts, G.E. Type "H," 60 cycles.....	\$11,801.81
417 k.v.a. Y connected. (3 transformers to a turbine and one spare).	
Oil and wire.....	243.10
	\$12,044.91

Account 9008.01—Condensers—Foundation.

These were plain foundations machine mixed, in the proportions of 1 cement to 6 sand and gravel. The vertical surfaces were 100 per cent. formed.

Account 9008.1—Condensers—Cost and Installation.

This covers the cost of 3 Alberger surface condensers and the labor of placing and grouting them in position. Each condenser has 7,600 sq. ft. of surface.

3 Condensers.....	\$19,436.04
Grouting, etc.....	127.51
	\$19,563.55

Account 9008.2—Condensers—Painting.

See account 9006.2

Account 9009—Jet Condenser—Hot Well Excavation.

This covers the cost of making small excavations for a few piers in red clay with pick and shovels.

Account 9009.01—Jet Condenser—Hot Well Foundation.

This covers a small amount of concrete for piers, hand mixed—6 sand and gravel to 1 cement.

Account 9009.02—Jet Condenser—Hot Well Supporting Structure and Tank.

This account covers the cost and erection of 5.76 tons of steel. There was a quadrangular tower 19 ft. 6 in. high, with about 12 ft. base, surmounted with a 10 ft. diameter by 8 ft. 6 in. high steel tank. It was furnished by the Kansas City Structural Steel Co.

Account 9009.03—Jet Condenser—Cost and Erection.

This covers the cost of one 28-in. Alberger type "F," barometric jet condenser and erection above the tank of account 9009.02.

Account 9009.12—Jet Condenser—Dry Vacuum Pumps.

These air pumps remove the air from the barometric condenser and are located in the power house. The account covers the cost of the material listed below and the labor of erecting the same.

Two 15-h.p. slip ring motors, 440 volts, 3 phase, 60 cycles, 565-r. p. m., with resistance controllers.....	\$739.92
Two 16 by 12 single-stage Alberger dry vacuum pumps.....	1,888.82
2 circuit breakers.....	39.88
Grout, cable, condulets, etc.....	191.99

\$2,860.01

Account 9009.13—Jet Condenser Dry Vacuum Pump—Painting.

(See account 9006.2.)

Account 9009.21—Circulating Pumps—Foundation.

These are about 15 ft. by 20 ft. by 10 ft. and are similar to 9006.01 in other respects:

Account 9009.22—Circulating Pump—Cost and Erection.

These air pumps furnish the circulating water for the barometric condenser. The cost here includes the price of the material listed below and the labor installing the same.

Two 35-h.p., 440-volt, 60-cycle, 570-r.p.m. motors,.....	\$1,687.50
Two 2 Lobe cycloidal jumps, 14 by 12, 17.8 gal. per rev.....	2,341.41
2 oil switches, 660 volt.....	39.89
Miscellaneous.....	66.88
	\$3,535.68

Account 9009.23—Circulating Pumps—Painting.

(See account 9006.2.)

Account 9010.01—Air Compressor—Foundation.

This concrete is 10 ft. by 20 ft. by 15 ft. high. In other respects it is similar to 9006.01.

Account 9010.02—Air Compressor—Erection.

This account covers only the erection at the smelter of the following Ingersoll-Rand two-stage compressor. It was bought from the mines and erected at the smelter power house. The compressor has a steam-driven cross-compound Corliss engine. The steam cylinders are 13 in. and the air cylinders are 22 in. and 13 in. and the common stroke is 36 in.

Account 9010.04—Air Compressor—All Piping Except Steam.

This account covers the cost and erection of all the piping to the Ingersoll-Rand Co. compressor except the steam piping.

Account 9010.05—Air Compressor—Wrecking and Transportation.

This account covers the labor of tearing down this compressor at Morenci, loading it on cars and the freight to the smelter. Such material is charged as was incident to these operations.

Account 9010.06—Air Compressor—Installation of Air Receivers.

This account covers the labor of installing a small air receiver in the power-house basement, for the compressor. No charge was made for the receiver.

Account 9011.01—Two Exciters, Two Air Pumps, Two Circulating Pumps—Foundation.

This is a large foundation about 15 ft. by 20 ft. by 10 ft. In other respects it is the same as account 9006.01.

Account 9011.02—Two Exciters—Cost and Installation.

This account covers the cost of the material listed below and the labor of installing the same. These are the exciters for the turbine generators. They are Ridgway tandem, compound, balanced, slide-valve engines, direct connected to 75-k.w., 125-volt direct-current generators with a speed of 275 r.p.m.

2 exciters.....	\$5,744.96
Cable wire etc.....	286.28
Miscellaneous.....	87.02
	\$6,118.26

In the labor cost is included wiring and connecting the machines to the switchboard, as well as erecting grouting and trying out.

Account 9011.03—Three Dry Vacuum Pumps—Cost and Installation.

These pumps are for the surface condensers. The account covers their cost, erection, grouting and trying out. They weighed 14,000 lb.

3 dry vacuum pumps 8-in. steam by 20-in. air by 12-in. stroke.	\$3,136.11
Grout, packing, etc.....	53.99
	\$3,190.10

Account 9011.04—Three Circulating Pumps and Engines—Cost and Installation.

These pumps furnish the circulating water for the surface condensers. This account covers the cost of the material listed below and the labor of erecting and trying out.

3 Lobe, 18 by 20, cycloidal pumps, capacity 49.5 gallons per rev., and Three 27-in. flexible couplings.....	\$4,425.25
Three 11 by 14 Ridgway, simple balanced, slide-valve engines for direct connection to above pumps	4,124.60
Grout, packing, etc.....	179.43
	\$8,729.37

Account 9011.05—Two Exciters—Painting.

See account 9006.2.

Account 9011.06—Three Air Pumps—Painting.

See account 9006.2.

Account 9011.07—Three Circulating Pumps—Painting.

See account 9006.2.

Account 9012.01—Two Motor Generators, One Air Pump, One Circulating Pump—Foundation.

These foundations are 28 ft. by 18 ft. by 11 ft. In other respects they are similar to 9006.01.

Account 9012.02—Two Motor Generators—Cost and Installation.

This account covers the material listed below as well as the labor of unloading, erecting, grouting, wiring to switchboard, and trying out.

Two 150-k.w. synchronous motor-generator sets to supply 250	
volt d.c.....	\$6,450.16
Conduit and wire.....	317.36
Miscellaneous.....	62.81
	\$6,830.33

Account 9012.05—Two Motor Generators—Painting.

See account 9006.2.

Account 9013—Transfer Table Pit—Concrete.

This is principally a plain concrete slab, 8 in. thick, mixed by hand in the proportions of 6 sand and gravel to 1 cement. About 10 per cent. of the vertical surface was formed. It was chuted to the basement and wheeled in barrows 100 ft. to place.

Account 9013.01—Switchboard—Concrete Compartments.

This is a concrete switchboard, the large dimensions of which are 40 ft. 4 in. long, 4 ft. wide and 13 ft. 2 in. high. Down the center of the board is a 4-in. wall and on each side are 30 pockets made with 2-in. dividing walls. Reinforcing was done with Clinton wire mesh in the 4-in. wall and $\frac{1}{2}$ -in. and $\frac{3}{4}$ -in. rods were used elsewhere. A great many bolts and insulators were set in the board. The entire board was cast sectionally in place, using 5 parts of sand and gravel to 1 part cement. This cost includes rubbing down the concrete with pumice stone and filling all the air bubble holes and small voids.

Account 9013.02—Switchboard—Cost and Erection.

This account covers the material price of the secondary switchboard slabs. The primary or concrete switchboard construction, however, is in account 9013.01. Here, too, is the material price of the entire equipment for all of the switchboards, both primary and secondary, and the labor of installing the same.

Account 9014—Steam Piping North and South Mains—Excavation.

This covers the excavation for numerous piers done with pick and shovel and cast to one side.

Account 9014.01—Steam Piping North and South Mains—Foundation.

This concrete composes the piers which support the long structural steel steam pipe supports. They were part mixed by machine and part by hand, in the proportions of 6 sand and gravel to 1 cement. About 50 per cent. of the vertical surface was formed.

Account 9014.02—Steam Piping North and South Mains—Steel Supporting Structure.

In these steam-pipe trestle supports 11.8 tons of corrugated iron and 75.01 tons of structural steel were used. (See Fig. 60.)

Account 9014.03—Steam Piping Mains, Hangers and Anchors.

This covers the cost of material and making of all hangers and anchors used for the steam piping between the boilers and the machines in the power house. The hangers were made of $\frac{3}{4}$ -in. rods and $\frac{1}{2}$ -in. by $2\frac{1}{2}$ -in. clamps running around pipe. Anchors were of same material fabricated to suit conditions surrounding place used. (See Figs. 60 and 61.)

Account 9014.04—Steam Piping—Cost and Erection.

Under this account all the material listed below is costed, together with the labor of its erection. These pipes run from the boilers to the power house in duplicate, making a complete loop about 1,120 ft. around. The main lines are 10 in. branches from boilers 8 in. and branches to engines of suitable size ranging from 4 in. to 8 in. All the joints are Van Stone, all valves and fittings are of cast steel. The line is required to stand 180 lb. pressure with 100° F. superheat. The gaskets used are corrugated bronze. The 10-in. lines are fitted with six 10-in. Harter expansion joints. (See Figs. 60 and 61.)

Six 10-in. Harter expansion joints.....	\$1,684.77
One 6-in. cast-iron separator.....	126.55
Two 10-in. cast steel vertical separators.....	843.47
One 10-in. cast steel horizontal separator.....	372.48
Two 6-in. separators and receivers (bot. outlet).....	591.77
One 5-in. cast steel separator and receiver (bot. outlet) ..	261.40
Three 4-in. cast steel separators and receivers (bot. outlet)	687.43
Two 4-in. cast steel separators and receivers (bot. outlet)	476.28
Corrugated bronze gaskets.....	251.93
Ten 8-in. Lagonda valves.....	1,315.52
Twelve 10-in. gate valves.....	2,079.00
Two 34-in. and one 33-in. Crane tilt traps.....	143.69
Best Mfg. Co. pipe and fittings.....	8,738.89
Extra pipe and fittings.....	526.18
Miscellaneous.....	522.89

\$18,622.25

Account 9014.05—Steam Piping—Covering and Erection.

Part of this work was contract by the people who furnished the material and part was force account. Therefore the material figure given covers a large portion of labor. The total unit cost is the only valuable unit. The steam lines and all fittings were all covered with 85 per cent. magnesia blocks of double standard thickness, wrapped with 6-oz. duck. All of the line was then painted two coats—cream color.

Account 9015—Exhaust Pipe—Cost and Erection.

This covers the cost of the material as listed below and the labor incident to installing it. Some of the piping is cast iron, designed for a vacuum of 14 lb. per square inch. The rest of the piping is lap-welded wrought steel with cast-iron fittings. The installation covers the three 20-in. atmospheric exhausts from the turbines, as well as the exhaust from the blowers, compressors, exciters, engines, and circulating pump engines, to the jet condenser. It covers likewise the connections between the exhaust of the dry vacuum pumps, exciters, engines, surface condenser circulating pumps and the heater house. The pipe ranged in size from 3 in. to 42 in.

Three 42 by 13 Wainwright turbine expansion joints....	\$656.70
Three 20-in. atmospheric relief valves.....	804.50
Three 42-in. low-pressure flanged base elbows.....	1,428.61
Three special 8-in. emergency stop valves.....	234.36
One 14-in. automatic atmospheric exhaust relief valve...	123.27
Pipe and fittings.....	4,585.74
Miscellaneous.....	882.48
	\$8,715.66

Account 9015.01—Exhaust Pipe—Painting.

All exhaust pipe was given one coat of green silica graphite paint.

Account 9015.05—Exhaust Pipe—Covering and Erection.

The exhaust pipe from the engines in the power house to the heater house were all covered with 85 per cent. magnesia single standard thickness, wrapped in 6-oz. duck. Where the magnesia is exposed to the weather, it is wrapped with No. 28 galvanized iron. This account covers the labor and material incident to the above work. (See Fig. 62.)

Account 9015.10—Air Piping—Cost and Erection.

This is not a valuable cost.

Account 9015.11—Air Pipe—Painting.

This covers the painting of 9015.01. It is of no value.

Account 9015.2—Exhaust Pipe—Foundation.

This covers a number of small concrete piers.

Account 9015.21—Exhaust Pipe—Supporting Structure.

This account covers the cost of a number of exhaust pipe hangers, supports and staging used in erecting exhaust pipe. It is not a good cost.

Account 9015.22—Exhaust Pipe—Excavation.**Account 9016—Water Pipe—Excavation and Backfill.**

This account covers the excavation of a trench about 3 ft. deep, through red clay and boulders, for a 16-in. wood stave pipe and the backfill after the laying of the pipe.

Account 9016.01—Water Pipe—Cost and Erection.

This account covers the cost of all the material listed below and the labor of its installation. Here is listed all the water piping about the power house the 30-in. cast-iron suction pipe line from the cooling tower to the pumps; the 20-in. wooden lines from the pumps to the equalizing tank; the 16-in. wooden line from the jet condenser to the cooling tower, and the 16-in. wooden lines from the equalizing tank to the cooling tower; the 12 in. cast-iron lines from the circulating pumps to the jet condenser; the 8-in. line from condenser to condensed water pump house; the 6-in. line from condensed pump house to heater house, etc. (See Fig. 63.)

1998.7, 4-in. machine banded redwood pipe with collars. (Not used at New Smelter).....	\$397.74
354.6 ft. 20-in. machine banded redwood pipe with collars	365.24
1,104.2 ft. 16-in. machine banded redwood pipe with collars.	861.28
22 flanged couplings.....	590.00
Freight on the above items.....	632.00
Two 12-in. check valves.....	97.00
Four 12-in. gate valves.....	172.00
Three 20-in. gate valves.....	283.50
Freight on the above items.....	176.38
Three 20-in. flanged, iron body, bronze mounted, double gate valves...	403.49
Five No. 20 gauge copper plates.....	36.28
Two cast-iron bell and flange fittings, 6 bell bends.....	81.11
Freight and patterns on above.....	78.00
220 lb. cloth insertion packing.....	91.50
Best Mfg. Co. pipe.....	9,668.92
Pipe, fittings, misc'l material	2,503.44
	\$16,437.88

Account 9016.02—Water Pipe—Painting.

This covers the painting of the pipe in 9016.01 that was above the ground.

Feed Water Heating Plant**Account 9017—Excavation.**

This covers the excavation of the sump and piers, of the feed water heating plant. The material was red clay, with boulders and sand and gravel. The work was done with pick and shovel, cast into carts and hauled 300 ft.

Account 9017.01—Foundation.

This account covers some miscellaneous concrete cast as piers, low reinforced walls, 4 in. to 8 in. thick, and floor slabs. The walls and slab were reinforced with $\frac{3}{8}$ -in. rods on 6-in. centers. The concrete was hand mixed in the proportion of 6 sand and gravel to 1 cement and wheeled 50 ft. in barrow to the forms. Seventy-five per cent. of the vertical surface was formed.

Account 9017.015—Reinforced Floors and Stair Treads.

These floors were cast over I-beams in the building structure by using forms which allowed the encasing of the I-beams with concrete. The stairs were structural steel with channel treads. The treads were filled with concrete. The floor and the treads were reinforced with $\frac{1}{4}$ -, $\frac{3}{8}$ - and $\frac{1}{2}$ -in. rods variously used and spaced. The concrete was hand mixed in the proportions of 5 sand and gravel to 1 cement, hoisted from the ground 20 ft. to place by means of a small air hoist. (See Fig. 65.)

Account 9017.016—Waterproofing Concrete Tanks.

This consisted of plastering the inside of the concrete tanks at the heater house with the following mixture. One cubic foot cement, 2.5 cu. ft. sand, 1 lb. Medusa waterproof compound mixed with sufficient water. The account covers the labor and material.

Account 9017.02—Steel Structure.

There were 26.63 tons of structural steel in this building.

Account 9017.021—Distribution and Equalizing Tank.

This account covers the material labor of fabrication and erection of this tank. Its dimensions are 13 ft. by 13 ft. by 5 ft. high. It has 8 holes in the bottom averaging 20 in. in diameter. It is located on a concrete floor base 23 ft. off the ground. The weight of the tank is 4,800 in. A material list shows the following. (See Fig. 69.)

17 sheets steel $\frac{1}{4}$ in. by 48 in. by 120 in.....	\$192.96
4 pieces angle iron 3 in. by 3 in. by $\frac{3}{8}$ in. by 20 ft. long.....	15.15
1,110 b.m. lumber.....	31.55
Rivets, bolts, nuts, etc.....	20.89

\$260.58

Account 9017.03—Tile Work.

The walls were started on the concrete floor of the feed water heating plant at an elevation of 26 ft. from the ground. The material was hoisted up by means of a small air hoist. In other respects this account is similar to the tile work elsewhere about the plant.

Account 9017.031—Unloading Tile.

Same as 9003.11.

Account 9017.032—Coping.

Same as account 8703.22.

Account 9017.033—Sills and Lintels.

The concrete sills and lintels for this building were separately moulded as 8821.3, which see. The sills are 3 in. by 8 in. by 3 ft. 9 in. and the lintels 4 in. by 8 in. by 3 ft.

Account 9017.034—Painting Tile Walls.

The tile walls were given one coat of Bay State cement paint after the small holes had been filled and mortar cleaned from the face of the brick.

Account 9017.035—Doors, Windows and Frames.

This account covers the cost of the doors, windows and frames set in place. A segregated material list is as follows. (See Fig. 70.)

3 doors 2 ft. 10 in. by 6 ft. 10 in. by 1 $\frac{1}{2}$ in. with frames	\$20.86
7 windows, 2 ft. 9 $\frac{1}{2}$ in. by 5 ft. 9 in. by 1 $\frac{1}{2}$ in. with frames	59.88
12 sets Caldwell sash balances, nails, locks, etc.....	19.24
	\$99.98

Account 9017.04—Roofing.

This account covers the material listed below, and the labor of installing the same. The wood sheathing was nailed directly to nailing strips attached to the purlins. The paper was laid over the sheathing. (See Fig. 72.)

Two 166 b.m. 2 by 8 Oregon pine S1S1E.....	\$47.60
10 squares asbestos roofing.....	49.00
Miscellaneous.....	13.28
	\$109.88

Account 9017.045—Ventilators.

This account covers the material used to make two 3 ft. 6 in. diameter ventilators, and the labor of installing one. The ventilators were made of No. 18 gauge galvanized iron. (See Fig. 71.)

Account 9017.05—Treating Tank—Concrete.

This is a concrete tank 8 ft. in diameter by 31 ft. 3 in. high, with a conical bottom, the tank is set directly upon the ground. The concrete

was mixed by hand 5 parts sand and gravel to 1 cement, and hoisted to place. The wall of the tank is 5 in. thick suitably reinforced in both directions with $\frac{5}{8}$ -in. rods. Two sets of forms were used each 3 ft. 3 in. high. This cost covers all the material and labor of building this tank save the waterproofing in 9017.016. (See Fig. 68.)

Account 9017.06—Receiving Tank No. 1.

This is a concrete tank 6 ft. 6 in. high with 5-in. walls, suitably reinforced in both directions. The tank rests on a concrete floor about 23 ft. from the ground. The concrete details are the same practically as 9017.05. (See Fig. 67.)

Account 9017.07—Receiving Tank No. 2.

This concrete tank is 12 ft. in diameter, 6 ft. high, with 5-in. reinforced wall, supported on reinforced concrete columns 7 ft. high. The column are here included. In other respects the tank account is similar to 9017.06. (See Fig. 66.)

Account 9017.08—Calibrating Tank.

This account covers the material, fabrication, and cost of erecting the following tank:

1 tank 54 in. high, 6 ft. diameter, of $3\frac{3}{8}$ -in. plate.

Account 9017.081—Tipping Meter.

This account covers the cost of material as listed below and the fabrication and erection of the same. The tipping meter is set directly over No. 2 receiving tank, in a wood frame attached to the top of the tank. The meter is shown in sketch No. 136.

Lumber, 2 by 4's, 6 by 8's, etc., 876 b.m.....	\$24.47
Galvanized iron No. 10 and No. 12.....	112.03
2 by 2 by $\frac{1}{2}$ angle iron.....	3.69
One 6 wheel revolution counter.....	34.74
Bolts, washers, nails, etc.....	52.57

\$227.50

Account 9017.09—Heater and Recorder.

This account covers the material listed below and the labor of installing it on the concrete floor of the feed water heating plant, 26 ft. off the ground. The magnesia blocks are the covering for the heater.

	Factory	Freight	Clifton
1 No. 760 Cochrane feed water heater and receiver	\$776.75	\$204.48	\$981.23
1 Wainwright closed feed water heater.....	775.00	74.64	849.64
One 6-in. Lea recorder and extra float valve.....	589.00	123.51	712.51
One 6-in. float for receiving tank No. 2.....			44.60
600 sq. ft. magnesia blocking $1\frac{1}{2}$ in. thick.....			161.03
300 lb. magnesia cement.....			22.53
Miscellaneous.....			42.30

\$2,813.84

Account 9017.1—Sewer Excavation and Backfill.

This was a long trench about 3 ft. deep through red clay and boulders. Both excavation and backfill are here taken care of.

Account 9017.11—Sewer Pipe—Cost and Laying.

This covers the cost of 100 ft. of 24-in. vitrified sewer pipe, cement and miscellaneous material, together with the labor of laying the same.

Account 9017.1.2—Lighting.

This covers the cost of wiring for lights in the feed water heating plant.

Account 9017.13—Painting.

This covers the cost of painting the underside of the roof, doors, all frames, and window sash. It was done with two coats of white lead and linseed oil, cream color.

Account 9017.14—Wood Walkway and Tank Covers.

This covers the cost of material and its installation as listed below:

Lumber 1,564 ft. b.m.	{	tank cover 1 by 6 O.P. S1S1E rails 2 by 3 and 3 by 3 S4S sills, 2 by 8 ROP 20 in. centers on tank walkways 2 by 8 O.P. S1S1E
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Account 9017.115—Alterations.**Condensed Water Pump House****Account 9017.20—Excavation.**

This was a side hill cut. It includes a backfill made later inside the building foundation for a concrete floor base. The excavation was in red clay, with boulders and sand and gravel. It was done with pick, shovels and wheelbarrows.

Account 9017.21—Foundation.

This concrete was cast as the walls of some square tanks. It was reinforced, mixed by hand in the proportions of 5 sand and gravel to 1 cement. One hundred per cent. of the vertical surface was formed. (See Figs. 73 and 74.)

Account 9017.2—Floor.

This concrete covers the small building walls, large side hill wall, floor slab and tank bottom. Three-eighth inch and three-fourth inch rods were used as reinforcing. One hundred per cent. of the vertical surface was formed. Hand-mixed concrete, 5 sand and gravel to 1 cement, was used.

Account 9017.24—Doors, Windows and Frames.

This account covers the cost of material below with its installation

2 windows 2 ft. 9½ in. by 1½ in. glazed, with factory ribbed glass.	\$6.51
2 frames for above.....	9.28
1 door 2 ft. 10 in. by 6 ft. 10 in. by 1½ in.....	3.02
1 door frame.....	3.87

\$22.68

Account 9017.242—Tile Work.

This covers the cost of tile, mortar, mason and carpenter labor and hauling. The tile was handled 400 ft.

Account 9017.243—Coping.

Same as 8703.22.

Account 9017.25—Roof.

This account covers the material entering into the roof frame, as well as the sheathing paper and labor of installing the same. The rafters were 2 by 10, with 2 by 8 sheathing tongued and grooved, covered with asbestos paper. (See Fig. 75.)

984 ft. b.m. lumber.....	\$33.49
5 rolls asbestos composition roofing paper.....	24.50
Nails, etc.....	2.84

\$60.83

Account 9017.26—Pumps and Piping.

This account covers the purchase price of the material listed below and the entire labor cost required in installing the same.

2 Goulds No. 2½ single stage, single side suction, centrifugal pumps, arranged for direct connection, capacity of pump 200 gallons per minute, against 80 ft. head.....	} \$530.81
Two 10-h.p. induction motors, 1,710 r.p.m., with starters.....	
Grouting, pipes, fittings, valves, etc.....	160.81

\$691.62

Account 9017.27—Lighting.

This covers the lighting of the condensed water pump house.

Power House Miscellaneous Accounts**Account 9018.1—Power and Lighting Transformers.**

This account covers the material as listed below, with the labor charge of installing the same, the labor of unloading, erecting, connecting up, drying out, and testing the transformers noted. These are located in the basement of the power house.

2 Burke air break switches.....	\$134.97
Four 200-k.v.a. transformers, 13,200-volt, 440-volt.....	2,999.80
Two 20-k.v.a. transformers, 440-volt, 220-volt.....	304.40
One 20-k.v.a. transformer, 440-volt, 110-volt.....	152.18
Freight on transformers.....	90.48
Wire, conduit, electrical material.....	614.04
Transformer oil.....	282.34
Miscellaneous.....	136.12

\$4,714.33

Account 9019—Lighting.

This account covers the material and labor of installing the lighting in the power house. The work was all in conduit.

Forty-eight 250-watt tungstens with reflector.

Forty-six 16-c.p. carbon lamps.

610 ft. 1½-in. conduit with weatherproof wire No. 12.

1,710 ft.- 1-in. conduit with No. 12 weatherproof wire.

25 ft. brewery cord.

Account 9020—Power House Oiling System.

This covers the cost and labor of installing the power house oiling system. Two 60-gallon feed tanks are so located that the oil is piped to all the engines by gravity. The waste flows to a water separator, thence through filters into a closed tank, from whence by air pressure it is forced up to the feed tanks.

Account 9021—Benches, Bolt Racks, Etc.

Under this account were built a number of benches, a rack for bolts, a telephone booth, wrench board, and a number of miscellaneous carpenter jobs about the power house including the replacing of about 50 broken panes of glass. The material account stands as follows:

Ceiling lumber.....	\$8.23
Common lumber.....	31.24
Grass.....	6.39
Miscellaneous.....	5.94

\$51.80

Account 9022—Instruments and Gauges.

This account covers the purchase price of the following material, and the labor of installing the same:

3 Tagliabue vacuum gauges.....	\$92.58
10 Tagliabue thermometers.....	92.85
1 vacuum gauge.....	30.86
1 Bristol pressure gauge.....	25.30
1 Bristol gauge.....	22.95
1 Bristol gauge.....	21.35
1 clock.....	46.63
Miscellaneous.....	10.54

\$343.06

Cooling Tower

Account 9050—Excavation.

This excavation entailed the making of a surface cut and two long trenches. The material encountered was red clay filled with caliche. It was done with pick and shovel and handled in carts and wheelbarrows.

Account 9050.01 Backfill.

This backfill was made to bring the ground up to proper level for the cooling tower floor. Fresnoes and scrapers were used to transport the dirt which was tamped in 4-in. layers.

Account 9051—Foundations—Sumps and Gutters.

This concrete was cast as a large number of piers about 1 ft. by 1 ft. by 4 ft. as a sump 35 ft. by 10 ft. by 13 ft. with reinforced walls 8 in. thick, and as two gutters 626 ft. long, in cross section about 3 ft. by 3 ft. having 4-in. reinforced concrete walls. The concrete was hand mixed; for the piers 7 sand and gravel to 1 cement; for the gutters and sump, 5 sand and gravel to 1 cement. In the sump $\frac{3}{4}$ - and $\frac{5}{8}$ -in. rods were used and in the gutters $\frac{1}{2}$ -in. The vertical surfaces of all the above were 100 per cent. formed. (See Fig. 107.)

Account 9051.02—Floor.

Between the gutters of the cooling tower is a reinforced concrete slab about 28 ft. 6 in. by 600 ft. by 4 in. thick. Clinton wire cloth 86 in. wide with a 4 in. by 4 in. mesh was used. No top finish was used but the concrete was straight edged and troweled. It was mixed by hand, 5 sand and gravel to 1 cement, and wheeled 150 ft. to place, in barrows. (See Fig. 107.)

Account 9051.03—Waterproofing Concrete.

See account 9017.016.

Account 9052—Woodwork.

The tower is built of wood and is 626 ft. long 35 ft. 6 in. wide at the base, by 20 ft. 6 in. high. Sketch No. 76 shows the design in other particulars.

Account 9053—Alterations.

Oil Supply Sump and Pump House

Account 9060—Excavation.

This excavation consisted of two deep cuts through red clay and boulders into sand and gravel. The work was done with plow, slips, picks and shovels.

Account 9060.01—Concrete.

This covers the making of a covered reinforced concrete sump, the general dimensions of which are 5 ft. by 10 ft. by 140 ft. with walls 8 in. thick—top and bottom slab $5\frac{1}{2}$ in. thick, also a pump house about 20 ft. by 20 ft. by 20 ft. high, with walls 10 in. at top and 20 in. at bottom. The walls were reinforced with $\frac{3}{4}$ -in. rods and the sump with $\frac{1}{2}$ -in. rods properly placed. The concrete was machine mixed in the proportions of 5 sand and gravel to 1 cement, wheeled in barrows an average of 125 ft. About 80 per cent. of the vertical surface was formed. (See Fig. 37.)

Account 9060.02—Pumps.

This account covers the cost of the following material and its erection in the pump house.

Two 5 by 8 Aldrich vertical triplex, single-acting pumps, 37 r.p.m. with metallic packing.....	\$1,597.91
Two 10-h.p. induction motors, squirrel-cage, 3-phase, 60-cycle, 440-volt, 850-r.p.m.....	287.22
2 auto starters	
2 overload releases calibrated from 6 to 18 ampere per terminal.....	124.36
Miscellaneous.....	26.09
	\$2,035.58

Account 9060.03—Inlet Piping to Sump.

This covers the cost of the following material and its installation between unloading tracks and the oil sump. (See Fig. 38.)

Six 10-in. wrought pipes 18 ft. long.....	\$85.54
Six 10-in. cast-iron cells.....	38.64
Miscellaneous.....	2.37
	\$126.55

Account 9060.44—Lighting.

This covers the cost of material and the labor of installing four drop lights in the pump house.

Account 9060.05—Roof Steel Work.

This covers the cost of the following material and the labor of installing it. (See Fig. 39.)

1,876 lb. 10-in. I-beams
320 lb. 8-in. I-beams

Account 9060.06—Doors, Windows and Frames.

This covers the cost of the following material and the labor of installing it in the pump house. (See Fig. 41.)

4 sash, 3 ft. by 2 ft. $6\frac{1}{2}$ in. by $1\frac{1}{2}$ in.....	\$17.79
2 sash, 3 ft. by 4 ft. by $1\frac{1}{2}$ in.....	14.05
2 sash, 2 ft. 6 in. by 4 ft. by $1\frac{1}{2}$ in.....	12.13
Lumber for doors and all frames.....	12.69
Hardware.....	8.36

Account 9060.07—Roof.

This covers the cost of the roof material as given below with the labor of placing the same. The "hyrib" was placed upon the steel of 9060.05 wired in place, covered with 2 in. of concrete, 5 sand and gravel to 1 cement, and plastered on top and bottom, with 1 part cement, 3 sand, $\frac{1}{2}$ hydrated lime. Later it was covered with P and B roofing, see 9003.52. (See Fig. 39).

784 ft. No. 24 "hyrib".....	\$77.41
7 squares P and B roofing	48.39
Concrete plaster material.....	36.86
	\$162.66

Account 9060.075—Ventilators.

This covers the cost of material in the making and placing of two ventilators shown in Fig. 40.

- 1 ventilator is 18 in. diameter, 40 ft. high made of No. 20 galvanized iron, using 130 ft. $\frac{1}{2}$ -in. guy wire.
- 1 ventilator is 18 in. diameter, 11 ft. high, made of No. 20 gauge galvanized iron.

Two 500,000 Gallon Oil Tanks**Account 9060.10—Wrecking and Transportation.**

This account covers the labor and material incident to taking down at Lordsburg, New Mexico, and transporting to Clifton, two 500,000-gal. oil tanks which had been in use there. The labor item is the cost of tearing down and loading these tanks. The material item is the freight on the tanks between Lordsburg and Clifton.

Account 9060.11—Excavation.

This account covers the making of a top slice to prepare the site for the foundation of the two 500,000-gal. oil tanks. It was done with plow, slips, pick and shovel and handled 150 ft.

Account 9060.12—Foundation.

This concrete was cast as two low circular walls, 1 ft. thick and 2 ft. deep, 62 ft. in diameter. It was mixed by machine, 8 parts sand and gravel to 1 cement, reinforced with two $\frac{1}{4}$ -in. rods and hauled to place 150 ft.

Account 9060.13—Erection.

This covers the erection of the two 500,000-gal. tanks, part on force account by the Kansas City Structural Steel Co. and part by the Arizona Copper Co. It likewise covers the cost of giving them one coat of paint, testing them out, and caulking. Here too are the material prices of the ventilators and gauges. The tanks were 60 ft. in diameter and 25 ft. high. (See Fig. 42.)

Account 9060.131—Roof Supports.

This account covers the cost of the material and labor of installing the same, used for supporting the sheathing over the two tanks. This material amounts to 10,000 ft. board measure for the two tanks. (See Figs. 42 and 43.)

Account 9060.132—Sheathing, Lath and Plaster.

Over the roof supports 1-in. sheathing was laid. On this No. 27 painted expanded metal lath was placed and plastered with a mixture of 1 cement to 3 sand, which was waterproofed with hydrated lime. Each tank has a cornice which ventilates by screened openings through its bottom. (See Figs. 42 and 43.)

Account 9060.14—Railroad Grading.

Along each side of the oil sump a large railroad grade was made. This account covers it.

Account 9060.15—Track Laying and Ballasting.

On the grade mentioned in 9060.14 the following track was laid. The account covers the material and labor of laying and ballasting.

802 white oak ties.....	\$852.60
1 No. 9 frog.....	158.39
Miscellaneous track fastenings.....	82.00

\$1,092.99

Account 9060.16—Oil Track Bumpers.

This account covers the labor and material incident to making and putting in place two bumpers at the ends of the tracks on each side of the oil sump. The material is divided thus:

Lumber.....	\$8.52
Iron bolts, nuts, etc.....	39.40

\$47.92

Account 9060.17—Bridges over Wood Pipe.

This covers four wooden bridges entirely buried in the fill over a number of wooden pipes. They were placed here to avoid the constant breaking of the pipes. The excavation and backfill are here included. (See Fig. 44.)

Lumber.....	\$70.02
Bolts, etc.....	17.12

\$87.14

Oil Supply Tanks for Reverberatories and Boilers**Account 9060.20—Excavation.**

This covers a deep cut for a retaining wall. It was in sand and gravel and made with pick and shovel. The material was hauled 300 ft.

Account 9060.21—Foundation.

This covers the concrete in a reinforced wall about 60 ft. long, 8 in. at top, 18 in. at bottom and 16 ft. high; $\frac{1}{2}$ -in. and $\frac{3}{4}$ -in. rods were used for reinforcing. The concrete was machine mixed 5 sand and gravel to 1 cement and wheeled 75 ft. in barrows to place. One hundred per cent. of the vertical surface was formed.

Account 9060.22—Cost and Erection.

This covers the cost of eight 163-bbl. steel oil tanks, with roofs and ventilators, erected on their foundations.

Account 9060.23—Piping.

This covers the material and labor cost for 785 ft. of piping, varying in size from 1 in. to 6 in. diameter. Here are also included the fittings and valves. The piping connects the tanks with the pumps.

Account 9060.40—Piping Excavation.

This includes all the trenching and backfilling incident to the oil lines from the 500,000-gal. oil tanks to the small 163-bbl. tanks. The trenches were 2 ft. wide and about 3 ft. deep on the average.

Account 9060.41—Pipe and Laying.

This account covers the cost of the pipe enumerated below and the labor of laying it.

172 ft.	12-in. wrought-iron pipe
270 ft.	16-in. wrought-iron pipe
850 ft.	8-in. wrought-iron pipe
596 ft.	2½ in. wrought-iron pipe

1,888 ft. total

A 16-in. line runs from oil sump to pump house, also from pump house to storage tanks. The 8-in. line runs from pump house to the 163-bbl. tanks. The 2½-in. line runs from the wilgus oil pumps to each of the reverberatories.

Account 9060.5—Heating Installation.

This account covers the material noted below and the labor required to install the same. This 2½-in. steam line is tapped off the steam line at the power house, run under ground through conduit and is packed in asbestos fiber. At the other end the pipe connects with a cast-iron oil heater.

1 cast-iron oil heater.....	\$303.82
1 No. 33 Crane tilt trap.....	35.91
280 ft. 8-in. conduit.....	547.49
Asbestos.....	29.00
2½-in. pipe, fittings, etc	151.82

\$1,068.04

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Draining Kerr Lake

BY ROBERT LIVERMORE, COBALT, ONT., CANADA

(Salt Lake Meeting, August, 1914)

It has been a noteworthy feature of the Cobalt camp, that many of the valuable ore deposits have been covered, wholly or in part, by small but usually deep lakes, such as Cobalt, Cart, and Peterson lakes, and with the subject of this article, Kerr lake. In the early days of the camp many water-covered areas, which have since been found to be valuable, were ignored, or neglected by owners, or simply perfunctorily staked for water rights. Kerr lake is probably the evidence of a fault or sharp fold, having a general east and west strike, and containing a series of ore-bearing fissures, parallel generally to the strike, although having some notable exceptions to the general rule.

On the north side of Kerr lake the formation is of diabase, while on the south side it is of Huronian slates and conglomerates. These Huronian measures extend out under the lake, dipping gently to the north to a lateral distance not yet definitely determined, owing to the lack of development under the deepest parts of the lake. Probably the contact between the diabase and Huronian or else between the former and the underlying Keewatin formation occurs somewhere near the center of the lake.

Kerr lake originally covered 45 acres. Of this the Kerr Lake Mining Co. owned 12, the Drummond Mine, 7, and the Crown Reserve Mining Co. the remainder of 26 acres. Since the latter's property was originally entirely under water, it was necessary for the owners to make land for buildings and shaft room. Accordingly, in 1908, a trench was blasted out which deepened the outlet, and lowered the lake 8 ft.

As many of the rich veins of the Crown Reserve and Kerr Lake companies were under water, mining was pursued under some disadvantages. Fortunately the rock is tight and solid, and mining has been done sufficiently far from the surface to avoid unnecessary risk. Nevertheless, there was always danger of encountering open seams, through which too large a flow of water for comfortable working might come. Careful soundings were made over the veins, by means of steel-shod pipe, through mud and water to bedrock, but there was sometimes an element of uncertainty as to whether actual bedrock had been reached. Furthermore, the necessity of leaving safe backs between workings and lake tied up large quantities of ore, and made development in certain

was early recognized, and several plans were proposed for its accomplishment, but owing to various difficulties, both of an engineering nature, and of securing agreement between the various companies affected, did not come to fruition.

Crown Reserve and Kerr Lake Companies Undertake Dewatering of Lake

During the summer of 1912, the Crown Reserve and Kerr Lake companies gave serious thought to the subject, and came to the conclusion that if the dewatering were to be done, it could best be accomplished by pumping rather than by an earlier plan of tunneling, both on the score of simplicity and expense. Preliminary surveys were made over the route the water was to follow, viz.—through the Kerr lake outlet, to Glen and Giroux lakes, and thence by the outlet stream of the latter to the Montreal river, a total distance of 8 miles.

In April, 1913, an act of Parliament permitted the draining of such bodies of water as were an obstruction to mining operations, which act removed one of the hindrances hitherto existing to this, and similar plans; and in May, 1913, the purchase of the 7 acres owned by the Drummond interests, under Kerr lake, removed the last vital objection to the undertaking. Permission was granted by the mining commissioner in May, 1913, and work was begun at once.

Kerr lake at this time covered an area of 30.35 acres having been reduced from the original area of 45 acres by the work of the Crown Reserve Co., above mentioned, and by filling in by the waste dumps of the two companies. Of the total area 18.5 acres belonged to the Crown Reserve, -6.54 to Kerr Lake, and 5.31 acres formerly of the Drummond Mine, to the two first-named companies jointly.

There are no inlets of importance in Kerr lake, and its one outlet carried off water running at the rate of 300 gal. per minute in the freshet season to nothing in the dry season. It was not thought that the lake was spring fed to any extent, but that it maintained its level simply from rain and melting snow. Soundings had established its greatest depth at 100 ft., nearly 20 of which was soft mud. It was estimated that the lake contained 400,000,000 gal. of water and liquid mud in all. In any plan for dewatering, the mud had to be taken into account, as a large deposit of this left behind would leave parts of the exposed surface in as bad shape as ever for mining purposes.

Pipe Line Direct from Kerr to Giroux Lake

It had been planned in the first surveys to pump the water through the old outlet, whence it would run through natural channels by Glen and

simple and feasible plan for the water only, the problem involved by the proper disposal of the mud prevented its adoption. Hence the final surveys were run in a direct line from Kerr to Giroux lake. A pipe was to take the water over this line, crossing the Kerr Lake property, the township highway, the tracks of the Temiskaming & Northern Ontario Railway, and several rights of way of power and compressed-air lines, before entering Giroux lake. The greatest elevation of the line above Kerr lake was 53 ft., and the linear distance from lake to lake, 2,400 ft. The difference in elevation between the two lakes was 20 ft. Giroux lake covers about 230 acres, and is of great depth, with an ample outlet, so that the disadvantages and objections of possible blocking of channels and flooding of other properties met with in the original plan, were done away with.

The problem which then presented itself was the installation of a pumping plant capable of handling both water and an indeterminate amount of solids against a static head varying from 53 ft. at the start to 153 ft. at the end of operations, when the lake should have been completely drained, through a pipe line large enough to eliminate the friction factor as much as possible, yet small enough to give sufficient velocity to mud-laden water. Allowing an ample friction factor, it was estimated that the total static and dynamic head would approximate 185 ft. The pumping plant, further, must have a variable base since the shore line would be constantly changing, and the pumps must have a great range of action to maintain an even flow of liquid of changing density against an ever increasing head.

Pumps Placed on Scow

On account of the steep and irregular shore and bottom of the lake, and the cumbersome nature of the machinery, a plant mounted on a movable base ashore presented obvious disadvantages, so it was decided to place the pumps upon a scow. This scow was to be kept near the shore because if anchored in mid-lake a good many pontoons would have been necessary to support the heavy pipe; also, since flexibility of the pipe line at some point was essential, on account of the changing level, it would have been difficult to anchor both scow and pontoons so as to be flexible enough and yet not too susceptible to wind and movement of the waters. Further, constant changing of mooring cables at many points would have been necessary as the level of the lake changed.

Since the Kerr Lake, Crown Reserve, and the neighboring Drummond (now Cobalt Comet) mines all took their water supply from Kerr lake, the plans included the erection of an auxiliary water-supply pumping plant on Giroux lake, large enough to fill all needs of the companies concerned.

Machinery and accessories were ordered in May, 1913, and with the receipt of the first shipments of pipe during the latter part of that month active work was begun.



FIG. 1.—SCOW IN POSITION, READY TO PUMP.

The scow on which the pumps were to be mounted was built from designs of the Kerr Lake Mining Co.'s engineer, by the Crown Reserve



FIG. 2.—LAKE LOWERED 10 FT.

Mining Co. at its shops on the shore of the lake. This scow (Figs. 1 and 2) was designed to be as compact as possible, yet to allow plenty of

bility, rigid construction, light draft, and carrying capacity up to 70 tons displacement were all essential, and were taken into account in the plans.

The hull was built of Western fir throughout, dimensions over all being 40 by 20 ft., depth of sides 4 ft., and dimensions of bottom 31 ft. 6 in., allowing a 4 ft. 3 in. overhang at bow and stern. For the bottom, 3-in. planks laid lengthwise were used; across these other 3-in. planks were laid at 2 ft. 3 in. centers as sills for the uprights supporting the deck timbers. The boat was divided longitudinally into three sections; the sides, the center, and the two intermediate divisions of the framework were built of 6 by 6 in. timbers at top and bottom, each division spaced 4 ft. 9 in. apart. The uprights were 6 by 6 in. posts set on the bed planks at 2 ft. 3 in. centers and bolted to the longitudinal timbers. The deck beams, also 6 by 6 in., rested on the latter at 2 ft. 3 in. centers, and were bolted to them and to the supporting posts. Deck and sides were of 3-in. plank laid lengthwise, and spiked and bolted to the frame. A 2-in. flush was given to the deck, and hatches provided fore and aft for entrance to the hold. The whole boat was thoroughly calked with tar and oakum. Two anchoring spuds were placed at each side of the stern, for holding the scow firmly against the thrust of the pumps. These were 16 by 10 in. fir timbers, 30 ft. long, iron shod, and held in place by guides 14 by 10 in., which were tied by iron rods to each other above and below water. The spuds were raised and lowered by rack and pinion with spoke attachment. Beside the spuds $\frac{5}{8}$ -in. wire hawsers were provided, to connect with shore at each corner of the scow. In practice it was found that on account of the shifting nature of the mud, and the difficulty of finding firm bottom for the spuds, the hawsers were more useful, and with the aid of small yacht capstans set up on the boat and on shore, were ample to hold the scow in any desired position.

After launching, which was accomplished without trouble, by jacking up the boat to the proper angle, and placing greased ways beneath, it was towed to the east end of the lake and ballasted with about 10 tons of bagged gravel. A temporary crane was rigged on deck for handling the pumps, motors, and heavier pipe and valve fittings, which were hauled by wagon from the cars, and shipped on board at this point. The pumps and motors were assembled on their base, the bed plates of each unit bolted through 8-in. sills to the deck, and the pipe and valve connections between the pumps made. The scow was then towed back to the yard for housing and final fitting. A weather-tight house of light construction was erected, to cover all of the deck except a small space at the end, where room was needed for operation of the spuds and capstans.

The plant consists of four single-stage centrifugal pumps, arranged in two units (Figs. 3 and 4). Each unit comprises a compound pumping outfit of two pumps, direct connected by a flexible coupling to the motor which is placed between the two. These have 12-in. side suctions leading outboard, and 10-in. discharges facing inboard. The pump shells are $1\frac{1}{2}$ in. in thickness, with removable side disks. The runners are of

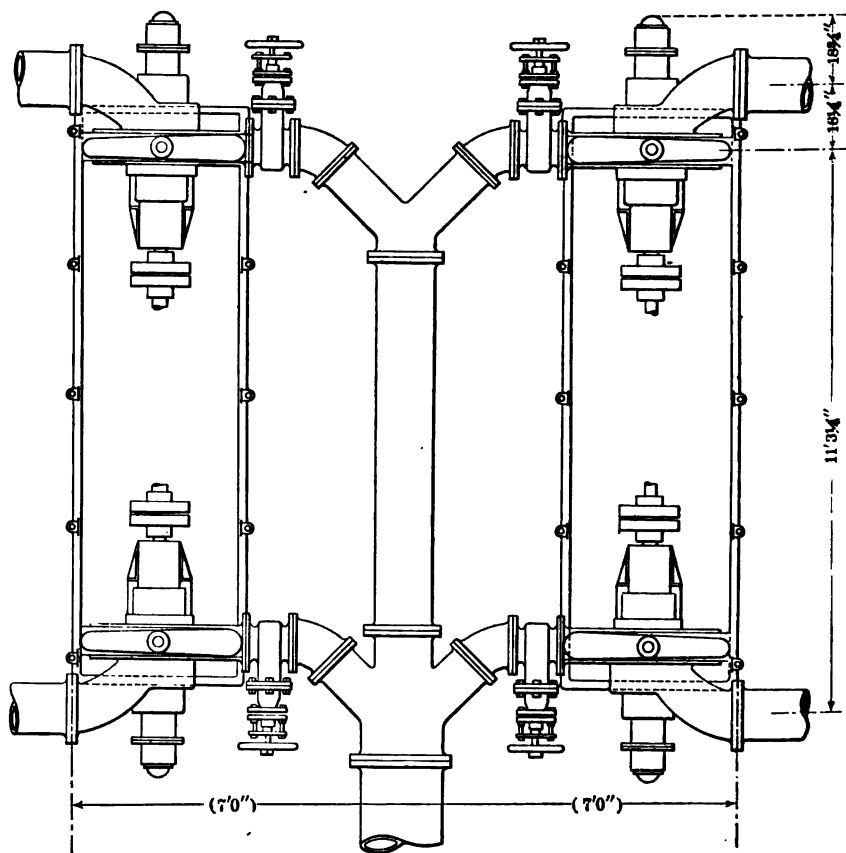


FIG. 3.—PLAN OF PIPING FOR PARALLEL OPERATION OF PUMPS.

the inclosed type, of heavy design, and capable of delivering solids up to 4 in. in diameter. The shafts are of steel extended on each side of the pumps, and carried by outboard ring oiling bearings, fitted with removable babbitt-lined shells. Adjustable thrust bearings are also provided to take possible unbalanced end thrusts.

Each unit was designed to deliver not less than 3,000 gal. per minute at the greatest elevation encountered during the operation, with a mech-

were to work in parallel (Fig. 3), each delivering through its 10-in. discharge into a central pipe, the flow from the two sternmost pumps carrying through a length of 14-in. pipe until abreast of the forward pumps, when the flow from all four entered the main 20-in. line. All pipe connections on the scow were specially cast for the work. For the parallel connection a flanged Y joined the stern pump discharges to the 14-in.

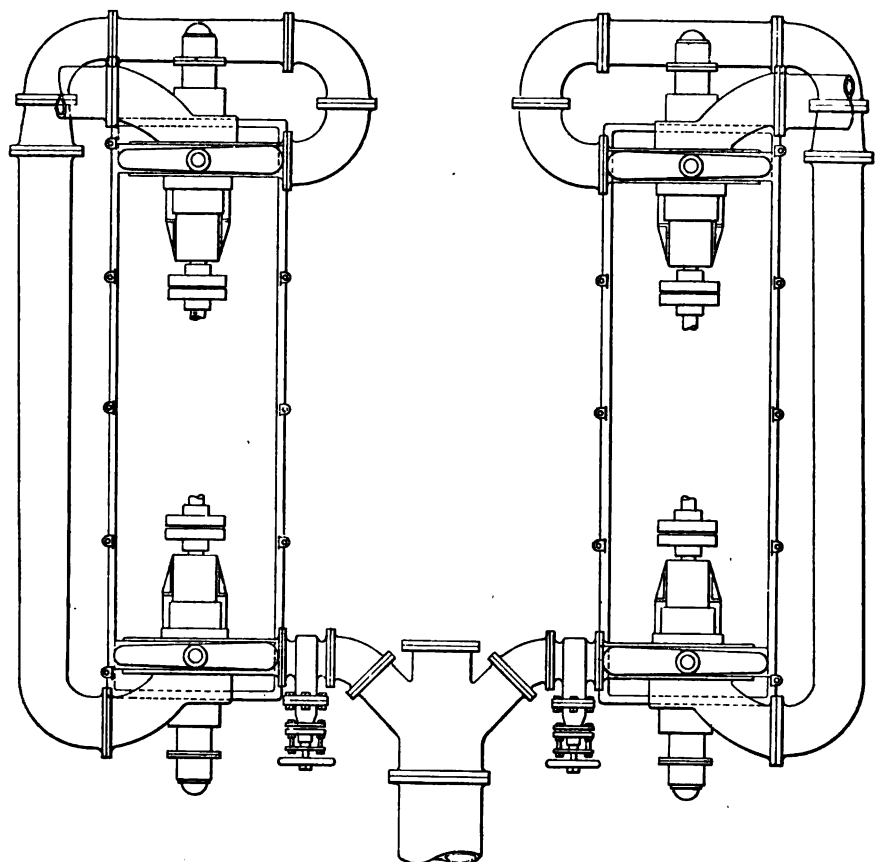


FIG. 4.—PLAN OF PIPING FOR SERIES OPERATION OF PUMPS.

pipe above mentioned, which extended along the center of the scow to another Y bored to receive it, and connecting the forward pumps to the main line. Gate valves were set in between each discharge and the main line, so that by disconnecting the motor and closing the valve any one pump could be closed off without affecting the others.

A change from parallel to series operation (Fig. 4) could be made at whatever point in the undertaking the increase of head and density of

suctions, stern Y, and center length of 14-in. pipe were to be removed. Cast U-shaped lengths of pipe were provided to carry the discharge from the stern pumps outside and around to the intakes of the forward pumps, where a blank flange fitted to the 14-in. orifice in the forward Y threw the water forward to the main line. By this arrangement two units, each of a two-stage tandem pump, were obtained at some sacrifice of volume, but at a decided gain in efficiency.

For priming and keeping pressure on the impeller bearings, a $5\frac{1}{2}$ by $3\frac{1}{2}$ by 5 in. air-driven plunger pump was installed on the scow, which took its power from the mine compressed-air supply, and its water from Kerr lake at the start, but after its waters became too muddy, from the auxiliary supply from Giroux lake. The main suction pipes were connected to the pump shell by 90° elbows, and extended to the water's edge where 8-ft. lengths of smooth-bore suction hose continued the intake. The suctions were fitted with flap foot valves and strainers having 3-in. apertures. The suctions were arranged on loose threads to act as a swivel joint, which, with the aid of a tackle, allowed the intakes to act at any desired depth within an 8-ft. radius.

The motors to drive the pumps are two in number, one for each unit, and connected to the pumps directly, as above described. They are 250-h.p. squirrel-cage synchronous induction motors, operating at 1,200 rev. per minute, with 2,200-volt, three-phase, 60-cycle current.

The power was obtained from the line of the Northern Ontario Light & Power Co., which passes through the property. A 2,200-volt line was carried without transforming, to a switch house built for the purpose, on shore near the scow. Here, oil and knife switches, meters, etc., were installed, whence heavily insulated copper cables were led to the scow. The connections on the scow were made through oil-switch, panel-mounted knife switches, and starting compensators for each motor. Lightning arresters were placed outside the deck-house. Cables inside the house were carried in pipe, and below deck where possible, to the various connections.

Main Line of Spiral Riveted Pipe

A 20-in., 14 gauge, spiral riveted pipe was used for the main drainage line. Bolted steel joints were used, except for a few flanged lengths on angles, and between the scow and shore. This type of joint was used on account of its great flexibility and capacity for taking up expansion and contraction. These qualities were very desirable here, where extreme changes of temperature occur, and where portions of the line, both at the feed and discharge ends, had to be shifted from time to time. This pipe was supplied in 32-ft. lengths, weighing 1,500 lb. to the length. The

keeping it in as straight a line, and as free from hollows as possible. What few angles there were, were calculated in the survey, and were met by specially cast flanged elbows, to which lengths of pipe flanged at one



FIG. 5.—CROSSING OF 20-IN. DRAINAGE LINE OVER T. & N. O. RY. TRACKS.

end were bolted. A little ditch work, and rock drilling and blasting were necessary, especially where the line crossed under the main highway by culvert, but in the main, inequalities of the ground were made up by the bents. The chief engineering difficulty met with was in carrying the

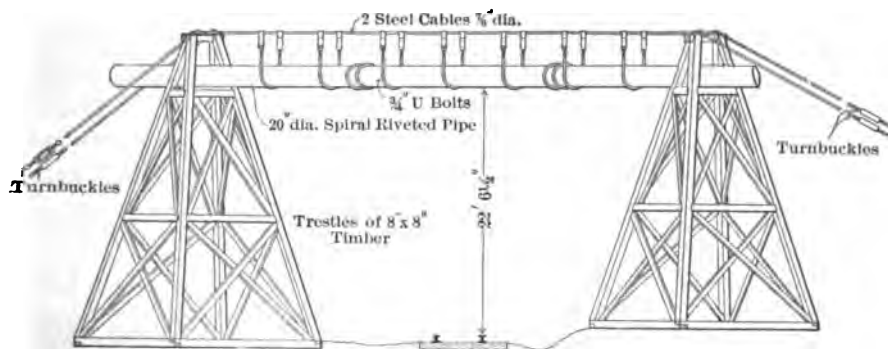


FIG. 6.—PIPE AND SUPPORTS CROSSING T. & N. O. RAILWAY.

pipe over the tracks of the Temiskaming & Northern Ontario Ry. at the minimum height allowed above rail, of 22 ft. 6 in., an action made necessary to avoid causing a hollow in the line if carried under the tracks.

At this point (Figs. 5 and 6) two wooden towers of the ordinary tram

type, built of 8 by 8 in. posts on a 16-ft. square base, well tied and cross braced, were erected on each side of the track. The angle of crossing made a span from one support to the other of 90 ft. Saddles for the pipe were provided 4 ft. below the top of the towers. Over the tops, which were iron-shod cross pieces, two 7/8-in. steel-wire cables were passed, 20 in. apart, and carried to ground. These were anchored by "dead-men" made of stout logs sunk to a depth of 6 ft., and weighted with stone. Turnbuckles were provided, by which the cables were stretched as taut as possible.

The pipe was then laid on the bents and joined as far as the first tower. Three flanged lengths were then laid alongside on the bents and bolted together to make the span. One end of the joined lengths was slung in a carrier, made of an iron loop ending in sheaves to run on the two cables, and pushed out over the railway to the far tower. Connection was then made with the pipe already laid. The span of pipe across the railway was supported by 2 by 1/2 in. strap-iron hangers passing under the pipe and hooking over the cables at 8-ft. intervals. Each hanger had a turn screw and nut, to take out all sag in the pipe. An angle flange and two connecting lengths carried the line from the tower to a rock-filled crib pier where another angle flange was connected, and anchored by long eyebolts passing to a "dead-man" to guard against displacement of the pipe by the force of the water coming over the sudden drop from the tower. The continuing line was carried on bents as before across a swamp and down the final incline to Giroux lake.

Flexibility of connection between the floating scow and the fixed pipe on shore was secured as follows: The Kerr lake end of the pipe (see Figs. 1 and 2), which reached shore at an angle of 15°, was anchored by strap and bolts set in a cement pier at water line. The end of the pipe facing the water was flanged, and to this was bolted a 20-in. flexible ball joint having a maximum swing of 27°. To this joint in turn two expansion joints giving a lateral play of 16 in. were bolted. At the start, one 32-ft. length of flanged pipe was connected to these, and to a similar ball joint coupled inboard on the scow at the main discharge. A 12-in. bypass, with a gate valve, was inserted on the flanged length, for draining the line.

Water Supply Pumped from Giroux Lake

The water supply for the different properties formerly taking water from Kerr lake was obtained from Giroux lake, as already mentioned. It was at first planned to mount the pumps on the shore of Giroux lake, and carry the suction intakes out on piers, but the difficulty of placing piers firmly on the steep, smooth bottom, and of protecting the suction pipes against freezing, made a change of plan advisable. A well, 10 ft. square by 12 ft. deep, was sunk by air drills in the solid rock on shore,

almost at water line, leaving a thin shell between lake and well. When the well was completed, a series of flat holes drilled at a depth of 5 ft. below water level and "looking out" under the lake, was put in. The holes were heavily loaded, blasted simultaneously, and a clear connection made between well and lake. A 14 by 20 ft. pump house (Fig. 7) was erected on cement foundations over the well, in which 10 by 10 in. timbers were set at 3-ft. intervals, as sills for the pumps and equipment.

The pumps for this water supply are two in number, of the single-stage turbine type, each capable of delivering 500 imperial gallons per minute against a total head of 175 ft. The speed of operation is 1,750 rev. per minute. The pumps are mounted on the same base plate with a 45-h.p., 2,200-volt, three-phase, 60-cycle motor, directly connected to the pumps with flexible pin coupling. The equipment comprises auto-starters, oil and knife switches, lightning arresters, and meters.



FIG. 7.—DISCHARGE OF 20-IN. DRAINAGE LINE, AND WATER SUPPLY PUMP HOUSE, GIROUX LAKE.

Water is pumped through an 8-in. spiral riveted bolted-joint pipe 1,300 ft. in length, against a 125-ft. head to a 46,000-gal. receiving tank erected on an eminence between the two properties. From this tank the supply for the different mines is delivered in 6-in. wrought-iron pipe, by gravity.

Owing to the severe cold of the winters, great care was necessary in protecting all water lines, and the methods used merit some description. The main 8-in. line was inclosed in a 4-ft. plank box, filled with sawdust and covered on top with paroid. A 1-in. steam line from each of the two mine heating plants was run alongside the pipe to the pump house, where it was run through a coil for heating purposes, and thence to the well

lake from the drainage operations, a filter dam made of jute, supported on spiles, was placed around the inlet, which formed a small pond of comparatively clear water. Two gate filters of 8-oz. duck were placed over the direct inlet, further to clarify the water.

A signal system (Fig. 8) from tank to pump house was installed, in order to keep the water in the receiving tank at a fairly constant level. This consisted of a semi-circular wooden disk revolving on bearings placed in the center of the curve, and supported on brackets fixed to the inside wall of the tank near the top. The disk was equipped with a 10-ft arm and 10-in. cylindrical float. On the edge of the disk were

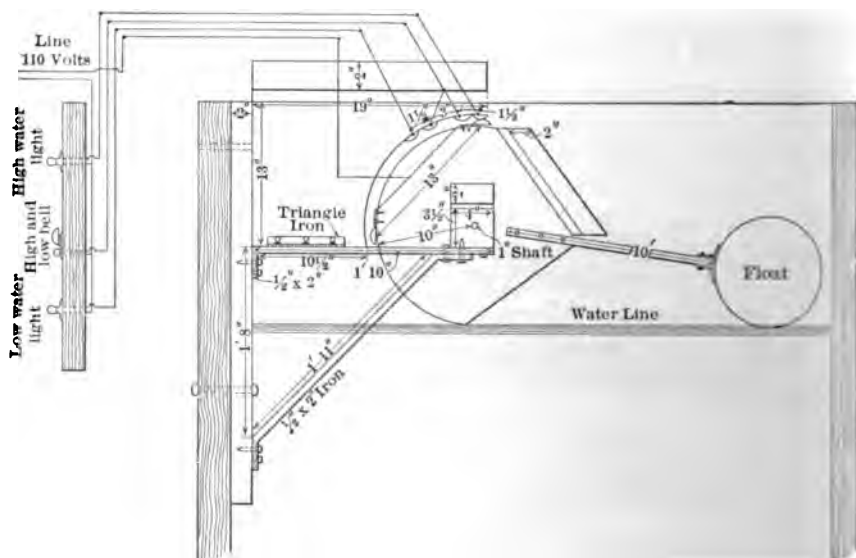


FIG. 8.—DETAILS OF ELECTRIC SIGNAL CONNECTIONS.

placed two electrical contact points one quarter of the circumference apart. Facing the disk edge was placed a stationary block, having a concave surface, matching, and clear of, the disk edge. On the upper part of this surface were placed two pairs of electrical contacts. Two incandescent lamps and an alarm bell were placed in the pump house, arranged on a panel so that the bell had a light above and below. Electrical connection was made by wire from a 110-volt circuit to the two moving contacts on the disk. Circuit was made between the inside contact of each pair on the stationary arm and the alarm bell, and between the outside contacts and the lights. The operation of the signals was now as follows: When the water had lowered sufficiently, the lower moving contact made connection with the lowest fixed contact, and lighted

the warning lamp on the panel, showing "near low water." The water continuing to lower, shortly afterward contact was made with the lower of the inside fixed contacts, and the alarm bell was rung. In the same way, when the water in the tank was nearing high mark, the lamp marked "near high" was lighted and then the bell was rung, only in this case the upper moving contact and pair of fixed contacts were in play. The contacts were spaced on the disk and concave surface so as to obtain these effects at the proper stages.

Drainage Operations

The Kerr lake pumps were started on Aug. 28, 1912. Some difficulties of a temporary nature were met with at the start, but there were, on the whole, few delays or hitches in the operation. It was found that too sudden stopping of the pumps caused vacuums to form which made one or two lengths of pipe show a tendency to collapse, but this was remedied by the insertion of check valves at threatened points. A tendency of the pipe to sag after receiving the full weight of water was observed on the span over the railway, and the structure was strengthened by the addition of two more $\frac{1}{2}$ -in. cables stretched and anchored in the same way as the original ones, and by the construction of arm props on each tower, hung out over the track by $\frac{1}{2}$ -in. rods, and set in notches on the legs, which shortened the unsupported span of pipe by 24 ft.

The water was lowered steadily until the depth under the scow became too shallow for convenience, when a new flanged length of pipe was inserted, and the scow moved out a corresponding distance. The extra length was supported by trestle bents having an adjustable block and tackle sling to allow for the falling level of the pipe. The operation was repeated as often as it became necessary to make a move until the suspended line became too cumbersome, when the shore ball joint was moved out to a new pier and the connecting pipe shortened accordingly.

It had been intended to stir up the mud as much as possible by agitation while there was plenty of water in the lake, so that the mixture flowing through the line might be as liquid as possible. Various methods were tried, such as directing a stream, pumped from the lake by an auxiliary plunger pump of 300 gal. capacity, through a 4-in. hose with monitor attachment, into the mud, both from shore and from a small scow, but it was found impracticable, since the mud was of such consistency that although the bulk of that exposed by the lowering of the water flowed into the deeper part almost without sluicing, that under water had a glue-like tendency to stay in banked masses near the shores as long as any water remained in the lake to hold it back. Such mud as was left above the water soon dried and became compact to a short depth, so that it offered little difficulty to mining or prospecting, but that in the center of the lake constantly increased in depth with the influx from

the sides, and as rapidly diminished the clear water area. Tests made on this mud showed that its water content was very high, nearly 80 per cent., so that it was decided to pump out the remaining water entirely, and with the suctions resting in the mud, trust to the pumps handling it with the aid of the small streams of water flowing into the basin from the mines and mills. The change of the pumps from parallel to tandem operation, provided for as above described, was made in order to obtain better efficiency with the heavier material, and after some experimenting with the proper mixture of mud and water, and with the size of the strainer openings, a fairly steady stream of liquid mud varying from 6 to 20 per cent. solids was maintained through the line.

Some trouble was had from the muddying of the water in Giroux lake near the mine supply pumps. This was met by carrying the pipe line on a curve of 45°, to which the flexible nature of the bolted joints adapted it admirably, to a wooden flume, built along shore, and extending to a remote cove of the lake, after which no more trouble was experienced.

At the time of the first proposals for the draining it had been feared that the health of the people in the rather thickly settled vicinity of Kerr lake might suffer, but fortunately these fears had no justification; in fact if anything the reverse was the case, since the more than doubtful water supply of Kerr lake was replaced by the purer water of Giroux. At one period when most of the clear water had been pumped away, some embarrassment was caused by the large number of fish which had been smothered by the mud and came to the surface. Great numbers ranging in size from small perch to eels and pike 30 in. long lay everywhere, and even clogged the suctions and entered the valves. Fortunately, the situation was well taken care of by the gathering of great numbers of scavenging gulls who soon disposed of the fish, to the mutual advantage of themselves and the operators.

The pumps were run through September and October, and at intervals during the month of November, but during the latter month the increasing cold made operation difficult because of the freezing of valves and of the surface of the mud, so that work was stopped for the season the last of the month. The operations to this point were successful from the standpoint of the companies concerned. The water was removed from the greater part of the important reserves of the mines, thus leaving them free for stoping and development. About 325,000,000 gal. of mud and water were pumped, at an average of 6,000 gal. per minute for 38 actual working days.

It was demonstrated that the liquid mud lying in the deeper parts of the lake could be pumped, and that eventually the ground underlying these deeper parts could be prospected and mined at will after the removal of the remaining mud in the following open season.

which had not been encountered in the underground workings, and some of which although previously known and followed, were not known to be so valuable as their surface appearance later showed them to be.

One of the most important questions decided was that the long immersion of the veins had had no ill effect on their value, as exposure showed that for the most part they had suffered no oxidation or leaching of valuable minerals. In one curious instance the glacial striations in a native silver outcrop appeared as burnished and bright as if made yesterday.

From an engineering standpoint there were few miscalculations or setbacks to the work, and allowing for the usual delays in transportation and delivery of ordered equipment, the work of installation was expeditious.

The managements of the two companies, under whom the work was conducted jointly, owe much credit to their respective staffs and other employees, to whose harmonious and efficient co-operation, and many ingenious ideas in perfecting details, the assured success of a somewhat novel undertaking is largely due.

Character of Title That Should be Granted by Government

BY GEORGE W. RITER, SALT LAKE CITY, UTAH

(New York Meeting, February, 1914)

OUR mineral-land laws need revising so as to provide definite title at the outset to the mineral deposits within any definite piece of land. The laws as they now stand, especially those applicable to lodes, breed uncertainty as to rights of possession prior to patent; and to my mind this is the worst feature of all, worse even than the apex bugbear.

It is now almost impossible for anybody to tell with certainty what mineral rights exist within any area that is not covered by patent, or whether any valid rights exist at all. Our system is unique in permitting individuals to appropriate and exploit mineral lands of the public domain without notice of any kind to the Federal government, and to relinquish them without any ceremony at all. The result is a clouded record that cannot be cleared up until each claimant is put to some sort of final proof. In order to accomplish this, new Federal legislation will be required. The requirements for a mineral survey and the proceedings for patent have always been cumbersome and expensive; and perhaps out of regard for the poor claim owner, who can ill afford the outlay for that sort of thing, the law does not require him to go through it unless he chooses to do so. Consequently, every active mining district has become encumbered with tangled groups of doubtful or conflicting locations, whose rights and boundaries, when any valuable ore is found, must usually be determined by lawsuits that are as expensive and as distressing as the stakes are high. So deeply has litigation entered into our mining system that a claim whose rights have never been disputed would probably be set down offhand, by a miner of the old school, as of no value. Think of the litigation over rights of possession on the Comstock lode—12 leading companies involved in an aggregate of 245 suits within a period of less than eight years!

Ambiguous local records; the chance of erroneously placed or obliterated monuments; doubt as to the sufficient performance of the statutory annual labor; uncertainty as to prior conflicting or overlapping locations; anxiety as to the changing attitude of the Department of the Interior and the Forest Service—such considerations as these make an unpatented mining claim an unclean thing in the eyes of possible investors. Is it any wonder that they keep themselves and their money at a safe distance,

and that prospecting for new ore deposits has become decadent for want of the funds necessary to carry mining ventures through the period of infancy?

The uncertainty as to rights of possession is not restricted to lode claims, but extends to placers, including lands appropriated for petroleum. The Department of the Interior has itself been guilty of much floundering in efforts to apply the law to lands of this latter class. The speculative spirit is easily aroused by signs of the existence of oil within a new district, and, since there is no such thing even as a preferential right within a legal subdivision of land until oil has actually been developed, rival speculators may often be found drilling wells alongside of one another in headlong haste to gain the first right to make a legal appropriation. This practice is not only wasteful, but it tends toward monopoly, since a race of this kind is necessarily to the swift and strong.

There seems to be no good reason why the present requirement as to discovery before location should not be done away with frankly, or why all mineral lands, including those containing lodes, should not be located and entered in units conforming to the public-land surveys, without requiring further survey or plat. The law already provides that for purposes of a placer claim a legal subdivision may be further subdivided into 10-acre tracts, so that a claim may be made up of strings or groups of 10-acre units, no further survey or plat being necessary.

There seems to be no good reason why the lands already appropriated for mining purposes should not be officially surveyed without delay, and the claimants required to proceed to patent.

There seems to be no good reason why the public-land surveys should not be extended at once over all mineral lands of the public domain, or why the respective Federal land offices should not be made the places of record for all new mineral locations. If appropriators were given the option of having an official survey made at the outset or of making the boundaries of a location conform to the public-land surveys, I believe that locations by legal subdivisions would soon become popular. There seems to be no good reason why such record in the Federal land office should not take the form of a mineral entry, so that the claimant might always know that he had at least preferential rights to the land, and so that other citizens could tell from consultation of the records exactly what lands were unappropriated.

Opinions may differ as to whether a mineral patent ought to be a final relinquishment of government ownership, as under our present system; or whether the title ought to revert to the government upon failure of the grantee to pay an area tax at stated intervals, as in Mexico; or whether some new system, such as leasing under government ownership and control, ought to prevail.

The Mexican system has several features to commend it. The filing

mineral land by triflers. And while to the *bona fide* miner these taxes are more expensive than are the burdens imposed by our own laws, he generally prefers the Mexican system because his grant is practically incontestable except for non-payment of taxes, and is so definite from the very outset that he and his grantees always know and his neighbors can know exactly what the grant covers and what it does not.

I hardly think many of us ever expect to see our Federal government participating permanently in the development and operation of mines. Such a system would not only be contrary to our national traditions, but commercial success could hardly be expected in competition with private enterprise. No matter how conscientious and efficient the several department chiefs and their subordinates may be, their activities are limited so strictly by the terms of each appropriation bill that they are handicapped in comparison with private concerns. A simple instance may illustrate this point. The surveys of the public lands call for the preparation of several copies of each plat, and until recently these copies were all made by the laborious long-hand process. After it had been demonstrated to the Commissioner of the General Land Office that the duplication of these plats by some mechanical or photographic process would tend toward both economy and accuracy, he was unable to change the method at once because the appropriation bill permitted the employment of draftsmen but did not provide for the purchase of mechanical apparatus.

Our form of congressional government is also a valid argument against the leasing of mineral lands. What is there in the government's past treatment of the mineral-land question to inspire enthusiasm over a system of direct Federal control? The dilly-dallying over the surveying of the public domain; the willingness for more than 40 years to continue disposing of the mineral lands under makeshift laws; the floundering on the question of lands containing petroleum; the hit-and-miss classification of the lands given away as subsidies to new railroads, the vacillation and recrimination over the coal lands of Alaska—with these bits of history fresh in our minds, how many of us would care to gamble our possessions against the whims of ephemeral law makers and transitory office holders? Is it not enough to be obliged to gamble against the whims of nature?

Congress acts only slowly, and then only in matters upon which the people have shown deep feeling. I think myself that the reason why we have made so little headway in obtaining rational legislation on the details of our mineral-land law, is that the people have been satisfied with the basic principle of the existing law, which declares that the mineral lands of the public domain shall be free and open to occupancy and exploration by citizens or intended citizens, and that the mineral

and that have shown good faith in developing them. Real property with strings to the title has never been popular in this country. Our forefathers had seen enough of leaseholds, restricted estates, entails, and that sort of thing in the old world; and when the opportunity came to them to start a new order of things, they were quick to cut away from forms of title other than in fee simple. Perhaps it is the dread of chance return to an old-world system that "puzzles the will, and makes us rather bear those ills we have than fly to others that we know not of."

Nodulizing Blast-Furnace Flue Dust

BY LAWRENCE ADDICKS, DOUGLAS, ARIZ.

(Salt Lake Meeting, August, 1914)

SOME three years ago the smelter connected with the Chrome, N. J., refinery of the United States Metals Refining Co. found itself embarrassed by constantly increasing piles of unsmelted blast-furnace flue dust. The charge was fine and 10 to 15 per cent. of it was blown over into the flue. Sulphur was at that time too scarce to make the sintering of the finer part of the charge attractive, and raw-ore smelting in a reverberatory with a partly oxidized charge was not to be thought of.

A great many schemes were considered, including leaching, blowing into the converters, and briquetting, the problem finally narrowing down to blast roasting and nodulizing. It was found by experiment that blast roasting would not yield a satisfactory sinter without the addition of coke dust or sulphide fines to augment the fuel value of the flue dust, and on this account, together with the low first cost of a kiln, the latter was decided upon.

A couple of tons of flue dust were tried out in the experimental kiln in the laboratory at Yorktown, Va., of J. H. Payne, who acted as consulting engineer in this connection. This kiln was approximately 2 ft. in diameter and 20 ft. long and was fired with fuel oil. A run of several hours yielded excellent nodules. The kiln showed no tendency to build in and form "nose rings" and gave a fuel consumption of 50 gal. of oil per ton of flue dust. There was no indication of any stack loss. The roasting was strongly oxidizing. A study of what full-size kilns were doing on cement clinker and on nodulizing pyrites cinder indicated that a 60 by 6 ft. kiln would certainly not consume over half of the oil per ton of flue dust shown on the test and as fuel oil at that time was selling around 2 1/2 c. per gallon it was decided to let it go at that.

A 60 by 6 ft. kiln was decided on, as that size is standard in cement practice, although fast being replaced by much larger ones with their greater fuel economy, and as it fitted in the building space available. A 6-in. brick lining brought the net diameter down to 5 ft. The inclination

The results, while satisfactory, were quite different from those anticipated. In the first place, the fuel consumption was far lower than had been expected. A granular sand can be made with perhaps 8 gal. per ton, a first-class smelting product with 12, and great chunks with 16. It appears, therefore, that such a test kiln as that used takes about four times the fuel that will be required on a 60-ft. installation, although this ratio might be changed for different materials with varying internal fuel values.

In the matter of formation of nose rings, the test kiln was deceptive. There was a decided tendency to such formations and it took some time for the operators to acquire the necessary skill to control this. Steady conditions of flame are very necessary. If the kiln is overheated semi-molten material forms on the walls and a subsequent over-chilling will plaster the sand on very rapidly. A number of devices were tried to meet this difficulty, but finally it was found that reasonable skill and care on the part of the attendant and an occasional shutdown of a few hours to remove any obstinate obstruction were the best remedies. A patented scheme using a hollow drill, delivering compressed air or water, gave indifferent results, although it seems to have given satisfaction in pyrites-cinder work. Next the accretions were allowed to grow in such a way that they collapsed by their own weight. This, however, took about as much skill as avoiding their formation and, further, greatly diminished the capacity of the kiln. When perfectly clean 75 tons of flue dust can readily be nodulized in 24 hr., while a choked-up barrel will deliver less than half of this quantity. There is no difficulty in regular work in delivering 50 tons a day, including all delays, and a few hours' work for two or three men once in two weeks will handle the accretions. Another experiment that was tried was the use of a boring bar every hour or so while the kiln was in operation, but here again, while the scheme worked fairly well, the cure was worse than the disease. The flame had to be temporarily cut off and the loss of heat resulted in bad product for some minutes after each attack. Finally it was found that spraying with a hose would cause disintegration of these lumps. The nose rings, however, proved to be more of an annoyance than a serious difficulty and caused but little increase in the operating costs.

The normal operating condition of the kiln is to have the 40 ft. next the flue acting simply as a preheater, without any real nodulizing action. The material then begins to stick and ride around, which is the first step toward the formation of nodules. The actual nodulizing should be completed in the next 15 ft., leaving the last 5 ft. for hardening the nodules in the cooler zone back of the flame. This is to keep the nodules from sticking together in a semi-fused mass when they drop into the receiv-

ing car. The nose rings will be confined to the 15-ft. nodulizing zone, where they are readily accessible from the front end when the fire hood is rolled out.

It was expected that delivering red-hot nodules to the blast-furnace would result in quite a saving, but it was found that they smelted so fast that they tended to run away from the rest of the charge and cause uneven operation; much better results were obtained by feeding them cold. This would probably not be true in the case of nodulized ores, where the product would not be self-fluxing.

No trouble was experienced from flue dust blowing out of the kiln. In dry process cement work, where 95 per cent. of the material fed will pass a 100-mesh screen, but 3 per cent. flue dust is made. In the case of our flue dust fed, about 25 per cent. would pass 100 mesh.

Shortly after the kiln was installed the fuel-oil market ran away, the top prices reaching nearly 6c. a gallon; consequently some experiments with coal firing were tried. A large fire box was built, along the lines of those used in reverberatory work, operating largely as a gas producer, and equipped with forced draft. This was an absolute failure, due to the impossibility of keeping an even temperature at the mouth of the kiln. Every time fresh coal was fired the nodules would dwindle down to sand. It is obvious that the same objection would apply to any gas-producer plant not of sufficient size to give a uniform quality of gas. The great majority of rotary kilns are fired with pulverized coal. This requires a fire-box temperature sufficiently high to get satisfactory ignition, and as this flue-dust nodulizing is distinctly low-temperature work it was questionable whether the considerable investment in coal drying and pulverizing would have been justified. Fuel oil is the ideal fuel for such work wherever its cost will allow it to be considered at all.

While the oxidizing action of the small test kiln was very marked, the sulphur being almost totally removed, it was much less so in the larger kiln. Rough grab samples of the entering flue dust and the nodules produced showed the following analysis, which may be taken as representative:

	Flue Dust	Nodules
Au, ounces per ton.....	0.12	0.10
Ag, ounces per ton.....	11.30	10.90
Cu, per cent.....	12.00	13.02
FeO, per cent.....	31.33	36.93
Insoluble, per cent.....	22.10	26.50
SiO ₂ , per cent.....	18.19	20.80
CaO, per cent.....	0.48	0.54
Zn, per cent.....	1.97	1.88
S, per cent.....	11.86	4.95

It will be noted that but little over half of the sulphur has been removed. There is also some loss in silver indicated, but the sampling is too rough

to permit drawing any conclusion. No special attention was paid to silver losses, as the material treated was all low grade, but rich fines would doubtless show a heavy loss, judging from the results obtained on the old Brückner roaster in the early days.

Physically the nodules are an ideal blast-furnace product. A screening test when making quite small nodules gave the following results:

Between	Per cent.
1/2 and 1/4 in.....	54.21
1/4 and 3/16 in.....	23.16
3/16 and 1/16 in.....	20.60
1/16 and 20 mesh.....	1.10
20 and 30 mesh.....	0.54
30 and 40 mesh.....	0.14
40 and 60 mesh.....	0.12
60 and 80 mesh.....	0.05
Through 80 mesh.....	0.08
	100.00

While nodulizing is comparable with blast roasting, the two processes have different limitations, much as in the case of a comparison between blast furnace and reverberatory smelting. Sintering requires a properly balanced internal fuel supply, while nodulizing is entirely independent of this. To properly nodulize, a material must have a sufficient interval between the temperature of becoming sticky (incipient fusion) and that of actual melting to give a practical range of operating flame temperatures. As a nodule is formed from individual grains stuck together, every piece has a solid structure and there are no defective ones full of unsmelted material. I believe the kiln is deserving of a somewhat wider application in the metallurgical field than it has as yet obtained, but the extent of this expansion can only be determined by experiment. While these notes cover but a single material, blast-furnace flue dust, and that from but a single plant, yet this material is much more like ore than pyrites cinder and other refractory products. It would be advisable to use a kiln 7 or 8 ft. in diameter to lessen the nose-ring difficulty, and this is in line with the experience in other classes of work. As to length of kiln for low-temperature work, 60 ft. is probably long enough; for refractory materials calling for a greater unit fuel consumption longer kilns will pay in fuel saving.

As far as operating cost goes, the crux of the whole matter lies in the cost of fuel. A single operator can attend to the firing of several kilns; and the repairs should be very small, as the machinery is very simple, thoroughly standardized, and not exposed to the heat.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

"Playa" Panning on the Cauca River

BY WILLIAM F. WARD, DENVER, COLO.

(Salt Lake Meeting, August, 1914)

ONE often reads of the rich placer gravels in many of the canoe-traveled rivers of South America. The apparent richness of these gold-bearing gravels impresses the traveler, and in fact he may see batea after batea showing enough colors to figure over a dollar per cubic yard.

These workings are called "playas" from the Spanish word *playa*, meaning shore or beach. Many workings have been reported in tropical rivers, but to my knowledge no dredge has made a success in such deposits. Investigation generally shows that the area of richness is limited and especially that the depth of the rich concentration is hardly ever more than a few inches.

The Cauca river in Colombia, South America, is noted for its many rich playas. This river rises near Popayan in the southern part of Colombia, flows north through a narrow valley between parallel mountain ranges and empties into the Magdalena river near Maganque. The air-line distance between source and mouth of this river is only about 500 miles. The rainfall is large and the lower Cauca appears to be about the size of the Missouri river in the United States. From its mouth up to Valdivia, in the State of Antioquia, it is navigable. Above Valdivia is a narrow gorge, above which the river is again navigable for some distance.

The part of the Cauca where the playa panning is most noticeable is from Valdivia down to where the Nechi river enters the Cauca, a distance along the river of about 100 miles. The current in this part is swift, so that the stern-wheel river steamers make the trip with some difficulty.

In the course of my work in this section of Colombia I have had the opportunity to make a study of several of these playa workings and will give in detail the conditions observed.

The Island "Playa"

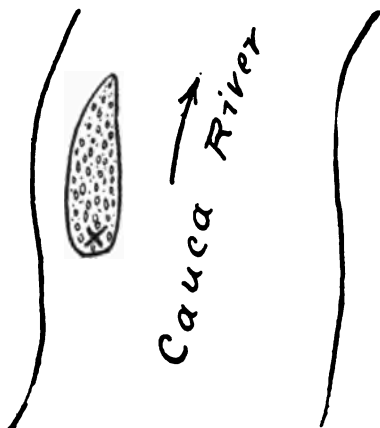
The richest of the island playas is about 40 miles below Valdivia and a little below the old town of Caceres. The shape of the river and the location of the deposit are shown in Fig. 1, which indicates the condition

at low water. The river at this place is about 600 ft. wide and the island about 75 ft. wide by 300 ft. long. The island is made up of coarse pebbles, some of which are as big as a man's head.

In times of high water this island is under several feet of water and in a swift current. When the river lowers the current is still fast enough to prevent sand or silt from covering the coarse pebbles.

At high water the banks of the river for several hundred feet above the island are continually caving in, and some of this material, together with the usual drag of fine gravel along the bottom of the stream, passes over this submerged island.

The coarse pebbles forming the surface act as natural riffles. In the course of a few months a mass of material is thus handled by natural agents, and a rich concentration of fine gold results. The natives work the playa when the water is low enough, which is two or three times a year.



Island surface made up of coarse pebbles. X Richest point of gold concentration.

FIG. 1.—SKETCH OF CAUCA RIVER BELOW CACERES, SHOWING "PLAYA" AT LOW WATER

As soon as the water goes down enough to permit wading on the island, the natives turn out with their small hand-shaped picks and their wooden bateas. They scrape the fine gravel into the bateas, rough out the coarser pebbles, then wade to the shore and complete the panning in the quieter water.

Many natives work at this place, so that by the time the low-water season is over the whole surface of the island has been gone over and only the coarser gravel remains, thus leaving the natural riffles thoroughly cleaned. At times it happens that the playa so worked over is even richer the following season.

At this playa a native for the first two or three days, makes as much as \$3 per day. The work continues for several weeks until only about 30¢ a day per native is realized, when operations are abandoned.

The Typical Beach "Playa"

Another noted playa just above Caceres is worked by the same natives; but it does not yield as much gold as the island playa. This is more nearly the typical playa or beach deposit, and is shown in Fig. 2.

The west side of the river is a beach of coarse pebbles which act as natural riffles. The river above turns in a way favorable to sending the drag of small gravel over the coarse beach gravel. The current at high water is swift enough to prevent sand or silt from accumulating, but is not so swift as to prevent some of the fine gold from depositing. Most of



FIG. 2.—A TYPICAL BEACH "PLAYA" ON THE CAUCA RIVER NEAR CACERES.

the playa deposits on the river are of this type, and are workable to a certain extent at all times except at extreme high water.

Cauca River Conditions Favor Playa Formation

The Cauca river is large and flows through a long narrow wooded valley, which makes it practically impossible for a heavy rainfall to produce a damaging flood. The greatest difference between high and low water is 10 ft. and usually it is much less. Had the valley been basin shaped resulting conditions would be quite different. The locations of many river playas remain the same year after year and only change when a marked change in the main currents takes place.

Near the mouth of the Tamana creek, about half way between Valdivia and the mouth of the Nechi river, a small arm of the river, which had

took place. Here for several days as much as \$4 per day was obtained by each native worker.

Once in a while a large snag will cause a change in the main current and incidentally a local playa will be formed below.

In the upper and swifter part of the river near Valdivia the shore line at low water shows boulders and angular rocks. These places are worked periodically by the natives, a small amount of gold being obtained.

The extreme lower parts of the river show playas only at very low water, the coarse gravel only showing at such times. One of these playas about 5 miles above the mouth of the Nechi river compared favorably with the playas in the upper river, since the water had not been low enough to permit working for several years.

The right to wash gravel on the playas is free. So when the high-water season is over the natives go prospecting in canoes for the richer playas. When a good one is found the news spreads and soon the richest places are cleaned up.

Most of the gold is obtained from the playas by women workers. While the pannings are rich many native men are seen on the playas; but the women keep at it longer, for they are content with less remuneration for their labor.

The bulk of the gold sooner or later gets to the Caceres storekeepers, the smaller merchants along the river, who make trips to Caceres once every two or three months, buying the gold at the various playa camps on the way. An average of about 15 lb. of gold is bought at Caceres monthly, about two-thirds of it coming from the river playas. Roughly about \$35,000 is, therefore, obtained yearly from the Cauca river playas.

From several drill holes in the river gravel itself near Caceres it was found that the value per cubic yard was from 1c. to 3c. Even this small amount of gold in the river gravel itself is enough to account for the source of the gold in the playas. This gold is small in size, often a dozen "colors" to the milligram; but, as is usually the case with such gold, it has a fineness of over 900.

One drill hole in the lower part of the river on an island near a rich playa gave results as follows: About 5 ft. of barren loam and sand; then about a foot of good gravel of 30c. grade; then to a depth of 50 ft. material of poor grade. The total hole averaged only about 2.5c. per cubic yard. The hole was not completed to bed rock.

The bench gravels on the sides of the river seem to have coarser gold and at places there are some very rich channels. At a playa just below a place where bench gravel had been caving into the stream some coarse gold of a dull color, was seen in the pannings along with the fine bright river gold.

There are evidences of extensive work in the bench gravels all along this stretch of river. A fair record of gold produced is credited to the district. Not enough prospecting has been done so far to determine whether or not the areas of pay gravel are large enough to work by modern methods.

The rich channels of the benches will be first worked out, but the playas for years to come will be a source of gold.

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Copper Ores of the New London Mine*

BY B. S. BUTLER AND H. D. MCCASKEY, WASHINGTON, D. C.

(Salt Lake Meeting, August, 1914)

Introduction

THE New London copper mine, about $8\frac{1}{2}$ miles east of Frederick, Md., was visited by the writers for a few hours in the spring of 1909 and the following brief notes on ore specimens collected are presented as a contribution to studies in chalcocite ores.

For certain data on the structural and areal geology of this portion of Maryland the writers are under obligation to the unpublished notes and maps of Arthur Keith. The detailed study of the ores is the work of B. S. Butler.

The deposit here discussed has many similarities to those of the Virginia district of Virginia and North Carolina, which have been described by Graton¹ and Laney², except that in the deposits of the New London mine chalcocite is by far the most important ore mineral, while in the Virginia deposits bornite is commonly the most important sulphide. In the New London occurrence calcite is the most important gangue mineral.

Field Relations

The town of Frederick lies in the center of a narrow belt of limestone ranging in age from Ordovician on the east to Cambrian on the west. This belt is about $\frac{1}{2}$ mile wide at Noland's Ferry, on the Potomac, and broadens to a width of 3 miles at Frederick. The general direction is east of north. The eastern boundary is, in places, a fault plane along which a series of metamorphic rocks of presumptive pre-Cambrian age have been thrust up. These ancient rocks, for several miles east and west of New London, are phyllites in the main, but closely interfolded near the New London mine are long, narrow lenses of epidote

* Published by permission of the U. S. Geological Survey.

¹ Graton, L. C.: *Mineral Resources of the United States*, pt. 1, p. 620 (1907).

² Laney, F. B.: *Economic Geology*, vol. vi, No. 4, pp. 399 to 411 (June, 1911).

and of quartzite, the entire series lying in the characteristic northeasterly strike of the Appalachian rocks. Crossing the boundary fault at a very slight angle is a persistent narrow diabase dike, probably of Triassic age, many miles long—the filling of a deep fissure cutting both Ordovician and pre-Cambrian rocks. The New London mine lies nearly in the center of the elongated group of exposures of epidote schist, which in turn lie about in the center of the broad belt of phyllites. The phyllites in general strike northeasterly and dip at high angles southeasterly. They are fine-grained, dark, and crumpled to microscopic degree. Their origin, whether sedimentary or igneous, is not wholly determined, though part are pyroclastic and part are sedimentary. The epidote schists are in part amygdaloidal—the characteristic Catoctin schists or greenstones of the Appalachians—and they are derived by regional metamorphism of ancient basaltic extrusives. The marble and the quartzite are considered to be of pre-Cambrian age. Intense forces have crumpled all these formations, and their present structure, where not massive, is trough or saddle form.

About $\frac{1}{2}$ mile west of the New London mine a narrow oxidized manganese vein is found at the contact between epidote schist above and marble below, dipping 55° S. 80° E. and striking N. 10° E. The strike of small pits and shafts along the outcrop of the New London vein was found to be about N. 62° W., magnetic, indicating a fracture across the prevailing strike of the rocks. The succession of rocks at the mine is marble overlain by epidote schist, in turn overlain by quartzite. Below the marble lens, and above the quartzite also, lies the soft, crumpled, fine-grained, black phyllite. The ore appears confined mainly to the marble, and ore solutions were probably limited in circulation by the phyllite. It is also assumed that the resistant marble and quartzite afforded the necessary fracturing for passage of the ore solutions, as the surrounding soft phyllites are tight, finely folded rocks and do not suggest openings of any kind.

The New London Mine

The New London mine is an old property on the southern end of the western copper belt of Maryland which extends northeasterly to the Liberty mine, formerly the largest copper producer of the State. The total output of copper of all Maryland mines has been relatively small, but these deposits have furnished limited quantities of rich ore. The principal workings of the New London mine in 1909, at the time of our visit, were a shaft 210 ft. deep and a stope in the bottom. Older workings consisted of an inclined shaft to the 173-ft. level and a number of stopes, all of which were pretty well filled with waste. The ore from the bottom

stope was principally chalcocite, with minor quantities of bornite, and with gangue of coarse crystalline pinkish calcite, blebs of white quartz, and interfolded and crushed country rock. The small output has consisted in part of hand-sorted ore running 10 to 11 per cent. in copper and more recently of concentrates of higher grade. Sampling of the ore at a depth of 210 ft. is reported to have shown an average content of 4.9 per cent. of copper. Pyrite was reported in small quantity at the bottom of the 200-ft. level, but none was observed by the writers.

Since the visit of 1909 a 100-ton concentrating mill has been put up on the property and the new inclined shaft has been sunk to the 300-ft. level. On the 200-ft. level the drift has been extended 305 ft. easterly and 95 ft. to the west. The ore shoot is reported to extend 150 ft. on this level, with an average content of 3.6 per cent. copper.

Description of the Ores

The ores may be roughly divided into two types, that may be designated as (1) typical "vein" ore and (2) banded or schistose ore.

The first type, so far as observed, occurs in the wider portions of the vein. It consists of coarsely crystalline calcite, quartz, and sulphide. Barite was noted in some of the thin sections examined, but was not detected in the hand specimens. Chalcopyrite^{*} is reported from the district, but was not observed by the writers. The sulphide is mainly chalcocite, with bornite scattered in small particles through it. The calcite and quartz are in crystals and aggregates, in some cases exceeding an inch in diameter.

The calcite varies from nearly white to light pink in color. No quantitative chemical work has been done on this mineral, but it contains the merest trace of magnesium, manganese, and iron. The presence of manganese is shown by the oxides of this metal that result from the surface alteration. The quartz and calcite are contemporaneous in origin, each mineral including the other, and the barite observed is contemporaneous with the carbonate, though its relation to the quartz was not noted. A large part of the sulphide in this type of ore is later than the quartz and calcite. It surrounds the crystals and enters along fracture and cleavage planes, and replaces both quartz and calcite. (See Fig. 1.) To a slight extent the sulphide is contemporaneous with the gangue minerals, being completely inclosed in quartz and calcite crystals that have surrounded the sulphide grains in the process of growth. The chalcocite is notably well crystallized, crystal faces exceeding $\frac{1}{4}$ in. in greatest dimension being present in the specimens.

Bornite occurs in small irregular grains through the chalcocite. At first glance the sulphide appears to be pure chalcocite, but on close ex-

^{*} Weed, W. H.: *The Copper Mines of the World*, p. 267 (1907).

amination small irregular areas of bornite can be seen in all the specimens of massive chalcocite examined.

Microscopic examination of polished and etched faces of the sulphide shows that very commonly the bornite occurs between the chalcocite crystals and frequently occupies fractures in them. (See Fig. 2.) In other instances there is a parallel intergrowth of lath-like crystals of bornite and chalcocite. In such cases the several lath-like bodies are apparently portions of an individual crystal, all having a uniform orientation. In many instances the bornite occurs as small specks through the

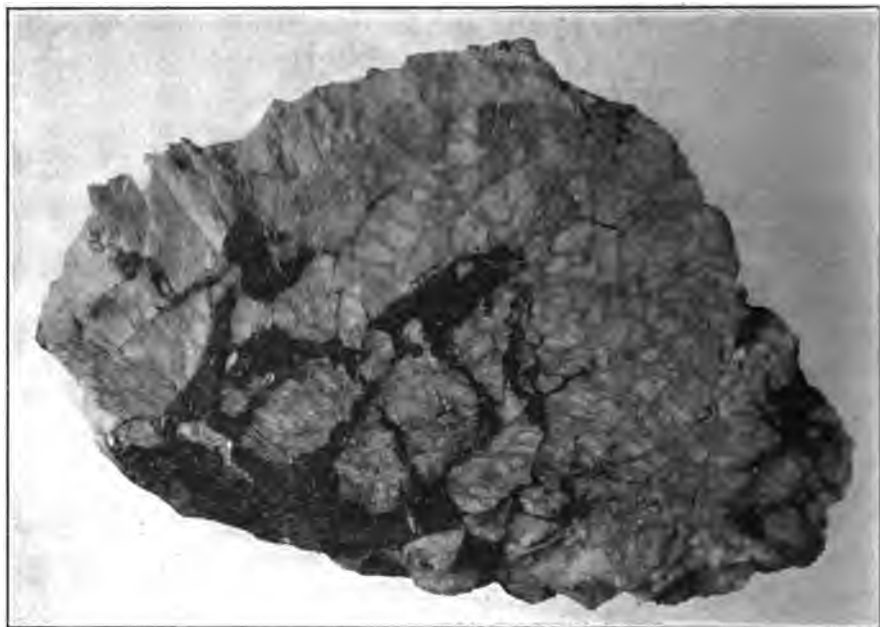


FIG. 1.—“VEIN” ORE, NEW LONDON MINE, SHOWING RELATION OF SULPHIDE TO GANGUE MINERALS. Dark areas, Sulphide. Light areas, mainly Calcite.

chalcocite, but frequently with a definite arrangement that suggests intergrowth.

A similar relation exists between different chalcocite crystals as that between chalcocite and bornite. Lath-like crystals of chalcocite are frequently inclosed in irregular chalcocite crystals, and chalcocite crystals occur between other crystals and extend into fractures.

As already indicated, some of the bornite is undoubtedly later than some of the chalcocite, and in no instance is there positive evidence that the bornite has been replaced by chalcocite. For the most part the two sulphides are contemporaneous. The interlocking of the chalcocite crystals is similar to that resulting from the crystallization of a metal

or that of a granular igneous rock. There is no structural indication that the chalcocite has resulted from the replacement of an earlier sulphide.

The banded or schistose ore may occupy the entire "vein" in the narrower parts or only a portion where the "vein" is wider. Typically it exhibits a finely laminated structure with successive layers of light and dark bands. In some specimens the individual layers have a thick-

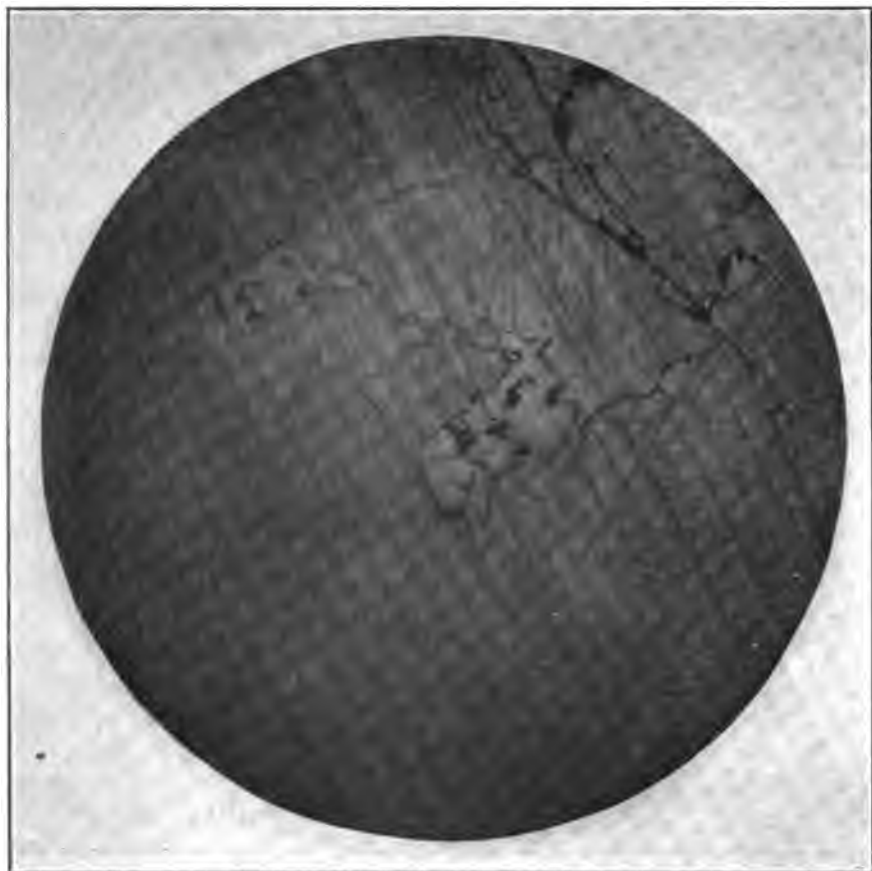


FIG. 2.—"VEIN" ORE. $\times 30$. SHOWING RELATION OF CHALCOCITE AND BORNITE. Small bodies lying between larger crystals and connected by narrow stringer, Bornite. Large crystals, Chalcocite. Bornite occupies fissure in Chalcocite.

ness not exceeding $\frac{1}{16}$ in., while in other instances bands composed largely of dark or light minerals will have a thickness of $\frac{1}{2}$ in. These banded ores include lenticular nodules in which the banding is much less distinct. The banded ore has the general appearance of a schist that has resulted from the metamorphism of a rock in which some of the more

resistant portions still retain their identity in part though drawn out into lenticular form. (See Fig. 3.)

In this type of ore the light bands are composed mainly of calcite and quartz, calcite being much more abundant than quartz, while the dark bands are due to the presence of the copper sulphides. Near the walls of the "vein" and in some instances at some distance from the

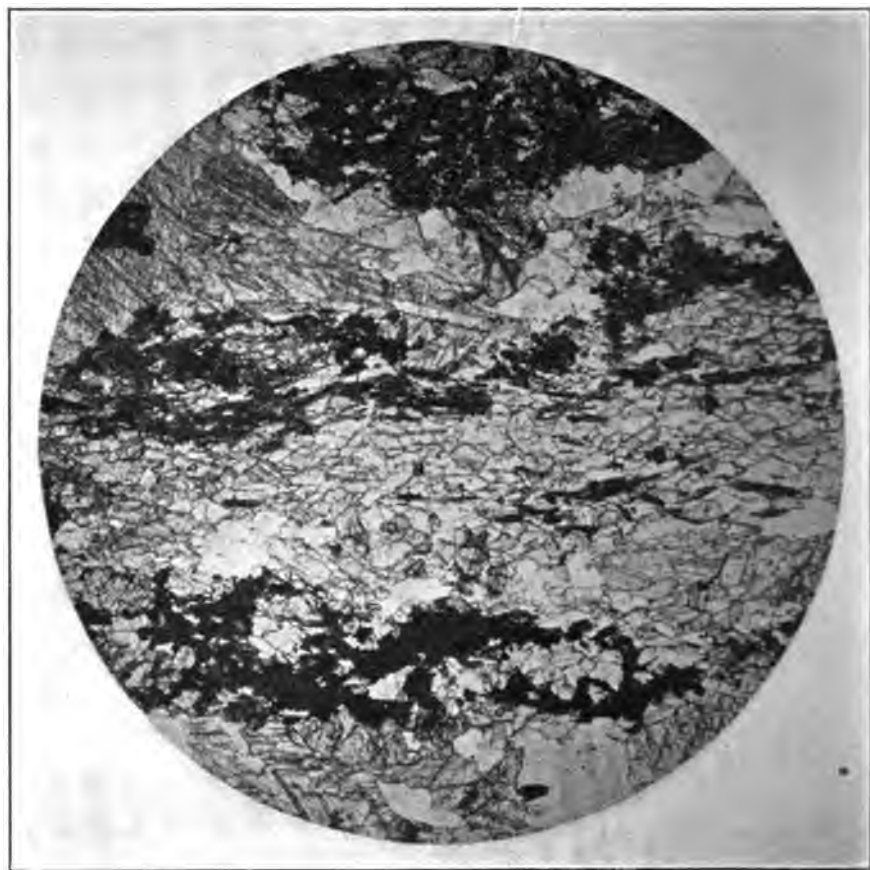


FIG. 3.—"SCHISTOSE" ORE. $\times 30$. SHOWING RELATIONS OF SULPHIDE TO GANGUE MINERALS.

Dark areas, Sulphide. Light areas, mainly Calcite.

walls, chlorite, specularite, and muscovite are abundant. Tourmaline and titanite are locally abundant in the banded ores and feldspar is present in small amounts. The calcite, quartz, and sulphide of this type of ore have been crushed and recrystallized. Abundant examples are to be seen when each of these minerals includes the others, indi-

cating contemporaneous crystallization. The brittle sulphide has been crushed and drawn out into bands composed of small fragments.

The tourmaline occurs in small crystals showing pleochroism from light to dark brown. These can rarely be seen with the unaided eye, but are readily detected with the hand lens when their presence is known. Under the microscope they are found included for the most part in the other minerals, but rarely sulphide is inclosed in the tourmaline crystals. The crystals in general lie in a plane parallel to the banding. (See Fig. 3.) The mineral appears to have formed in the early stages of the recrystallization including fragments of the sulphide, but later it was included in the carbonate, quartz, and orthoclase, and also in recrystallized chalcocite. The titanite is in small granules apparently formed at about the same time as the tourmaline and the two are frequently intimately associated.

As already stated, near the walls and apparently wherever the wall rock has been included in the vein, chlorite and specularite become abundant in the ore and muscovite is of common occurrence. These minerals are similar in their character and occurrence to those in the schists which form the wall rock.

The sulphides recognized in the schistose ore are chalcocite, bornite, and pyrite.

The relations of the chalcocite and bornite are in every way similar to those already described. Only a few small grains of pyrite were noted. They were apparently among the first minerals formed, and in no instance observed was there any replacement of pyrite by copper sulphide.

Relation of the Two Types of Ore

Where the two types of ore come together there is commonly a gradual transition from the one to the other.

This relation suggests that the schistose ore may have resulted from a metamorphism of the vein due to slight movement under conditions of considerable heat and pressure. The dynamic metamorphism of ore deposits in the Appalachian region has been described by Emmons,⁴ and need not be discussed here further than to say that it has been noted by several observers in widely separated localities in the southern as well as the northern Appalachians.

Genesis of the Sulphides

The genesis of the deposits will not be discussed except as to the origin of the sulphides; namely, whether they are "secondary"—that is, resulting from the replacement of earlier sulphides—or whether they were deposited essentially as they are now except as affected by dynamic metamorphism.

⁴ Emmons, W. H.: *Economic Geology*, vol. iv, No. 8, pp. 755 to 781 (Dec., 1909).

nized in the deposit, and further, that there is no indication in the structure of the chalcocite that it has replaced any earlier sulphide.

The intimate intergrowth of the sulphides with carbonate seems a strong additional argument against considering them as resulting from downward sulphate solutions. Theoretically one would expect the copper sulphate solutions would react with the carbonate of the gangue to form the relatively stable copper carbonate and that there would be slight migration of the copper solution. Such is known to be the case in instances where sulphides are intimately associated with a carbonate gangue.⁵ Spencer⁶ has recently shown that calcite does not precipitate copper from a solution which contains both cupric and ferrous sulphates. Some of the carbonate in the gangue of the ore contains a little iron, but that it is a precipitant of copper is amply shown by the presence of malachite along cleavages in the carbonate where incipient oxidation of the chalcocite has taken place. It has also been shown experimentally that the calcite of the gangue will readily precipitate copper as carbonate from a sulphate solution. It does not seem probable, therefore, that the copper could have been transported as sulphate and have replaced another sulphide entirely inclosed in a carbonate grain.

The coarse crystalline structure of the "vein" ore and gangue suggest deep-seated and not surface origin.

The evidence seems to the writers to point to the conclusion that the chalcocite, which is the principal sulphide in the deposit at the New London mine, was deposited without the replacement of an earlier sulphide.

⁵ Bard, D. C.: *Economic Geology*, vol. v, No. 1, p. 59 (Jan., 1910).

⁶ Spencer, A. C.: *Economic Geology*, vol. viii, No. 7, p. 648 (Oct., 1913).

Effects of the Bag House on the Metallurgy of Lead

BY L. DOUGLASS ANDERSON, MIDVALE, UTAH

(Salt Lake Meeting, August, 1914)

FOR some years past the annual reviews of the metallurgy of lead have almost uniformly stated that there have been no great changes, such as there were, being more particularly noticeable in the refinement of details. While this is quite true, nevertheless these changes in details are gradually bringing about a condition that, taken as a whole, constitutes a considerable advance in the art. Local circumstances have often compelled metallurgists to adopt certain peculiarities of practice which upon trial have been found advantageously applicable to more than the restricted district of origin. This paper will deal with some of the more novel features developed in recent years at the United States smelter, Midvale, Utah, particularly with reference to the influence of the bag house.

The plant is equipped with six blast furnaces, 48 by 160 in., mechanically fed; four Wedge furnaces for roasting matte and sulphide ores; four Dwight-Lloyd sintering machines; 20 converter boxes for blast roasting; storage bins for ore, fuel, and fluxes; "oxide" and "sulphide" sampling mills; bag house taking all the gases of the plant; arsenic plant for treating the dust gathered there; laboratory, shops, etc.

The most marked feature of the plant is its complete bag-house system. Of the many schemes thus far tried for preventing damage to surrounding agricultural interests the writer is aware of none as yet quite as efficient as the bag house. A common remark of visitors upon approaching the plant and viewing the clear stacks is that it appears to be shut down, whereas once inside all the evidences of industry are found.

Bag-housing blast-furnace gases presents no peculiar difficulties. The gases from properly run furnaces are comparatively cool, contain scarcely a trace of acid and do not burn the filtering medium. All that is required is a proper mechanical design of the bag house and its accessories, the provision of ample filtering area, and of convenient means for handling and treating the product. While cotton or duck bags have been used, on the whole woolen bags woven of the raw wool appear to give sufficiently longer life to warrant their heavier expense. As for the filtering

area required, no definite rule can be laid down, as it is dependent upon the amount of and physical character of the fume carried in the gases and the frequency and efficiency of shaking. However, from 0.3 to 1.5 cu. ft. per minute have been filtered per square foot of area supplied, with 0.5 to 0.7 cu. ft. per minute the more usual quantities. The pressure carried is usually from 1 to 2 in. of water. Higher pressures can be carried, with consequent greater filtering capacity, but at the expense of more power and a somewhat shorter life of the bags.

The dust in the Midvale bag house is caught in V-shaped steel hoppers, through the bottoms of which run 9-in. screw conveyors by means of which the dust is regularly removed into tram cars which carry it to the arsenic plant for treatment. This idea, which was developed here, has resulted in much healthier conditions for the attendants and has been applied since to other bag houses, some of which use no hoppers but let the dust form its own slope in closed cellars down to the screw. How successful this is cannot be definitely stated, but as most dust tends to "hang up," requiring some rapping of the steel hoppers to bring it along, there is the possibility that screw conveyors may not be entirely effective when applied in rectangular cellars. It is to be noted that screw conveyors as ordinarily used run in semicircular or square boxes. The natural extension of the idea to dust hoppers would be to simply attach the boxes to the hoppers as in *A* or *B* of Fig. 1. With bag-house dust, however, such arrangements will invariably choke up and stop the worms. A better arrangement is shown in section at *C*. This arrangement seems to give the dust a chance to relieve itself on the sides instead of choking and breaking the screws. The present manner of shaking the bags was developed some years ago at this plant in an effort to save expense in the replacement of torn bags. It consists simply of cutting off the blast from any one hopper and applying a slight vacuum, repeating the operation two or three times. It is not as efficient as hand shaking or shaking by mechanical means. It is, however, healthier than hand shaking and, when done often enough (every 8 hr. at Midvale), is sufficient. Furthermore, bags often reach a peculiar state of fragility, when they are still efficient filtering mediums but so weak that they tear if handled roughly. It can be readily understood that gentle alternate deflation and inflation in place of rough shaking will greatly prolong their life when in this condition.

In contrast to blast-furnace gases, those from the roasting operations present many difficulties, due chiefly to their heat and the fact that they carry appreciable quantities of sulphur trioxide (SO_3). In cooling the gases sufficiently to avoid charring the bags this sulphur trioxide unites with the water vapor and becomes exceedingly destructive to the bags, whether of cotton or of wool. Under some conditions of roasting, where the ores are high in lead or zinc, the fumes from these metals unite with

bag-house their roaster gases. As a matter of fact, blast-furnace gases carrying practically no acid should not be nearly as harmful to vegetation as those from roasting operations, which are so hot as to permit little settling of lead and arsenic fumes and usually carry noticeable quantities of sulphuric acid. The neutralization of this acid effects two things,

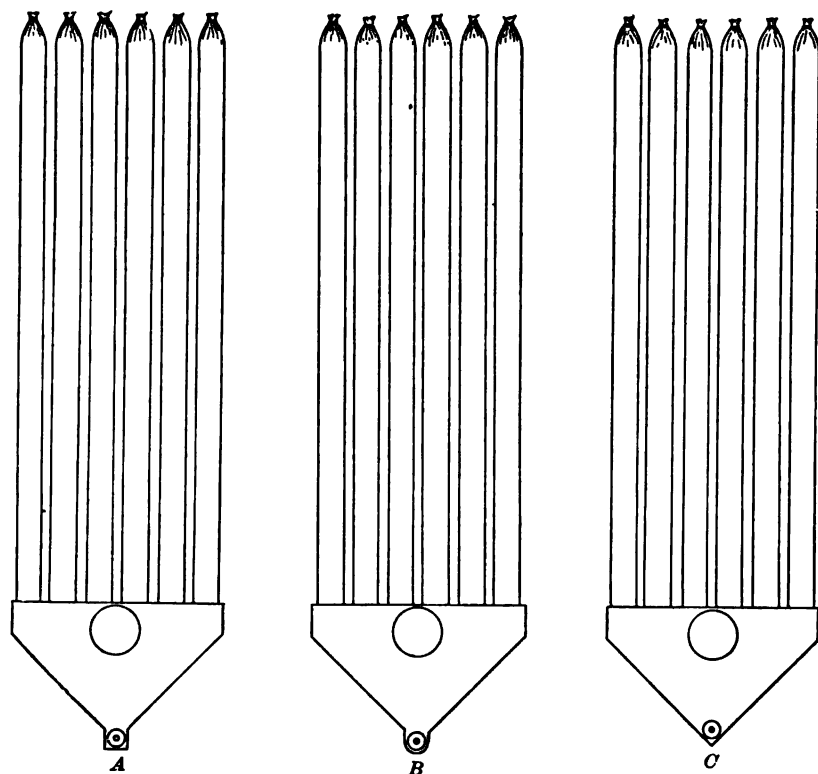


FIG. 1. Different Arrangements of Hoppers and Conveyor Boxes.

permitting filtration through woolen bags and nullifying the harmful effect of the acid on vegetation.

In the matter of metallurgical recovery the bag housing of roaster gases is also more important than that of furnace gases. In his paper on the bag house before the Institute in 1912,¹ Anton Eilers gave figures which showed that the operation of the bag house on furnace gases involved an actual loss financially when all items were taken into account. This should prove true in all plants where the furnaces carry a small

¹U. S. patents Nos. 931515 and 992391.

²Notes on Bag-Filtration Plants, *Trans.*, xlv, 708 to 735 (1912).

all values that are economically worth while. With more lead on the charge, however, and when agricultural interests must be protected the bag house is well worth while. But from the standpoint of metallurgy the roaster gases are those which will pay the largest returns on the investment. If hot tops in blast-furnace work are the cause of metal losses it is just as certain that the heat engendered in roasting likewise produces metal losses. This was indeed the primary reason for abandoning the old "fusion-box" type of roaster. At Midvale the fume collected in the bag house from the roaster gases carries 35 to 42 per cent. lead and about 10 oz. silver per ton, whereas that from the furnace gases carries only 18 to 22 per cent. lead and less than 1 oz. of silver per ton.

In his article on the progress of the metallurgy of lead in the first volume of *The Mineral Industry*, published in 1893, Professor Hofman called attention to the increasing exhaustion of the oxidized and carbonate ores of lead and the necessity of handling more of the sulphide ores, which were developed as the mines increased in depth. Roasting is a practically indispensable part of the treatment of these sulphide ores. In his paper on the bag house already quoted Anton Eilers mentioned the increased metallurgical losses due to the greater amount of roasting now being done. In this connection may be mentioned some experiences with the Dwight-Lloyd sintering machines, which have done so much for the whole industry. When originally installed at Midvale they were equipped with gasoline burners for ignition, which were somewhat troublesome and erratic in action. Some bad slags were produced in the blast furnaces when handling their product, which were finally traced to green sulphides carrying zinc blende coming over unignited with other parts of the charge which had properly sintered. Screening the roast was resorted to, with beneficial results. Finally a muffle type of burner using a low-grade gas oil, understood to have been developed first at Monterey, was applied. A good solid sinter was obtained without green spots and screening was abandoned; indeed, had to be, or otherwise the furnace charge became too coarse for good reduction. But the terrific heat developed by the new burners, and apparently necessary for good ignition, caused an increased loss of lead in the sintering operation. Many smelters throughout the country have lately begun to realize this. Yet the Dwight-Lloyd machine is too valuable to be impeded in its progress by this development. A more gentle ignition may cut down these metal losses. Nevertheless if they would be prevented to the maximum extent possible the bag house remains the final resort, not only for the gases from these machines, but for those from all lead-roasting operations.

It may be claimed that sufficient flues and settling chambers should throw down practically all the As_2O_3 and SO_3 , but it is not recalled that

any lead smelter in this country has such an extensive system ending in a stack as to prevent a dense cloud escaping from that stack, unless all the fume is filtered.

Aside from the acid nature of the roaster gases, overcome by the Sprague process of neutralization, the cooling of these gases has been the most serious problem in bag housing them. Many efforts have been made to produce filtering fabrics which would withstand both heat and acidity, such as by chemically treating cotton and wool, using asbestos, etc. These efforts are to some extent misdirected, for effectual filtering cannot be accomplished until the gases are cooled sufficiently to get the fumes into a condition more nearly approximating what is considered solid matter, nor can the gases be so cold that condensation of moisture occurs to destroy the filter fabric. Any attempt to filter hot gases will be disappointing, resulting in visible clouds of smoke as the gases passing through the bags meet the outer air, become cooled, and precipitate the fumes which had escaped the bags in a fine state of division. As long as the gases are hot the minute particles of fume seem to repel each other and exist in a sort of smoky cloud. Cooled, however, they tend to coalesce and thereby reach a state wherein they are easily retained in the meshes of the bags. Those who have given any study at all to the matter are well aware of the distinction between true flue dust and fume. The former can be quite readily settled out of the gases by large chambers, centrifugal devices, or wet washers. But lead-smelter gases carry as their chief solids, in addition to flue dust, various metals, principally lead, zinc, and arsenic and their compounds, in a state of extreme fineness, popularly mistaken for volatilization. The failure to recognize the distinction between flue dust and fume has resulted in many failures in efforts to apply to smelter gases wet washers or centrifugal devices which were quite effective on the gases from iron blast furnaces. What success has been attained with wet washers has been chiefly due to the cooling action effected. The importance of cooling cannot therefore, be overestimated.

Two difficulties brought on by simple cooling in flues are overcome by the bag-house system as now developed. The poor draft in stacks resulting from cool gases is taken care of by the forced draft used. The precipitation of water vapor and its combination with SO_2 to form actual H_2SO_4 is overcome by neutralization. The effectiveness of this neutralization is illustrated by measures taken to preserve certain steel cooling flues. Originally most of the neutralizer was admitted to the flue system where it would be most effective; that is to say, some distance from the roasters, where the gases were cooler. But cooling had proceeded far enough between these two points to cause the gases to seriously corrode the steel plates. Finally a certain amount of slacked lime was fed into the gases at the beginning as well as at the end of the cooling flues. From that time on internal corrosion ceased entirely.

Three methods of cooling are commonly in use: by dilution with external air, by passing through thin metal flues, and by spraying with water. The first method has only a limited application, lest the quantity of gas handled become excessive. The second method is probably the more extensively used. The rate of heat transmission through sheet-steel plates about $\frac{1}{8}$ in. thick under smelter conditions appears to be about 0.02 to 0.04 B.t.u. per minute per square foot of cooling surface per degree Fahrenheit difference between interior and exterior temperatures, depending upon cleanliness of the flues, convection by winds, shading from direct rays of the sun, etc. In computing the required area of a flue for cooling it is not sufficient to assume the mean interior temperature as the arithmetical mean of the initial and final temperatures, for the drop in temperature proceeds more slowly as the gases become cooler. If t_1 be the initial temperature of the interior gases, t_2 the final temperature, and t_s the temperature of the external air, the mean temperature difference θ_m between inside and outside will be

$$\theta_m = \frac{t_1 - t_2}{\log \frac{t_s - t_2}{t_s - t_1}}$$

In the use of sheet-metal flues it has been found by experiment that the cooling, within certain limits, is proportional to the square root of the velocity. For this reason it is advisable to keep the velocity high in such flues, which also tends to avoid too heavy a deposition of the heat-insulating dust.

Cooling by water sprays has been used to some extent and will undoubtedly be extended now that efficient atomizing nozzles are obtainable. Care must be used, however, that too much water be not used and the saturation point of the gases be not too closely approached, as any dampness of bags is fatal to their life. It is an interesting fact that by full utilization of the latent heat of evaporation gases can be cooled by warm water below the temperature of the water itself. Those who are interested in this subject are referred to a paper by Willis H. Carrier, entitled *Rational Psychrometric Formulæ*.³

In the Midvale bag house, roaster and blast-furnace gases and dusts are kept entirely separate. The roaster dust, carrying 9 to 16 per cent. arsenic, is briquetted with other flue dust and fine ores. The lime fed in as a neutralizing agent provides an efficient binder. These briquettes are then charged to the blast furnaces. The dust from the blast-furnace section of the bag house carries from 35 to 45 per cent. arsenic. This dust is treated in the arsenic plant, where by two successive volatilizations a high-grade fine white commercial arsenic product is made, running better

³ *Transactions of the American Society of Mechanical Engineers*, vol. xxxiii, pp. 1005 to 1039 (1911).

than 99.5 per cent. As_2O_3 . It is seen, therefore, that there is a gradual concentration of the arsenic through the successive steps of the roasters, the bag house, the briquetting press, the blast furnaces, the bag house again, and finally the arsenic plant.

The details of arsenic manufacture were ably treated last year before the Institute by James O. Elton. Further comment must be restricted to mentioning the troubles experienced when the blast-furnace bag-house dust approaches 45 per cent. in arsenic content. At all times it is an inflammable material. The success of the hoppers with their screw conveyors is due to their being kept cool by free air circulation around them and regular frequent removal of the dust. If this were allowed to accumulate in large quantities it would catch fire and burn to crusts, which would destroy the revolving screws. Furthermore, the arsenic content must be kept under 45 per cent. or the dust will tend to burn in the cars on the way to the arsenic plant, giving off heavy white fumes, making it well-nigh impossible to handle. For this reason the metallurgist in a plant of this kind receiving many ores high in arsenides has to watch his charges for arsenic in addition to the usual elements.

Having caught the material which formerly escaped from the stacks, his problem is to extract from it the valuable metals and get the arsenic out of the system as quickly as possible, minimizing any closed circuits. Such a one, for example, is formed in the briquetting of the arsenic-plant residue or sinter after roasting. What arsenic was not roasted out of the bag-house dust on the Brunton hearths goes back to the blast furnaces, then to the bag house, and from there once more to the arsenic plant. It will also be found that certain of the less common elements will tend to build up in the sinter unless it be eventually removed from the system for further treatment, cadmium being especially noticeable, in addition to small quantities of tellurium, selenium, etc. All told, then, it will be seen that the bag-house system introduces some interesting new problems that the lead metallurgist often neglects to consider.

It is worthy of note that the bag house should not be depended upon as a "cure all" for bad furnace running. No metallurgist should assume that because he has a bag house to catch the fume he can be reckless about the condition of his furnace tops with the idea that the bag house will catch the values which may be given off. In the first place, hot tops will produce hot gases, which will be hard on the bags. In the second place, even if the lead fume is arrested in the bag house it has to be treated again at heavy expense, and with a second loss due to handling, slag losses, etc. The blast furnaces should be run fully as carefully with a bag house as without, if the maximum recovery would be obtained. To get the maximum percentage of the lead into the bullion with a minimum output of by-products is still the rule for high recovery.

Where all the gases of a lead smelter are put through a bag house it

that this applies only when all the gases from every operation whatsoever are filtered. The slag-settling furnace has proved of considerable aid in cleansing the slags finally rejected. Its actual saving, however, cannot be figured from the difference between the lead in the slag fed to it and the lead in the slag thrown away. A considerable amount of lead must escape from the stack together with the products of combustion. It is on record that certain metallurgical plants were able to make wonderful records for clean slags, only to find after a year's campaign that there was a distressing and mysterious large percentage loss. Such experiences as these are forcing metallurgists to recognize the fact that lead is rather an elusive metal and that it has a very large avenue of escape in the fumes which have been poured out so unrestrainedly in the past.

The installation of four Dwight-Lloyd sintering machines in this plant introduced an unforeseen complication in connection with the bag-house system. Soon after they were gotten into operation mysterious sudden rises of temperature and pressure in the flue system began to occur. The dust caught in the bags changed in character, becoming sticky and clogging them up so much as to require vigorous hand shaking to free them. Finally one of the aforementioned sudden rises in temperature became so great as to set fire to several hundred bags, the dust in the hoppers and flues also burning vigorously. It began to appear as if the gases from the Dwight-Lloyd machines could not be filtered. As the plant could not run unless all gases went through the bag house, extensive experiments were set under way in an effort to solve the difficulty. It finally became clear that the trouble was due to the production of free elemental sulphur, which occurred in the gases in a state of vapor. This free sulphur vapor would apparently ignite on small provocation, there appearing at times to be an actual spontaneous combustion of the material with the arsenic present, considerable arsenious sulphide being produced, while at other times probably sparks drawn through the wind boxes of the sintering machines were the cause of ignition. The steam generated from the moisture contained in the sintering-machine charges probably produced the neutral or reducing atmosphere required in the Hall process of producing free sulphur. Here, however, this free sulphur was most unwelcome and every effort was made to oxidize it or arrest it before it reached the bag house, but with very small success. Water sprays of many types were tried, but the free sulphur vapor proved practically as elusive as lead fume, which all metallurgists know is scarcely touched by water. Strangely enough, the percentage of free sulphur produced became less and less as the weather became warmer and the air drier. Probably if the machines could be supplied with dry air, after the manner of the pyrites burners in the Mannheim contact process for the manu-

schemes tried was the use of crushed limestone on the grates under the ore charge proper. This did to a certain small extent diminish the percentage of free sulphur produced. It was not sufficient, however, to prevent a second explosion in November, 1913, when the atmospheric air had again become damp. When this occurred the machines were shut down except for sintering pre-roasted matte, which operation did not appear to generate free sulphur, and for sintering sulphide ores low in pyrite.

"Conservation of resources" is in the air everywhere. A great contribution to this conservation is the closer saving of the raw materials being made in all branches of industry. The lead smelters have not yet reached the perfection of saving made by the packers who boasted of using all parts of the pig except the squeal—and are accused of utilizing even that now that they are under investigation by the Federal government! Yet progress is being made. The increasing development of hydrometallurgy may afford a market for the acid which could be made from the sulphur; even the iron of the slags may be recovered in such processes. In this progress a most important contribution has been made by such men as Percy, one of the first to suggest filtering mediums for smelter fumes; Lewis and Bartlett, whose successful bag process was the inspiration of many bag houses; the able and distinguished technical staff of the American Smelting & Refining Co., who solved many troublesome details; Robert D. Rhodes, whose patience and determination brought roaster gases within the province of the bag house; Sprague, whose neutralization process removed a large stumbling block, and many others whose names time does not permit us to recall.

Chloridizing Leaching at Park City

BY THEODORE P. HOLT, PARK CITY, UTAH

(Salt Lake Meeting, August, 1914)

Outline of the Process

THE Mines Operating Co.'s plant at Park City, Utah, was designed to treat the low-grade fillings in the old stopes of the Ontario mine. These fillings carry 6 to 14 oz. of silver, 1 to 2 lb. copper, 0.01 to 0.015 oz. of gold, and a small percentage of lead and zinc. The treatment consists in mixing the crushed ore with coal dust and salt and then roasting in a new type of furnace by combustion of the contained fuel. The roasted ore is leached with an acid salt solution to dissolve the silver, gold, copper, and lead. At present these metals are precipitated together on scrap iron and the product sold to a refinery.

The only new feature of importance in this scheme of treatment is the roasting process, which makes possible the chloridizing of ores without any loss of the valuable metals, and at a very low cost.

History of the Development of the Process

The roasting process had its beginning in connection with research work at the Utah State School of Mines. N. C. Christensen, Jr., who held one of the School of Mines' fellowships, was engaged in some experimental work on blast roasting. At the same time I was doing some work on the insoluble gold in Mercur base ores. Mr. Christensen roasted some of this base gold ore in his pot furnace, and found that when he mixed a sufficiently low percentage of fuel with the ore, it did not sinter but roasted to a leachable product.

We considered this a new application of blast roasting, and proceeded to test out a large number of ores in several different types of small roasters which we constructed. This experimental work continued well into the summer of 1911.

In August, 1911, the Consolidated Mercur Gold Mines Co. installed a

time drawing off the roasted ore from the bottom. By proper adjustment we hoped to maintain a permanent roasting zone near the center of the column, the ore moving downward as roasted. However, we experienced difficulty in getting the moving mass to roast properly, and also other troubles of a mechanical nature developed. Mr. Christensen at this time disposed of his interest in the process to G. H. Dern.

We continued experimental work on a number of machines of varying design. Up to the time construction work began on the Mines Operating Co. mill no satisfactory continuous roaster had been developed, so we decided to install an intermittent roaster, which our experience had demonstrated would give a satisfactory product for subsequent leaching operations. Accordingly, we installed roasters similar to the one shown in Fig. 2, the construction and operation of which will be described in its proper place in the description of the plant.

The Plant

The Mines Operating Co.'s plant is installed in the old concentrator of the Ontario Silver Mining Co. An outline of the present scheme of treatment may be followed by reference to the accompanying flow sheet (Fig. 1).

A second-motion electric hoist, operating 1.2-ton skips in balance, delivers the ore from the haulage tunnel of the mine to a bin above the crusher. From here it is fed by a Stephens-Adamson apron feeder to a trommel with $1\frac{1}{2}$ -in. openings. The undersize of the trommel passes through a revolving drier, where the moisture is reduced to about 5 per cent. The oversize of the trommel falls upon a picking belt, where mine wood and waste are removed. It then passes through a No. 5 Gates gyratory crusher and, joining the drier fines, is elevated to the crushed-ore storage bin. This bin has a capacity of 400 tons of ore. Sections of the same bin are respectively used for salt and coal storage. From these bins the ore, salt, and coal are fed, in the required proportions, upon a belt conveyor and delivered to the rolls for final crushing. The feeders by means of which the proportioning and mixing are done are of the plunger type, the feed being varied by changing the eccentric throw. This type of feeder works very well for the salt and coal dust, but is not very satisfactory for ore, on account of rapid wear. The mixed ore is crushed in two sets of 15 by 36 in. rolls and is elevated to the mixed-ore bins above the roasters.

There are eight shaft roasters of the intermittent type, similar to the one shown in Fig. 2. The roaster shown is 10 by 10 ft. inside the walls. The walls are reinforced concrete 10 in. thick. About 2 ft. above the

bottom is a wood grating, supporting a layer of coarsely crushed rock. The space below this grate forms an air chamber. The grate supports the roasting charge, and also distributes the air blast.

The method of operating the roaster is as follows: A special starting "mix" of about 1 ton is prepared in front of the roaster. A layer of coal dust mixed with oil is then spread over the gravel floor of the roaster.

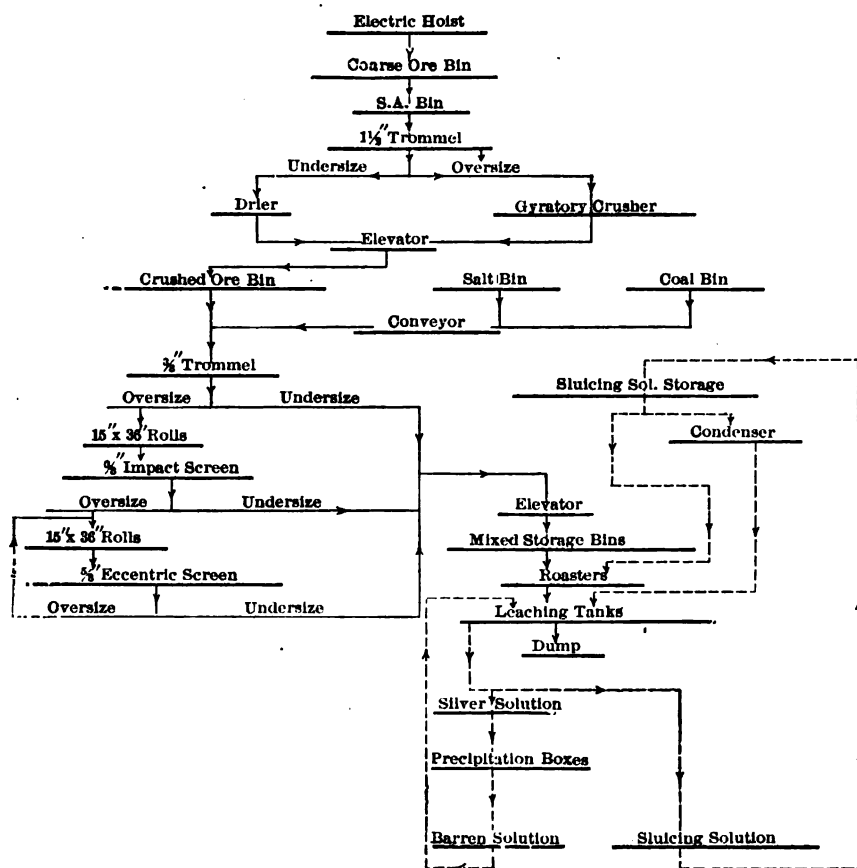


FIG. 1.—FLOW SHEET OF MINES OPERATING CO.'S MILL, PARK CITY, UTAH.

This is ignited and sufficient blast admitted to burn it rapidly to glowing coals. The special mix is then shoveled evenly over the surface. By the aid of the air blast, the coals ignite the fuel in the ore, which begins to roast. When the starting layer has roasted through, so that the fire appears on top, the air is shut off for a few minutes, while the first charge of about 5 tons is dropped from the mixed bin gate into the roaster. This is spread out by hand and the air again turned on. The roast is allowed to

proceed until it appears on the surface of the charge. Then a second charge of about 10 tons is dropped on and 4 hr. later a final charge of about the same amount. This brings the total depth of charge up to about 7 ft. Under normal conditions this will roast through in 8 to 12 hr.

When the roasting zone has reached the surface at all points the air blast is shut off. The discharge door is opened and a sluicing apron inserted, to connect with the launders. The hot roasted ore is then sluiced out with mill solution. The sluicing nozzle is made of hard wood and has

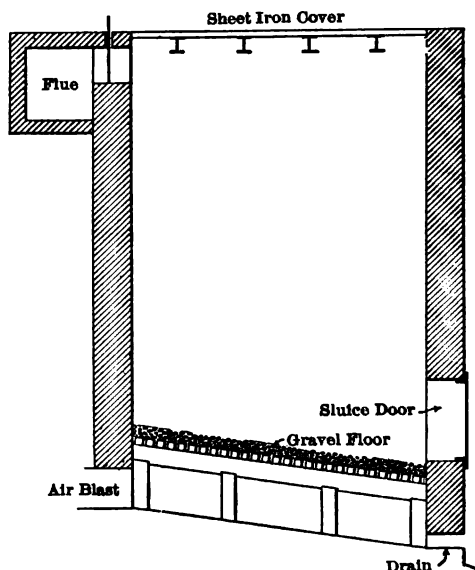


FIG. 2.—SHAFT ROASTER.

a $1\frac{1}{2}$ -in. nozzle opening. It operates under a head of 45 ft. The nozzle is suspended in position in front of the discharge door. The operator stands at one side, to avoid the numerous steam explosions, and directs the nozzle with a long rod. Under normal conditions it takes from 1 to 3 hr. to sluice out a roaster charge of 24 tons.

From the roasters the hot ore mixed with mill solution passes by means of launders to the leaching tanks. The leaching tanks are 20 ft. in diameter and 12 ft. deep. There are six of these, each holding 135 tons of ore. In the bottom of each tank is a 12-in. round discharge hole for sluicing out the tailing. This is closed by a turned wood plug, having a 4 by 4 in. stem extending to the top of the tank. For convenience in sluicing out, the discharge hole is surrounded by a box extending to the top of the tank and provided at intervals with 12 by 12 in. sluice gates.

Of the various filter bottoms tried, one made of 4-in. drain tile has

side. This tile system connects at one side of the tank with a 3-in. wood-pipe solution line.

The solution lines from the leaching vats lead to a distributing box. Here the solution is directed either to the "weak" tank or the "silver" tank, depending upon its metal content. From the weak storage tank the solution is pumped back for sluicing purposes, while from the silver tank it goes to the precipitating boxes.

The iron precipitation boxes are similar in design to the wood zinc boxes sometimes used in cyanide mills. They are larger, however, and so constructed that the iron tie rods are not in contact with the acid-soaked wood. From the precipitation boxes the solution passes to the barren sump and is returned as a leaching solution to the vats.

New Features

Having followed the ore treatment through in a general way, we will now return to discuss more in detail certain features that are more or less new.

Importance of Proper Mixing

Perhaps the most important step in the whole operation is the proper mixing of the ore preparatory to roasting. Roasting for lixiviation requires a very close regulation of temperature. The best results on many ores are obtained between 600° and 700° C. In any case, to sinter parts of the ore is to render it unfit for subsequent lixiviation. The three chief factors which determine the temperature are: (1) the percentage of fuel in the mixture, (2) the percentage of moisture in the mixture, and (3) the amount of air blown through the charge per unit of time. The first of these three is much the most important.

Coal dust is used as fuel. The present supply costs 25c. per ton f.o.b. cars at the coal mine. Since this class of material takes a cheap freight rate it is the most economical fuel to use under our conditions. The amount required varies from 2.4 to 3.0 per cent. of the weight of the ore being roasted.

The ore as it comes from the mine is very wet. The drier is supposed to reduce the moisture to about 5 per cent., but the variation is considerable. We might suppose that an increase in moisture would lower the temperature of the roast. It has exactly the opposite effect, however, and it is necessary to reduce the percentage of fuel as the moisture in the ore increases.

The mill was designed to treat 150 tons per day. It is handling considerably more than this, the average for the month of March being 166

tons. This tonnage is mixed and rolled on one shift. The mixing is in charge of a trustworthy man who checks up the feeders by weighing the output of each frequently.

The New Holt-Dern Roaster

During 1913, a new continuous roaster was developed, which overcomes the difficulties experienced with previous machines. A full description of the roaster will not be attempted in the present paper. A few of its general characteristics and the results obtained will be noted.

We installed a commercial-size machine at our plant in December, 1913. Except for a few brief delays for changes, it has been in continuous operation. In the Holt and Dern roaster the column of ore moves down at intervals as the roasting zone travels upward. The air blast travels in an opposite direction to the ore. In this way the air passes first up through the hot roasted ore and becomes highly heated. It then passes through the roasting zone where active combustion of the fuel in the ore is taking place, and finally through a layer of moist unroasted ore. Hence when it leaves the roaster it is fairly cool and entirely free from dust.

Our experience at this plant has demonstrated the following advantages in this roaster over the intermittent type already described:

Comparative Cost Data per Ton of Ore Roasted

	Old Roaster	Holt & Dern Roaster
Labor.....	\$0.380	\$0.054
Power.....	0.049	0.045
Repairs.....	0.026	0.020 (estimated)
Starting oil.....	0.038	
Coal dust.....	0.046	0.046
	\$0.539	\$0.165

A series of extraction tests for a period of five weeks gave an average of 80.3 per cent. of the silver for the old roasters and 86.4 per cent. for the new roaster. This is on the coarse crushing we have found most economical in the old type of roaster. Following is a screen sizing test on the ore fed to both roasters:

Size	Per cent. by Weight
+ 1/2 in.....	10
+ 1/4 in.....	22
+ 1/8 in.....	33
+ 1/14 in.....	15
- 1/14 in.....	20

The ore treated is a dense one and the recovery would be materially increased by finer crushing. With the old type of roaster this is not eco-

nomical, while in the new roaster 10 mesh, and even finer product, may be handled very well.

The salt used in both cases was equal to 7.5 per cent. of the weight of ore roasted. We are of the opinion that this can be economically reduced in the new roaster, but this has not been determined.

When compared with the roasting furnaces commonly employed for preparing ore for leaching purposes the following advantages may be pointed out.

1. There is no perceptible volatilization loss of the valuable metals.
2. There is no dust loss.
3. An ideal product for percolation is furnished even when the raw ore contains much slime.
4. Only a low percentage of an inexpensive fuel is required.
5. A high recovery without fine crushing is possible.
6. The roaster gases are concentrated, cool, and easily condensed for leaching purposes.

The Leaching Process

The leaching process is very simple. There is in fact but one mill solution, which is designated as "silver," "weak," or "barren," depending on its value in silver. The solution is made up merely of soluble salts from the roasted ore, carrying in addition about 4 lb. of free acid. The acid is in part supplied from the condensed roaster fumes, the remainder being added as sulphuric acid. At such times as the ore carries a considerable percentage of pyrite no extra acid is required. Of the salts taken up by the solution from the roasted ore, during leaching, NaCl forms the greater part. Chlorides and sulphates of the various other metals are also present.

The sluicing out of the roasters into the leaching vats is done with weak solution. This dissolves the greater part of the silver and copper from the hot roasted ore while conveying it to the leaching vat. The solution passes from the vat as the "silver" solution and most of it is pumped to the precipitation boxes. When the leaching vat is full it is leveled off, and leached with barren solution for 24 hr. It is then washed with water down to from 5° to 10° B. (specific gravity, 1.036 to 1.075) and sluiced out. The washing is regulated so as to keep the solution at the proper density. Experiments have shown this to be about 24° B. (specific gravity, 1.200). The excess barren solution is run to waste from time to time.

The best results are obtained by leaching with warm solution. During the summer months the heat from the roasted ore is sufficient to maintain a temperature of from 30° to 40° C., but in cold weather steam is blown into the solutions. This is best done in the precipitation boxes, as maximum efficiency is obtained by heating the solutions at this point.

The cost of leaching varies considerably. The minimum of 17c. was attained during the summer when the heating plant was closed down, and the roasters were making the necessary acid. On the other hand, during the month of March when it was necessary to buy both acid and coal the leaching cost reached the high figure of 43.8c.

Precipitation

The precipitation boxes are packed with scrap iron of every description. Thin sheet scrap, old screens, etc., are desirable on account of the large surface presented. The metals in solution are deposited on the iron in accordance with their position in the electromotive series; that is, the metals of greatest potential difference are deposited first. This segregation of the metals, however, is not sufficiently clear cut to be of value in their separation.

A general cleanup of the precipitation boxes is made once each month. An intermediate is sometimes necessary on account of the boxes clogging.

The method of cleanup is rather crude, due to the present layout. The loosely adhering precipitate is brushed from the scrap iron, put into filter boxes and washed. It is then dried and sacked for shipment.

A partial analysis of the product is as follows: Au, 1.78 oz.; Ag, 6,868.5 oz.; Pb, 13.2 per cent.; Cu, 36.94 per cent.; insoluble, 7.5 per cent.; Fe, 2.5 per cent.; S, 1.6 per cent.; Zn, nil.

Our original plan was to precipitate the gold and silver on cement copper in an acid-proof press. The copper used could then be recovered on scrap iron as cement copper and returned to the precipitator. Two difficulties presented themselves. In the first place, it was hard to form and maintain a uniform cake of cement copper on the filter leaves. Secondly, the filter cloth and cake soon became impervious to the solution.

We are at present developing a method of handling the copper precipitation of the gold and silver, which promises to be both simple and efficient. However, as this is still in the experimental stage it will not be considered here.

The consumption of scrap iron amounts to about 1 lb. per pound of product recovered. The grade of scrap used costs \$5 per ton delivered at the mill or 6c. per ton of ore treated. The total cost of cleanup, including drying and sacking, amounts to 6.5c. per ton of ore milled.

Pumps

Our practice of sluicing out the roasters makes it necessary to handle large volumes of solution. The mill solutions are corrosive. In addition

combination, and pumps on the market, of suitable material, are either too small or too expensive to be considered. Our sluicing alone requires a continuous supply of 140 gal. per minute, which must be elevated 89 ft.

After considerable experience with pumps that work for brief periods, and others that work not at all, we decided to install the Pohle air-lift system. These air lifts are constructed of wood pipe and a rubber air

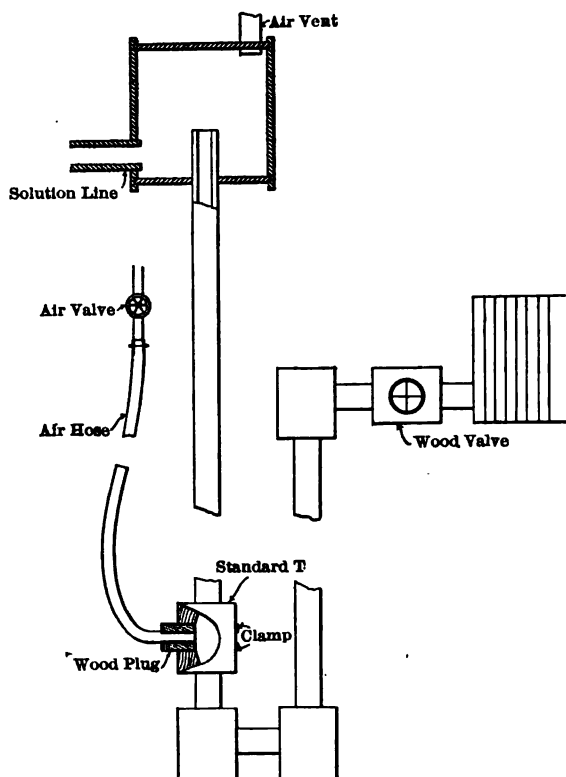


FIG. 3.—POHLE PUMP.

hose as shown in Fig. 3. Wood stave pipe, with bands well protected with asphalt or similar material, withstands the solutions very well.

The highest lift of 89 ft. is made in three steps. The highest single lift is 37 ft. and this determines the air pressure used, which is about 27 lb. The air-lift pumps have been in continuous operation for over a year and have given complete satisfaction.

Every department of the mill suffers the inconvenience of being housed in an old plant not designed for the process. Furthermore, several fea-

has taught us many things that would add to the efficiency of a plant we might now design.

Conclusions

The roasting process is not limited to chloridizing but may be advantageously applied to oxidizing gold ores for cyanide treatment, and oxidizing and sulphatizing copper ores for subsequent leaching.

However, it is our opinion that the most favorable field of application is in chloridizing roasting. In this class of roasting we not only have the benefit of cheaper operating cost, but, in addition, have overcome the volatilization loss, which in the past has been a serious difficulty.

We have tested out samples of ore from 38 different mines. These have given us a large variety of combinations. In many cases the recovery of gold, silver, and copper has been well above 90 per cent. The ideal combination, we may suggest, is a siliceous copper-silver ore carrying 6 to 10 per cent. pyrite, and favorably located with respect to a salt supply. Under such favorable conditions, the entire cost of milling in a well-designed plant will be less than \$1 per ton, on the basis of 150 tons per day. When the ore carries gold it is necessary to carry free chlorine in the leaching solutions and this adds a few cents to the cost of treatment.

Ores high in silica, iron, and manganese chloridize readily. A small percentage of lime does no harm as it is readily neutralized in the roast. A large percentage, however, is detrimental.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Ancient Auriferous Gravel Channels of Sierra County, California

BY MARK N. ALLING, DOWNIEVILLE, CAL.

(Salt Lake Meeting, August, 1914)

INTRODUCTION

A FEW years ago, gravel mining in the ancient river beds of Sierra county was the principal industry of the county. Operating drift mines were very numerous and many large hydraulic mines were in action. Eventually the State laws put an end to hydraulic mining, it being claimed that the silt carried down the streams to the valleys filled the rivers and caused winter floods, which were damaging to the surrounding farms. The ancient river deposits which could easily be found were, in the course of 40 years, worked out or practically so. The prohibition of hydraulic mining was a severe blow to the industry in general. The deposits were not developed and the gravel-mining industry practically stood still for many years. In Sierra county, nearly all the hydraulic mines were at the points where the ancient channels were exposed to view by reason of the modern stream erosion, or, in the miner's language, where the channels "enter" or "break out of" the ridges. The claims covering them also covered large areas in which were long segments of lava-capped channels, suitable for drift mines, and the points being hydraulicked were usually the controlling points of attack. Men who have followed hydraulic mining for any length of time seldom are content to work a drift mine. Knowing the great value of their deposits and feeling that they had been robbed by the State, these men have stubbornly held to their claims. The gold that is carried in these gravel channel deposits is more or less scattered throughout the depth and width of the deposit; the percentage that can be recovered by the process of drifting ranges from 50 to 75 per cent. of the total. In hydraulicking, practically all of the deposit is put through the sluices, and as the cost per cubic yard is only about 10 per cent. of the cost of drifting, the deposits, if classed as hydraulic mines, are much more valuable than if classed as drift mines; hence the old timers have held their claims (now worthless as hydraulic mines) at prices which the drift-mining industry could not stand; and such mines

owners, realizing the truth, have sold their mines for reasonable figures, and many of these properties are now being reopened.

But there is another and very important source of future drift mines. Only the deposits which could easily be discovered have been found. There are many unworked segments of ancient channels in this rich district; some have been discovered in a general way but never exactly located or traced through the country; still others have been partly opened up, requiring the investment of more money than the owners possess. The district is isolated, timbered and brushy (especially in the vicinity of the moist gravel deposits), and hence the more difficult forms of prospecting have been left to the coming generations.

The long period of comparative inaction in the gravel-mining industry has given rise to the common opinion that the gravel deposits are all worked out; but such is not the case. The field is young and large, and yet to the stranger passing through the county for the first time the impression is given that the district is an old and exhausted one. This is principally due to the fact that in the boom days, when the shallow placers were yielding fortunes for the thousands who worked them, large towns sprang up, only to be deserted in whole or in part upon the exhaustion of the shallow placer deposits and the desertion of the hydraulic mines.

Following the exhaustion of the discovered placers, and the suspension of hydraulic mining, came the search for the sources of the gold. Mining companies began to reopen the old drift mines, exploring for the quartz ledges, and with such remarkable results that the annual gold output of the county was increased last year by about \$1,250,000. Other companies are now exploring for the undiscovered channel segments.

HISTORICAL GEOLOGY

The Ancient Drainage System of California

It is claimed by the best geological authorities that at the end of the Tertiary period the entire county now occupied by the Sierra Nevada mountains had been leveled down into the form of a great plain (peneplain). The streams were filled with quartz gravels, almost to the exclusion of softer rocks, and these streams were rich in gold. The original surface undoubtedly contained much more quartz than is evident upon the general surface to-day. The quartz, being the hardest and most resistant rock of the district, was battered about through time, and while vast quantities of it were ground to sand, enough survived to fill the streams as gravel, while its softer associates were converted to sediments and washed into the valleys. Even at the present time, in certain high

localities where the sheltering lavas have but recently been removed, we find wide zones containing large ledges and massive bodies of solid quartz.

The flow of these rivers was interrupted, first by an unknown agency and later by the intermittent flows of three distinct types of lava.

That there was an interruption prior to the coming of the lavas is evident from the fact that we find two systems of channels, running in parallel courses through the same ancient valleys, and both of prevolcanic age; the one system being secondary to the other yet both buried by the same initial ejection of acid lava ash. This condition has come under my observation, and I find that H. W. Turner suspected something of the sort when studying the gravels of northern Sierra county, 20 years ago.

The blocking of the original channels may have been due to one of three causes:

(a) To an uplift of the coast line, having the effect of lessening the grades and thus causing the channels to choke with débris and forcing the waters to seek new courses; or (b) to local undulatory movements of the earth's crust. As we have some direct evidence of such movements, cause (a) seems more probable than (b).

2. To débris washed into the rivers during great storms which may have accompanied or immediately preceded the long period of volcanic disturbances.

3. To a light ejection of volcanic ash, temporarily drying up the rivers and causing the waters to choose new courses, although still confined to the same ancient basins or valleys; the nature of this material being such that eventually it was washed away, leaving no boulders or other easily recognized evidence of its visit. Thus the "unknown agency" may really have been a very premature warning of the approach of the volcanic age.

Regardless of the cause, there was a long uninterrupted period of stream action following the end of the more ancient system of prevolcanic rivers, during which period the secondary channels of this age had time to cut their new beds, from a few feet to 100 ft. or more deeper than the level of the originals.

While the lava outbursts were intermittent, and this was true to a greater degree in some districts than in others, the entire country, in time, was covered with three general types of lava.

1. The rhyolite, a light-gray, fine-grained, acid lava ash. This was the first lava to be thrown out, and, as compared with the other lavas, in very limited amounts.

2. The andesite, a heavier, semi-basic, darker, mottled lava. It was ejected first as tuffs and mud flows, mixed more or less with the rhyolite, then as breccias, and later as massive flows of heavier and darker boulders and mud, semifluid in consistency. It flowed into the depressions, ravines, rivers and valleys.

3. The basalt, a dark, almost black, heavy, basic flow mass, in places

so fluid that upon sudden cooling a natural glass (obsidian) was formed. This lava was the last to be extruded, pouring out as a capping over the others.

The darker the lava, the more basic, and the more basic the more fluid, so that the basalts flowed the farthest from the craters.

The rhyolite was washed into the river beds from the hillsides, covering the gravels and forming what is known to the California miners as "pipe clay." On the top of this lie the andesitic tuffs, breccias, and muds with the hard resistant basalt forming a protecting roof to the ancient river deposits, and where the river valleys were not entirely filled this took the form of long, narrow, level surfaces, conforming to the sinuosities of the underlying channels. In some portions of the State subsequent erosion of the adjacent, softer country rock has resulted in leaving these lava-covered areas as long sinuous mesas, standing high above the surrounding country, so that the ancient channels have been little disturbed by modern erosion. In all parts of the State this condition existed up to a certain late period; but in the northern portions subsequent lava flows have brought about a different effect; the modern streams having cut to pieces the ancient system.

The action of the craters was intermittent. In Sierra county there were at least three long periods of rest between the outbursts, during which times the streams were endeavoring to reinstate themselves.

It appears that the longest rest periods, with the exception of the last one, were at the beginning of the disturbances, for we find that the intervolcanic streams of the first and second rest periods usually flowed in the same ancient valleys occupied by the original rivers and on one side or the other of the same, while it is not uncommon to find a secondary channel on each side of an original. Their courses were governed by the fact that the valley was not filled with the lava; the heavier lavas sought the center of the depression, and being more viscous, upon cooling, formed a harder and more resistant capping than the lighter, looser and drier material that had been crowded to the edges. Due to this same weakness, this material along the edges was more easily eroded by the waters rushing from the hillsides, so that the new intervolcanic channels are found skirting the edges of the original channels, or crossing them now and again. The point of crossing is usually at the place where two branches of the original river came together, or where there is a sudden turn in the original river. The course of any stream generally changes at the point of confluence of a large tributary; the extra width of the valley at such points would allow the cooling lava to settle more than in the narrow places, and the new channels, having a higher and wider valley to begin with, would naturally flow straight until turned by the opposite bank.

During each period of rest there was evidently some segregation of

materials in the craters, for preceding each extrusion of heavy lavas there was an ejection of more or less ash. This rule of action seems to have been followed until well along toward the last of the volcanic period; for with the exception of a few of the latest intervolcanic channels we find, in all of these ancient channels, some kind of an ash, covering the gravels, and in various stages of decomposition. The later the age of the channel the darker and usually the coarser is this ash, and, so far as my observations go, the quantity is also decreased.

A white pipe clay does not cover the gravels of an intervolcanic channel; neither does a dark, flinty, granular or "chocolate" pipe clay cover a prevolcanic channel. It was long held that the color was due to chemical changes after the ash was ejected, brought about through alteration of the constituent minerals under the action of the infiltrating waters, etc., but, in my judgment, this is not the case. The color and texture, in most cases, depend upon the relative age in which the ash was ejected (although of course alteration has taken place in some localities), the acidity depending upon the period, and the color upon the acidity.

Relative Economic Value of Ancient Channels.—The history of drift mining has proved that these ancient channels are valuable in the following order:

The old, original, prevolcanic, high-quartz gravel channels are always rich; likewise the secondary, prevolcanic, quartz gravel channels. The intervolcanic channels of the earlier periods are generally rich, depending upon their position relative to the prevolcanic streams and upon the character of the country rock. The third and fourth age intervolcanic channels are very uncertain, as their values depend principally upon their association with the older channels, having had very little chance to collect minerals from the underlying bed rock. The fifth-age channels, or modern streams, were rich, having obtained their gold from the long stretches of ancient channels cut away and from the ledges eroded below the level of the same, as the modern streams have enjoyed a very long period of rest, and have worn their way down more than 2,000 ft. into the mineralized country rock below the level of the ancient streams.

The Uplift of the Sierra Nevada Mountains.—The most widely accepted theory, among modern geologists, is that during the volcanic period, or immediately following it, the entire eastern portion of what is now California was elevated many hundreds of feet. The granites and other igneous intrusives forced upward the overlying sedimentary rocks, the country faulting along a north-and-south line approximately the same as that which now marks the boundary line between the States of California and Nevada; and the old peneplain, with its lava-filled rivers, was tilted to the westward on about a 2° slope.

The effect has been that the original grades of the southwest-bound rivers have apparently been increased by about 1 per cent., or about 50

ft. per mile; so that channels which originally flowed on a grade of say 100 ft. per mile are now found beneath the lava with a grade averaging 150 ft. per mile. In localities where the channels flowed northerly the opposite effect took place.

The country, however, was elevated *en masse*, so that the ancient channels still lie in their proper relative elevations and positions. Local faulting has been discovered in southern Plumas county, but in the Sierra county drift mines little or no faulting of a serious nature has been met with.

Following the uplift, the modern rivers flowing westerly cut their way down through the lavas and old channels and deep into the country rock. This erosion has been a rapid one, as the streams flow in deep narrow gorges, and the work of the waters was no doubt aided by the action of glaciers.

The modern rivers which have effected the Sierra county district are the middle and north branches of the Yuba river, with their numerous tributaries.

The Economic Geology of Sierra County

Sierra county is about 10 miles long, east and west, and about 20 miles wide. The summit of the Sierra Nevada mountains passes through the central portion of the county in a northerly and southerly direction. All the mines lie on the western slope of the mountains.

In the extreme western part of the county the rocks are principally black, highly metamorphosed clay slates, designated by the government geologists as the "Delhi slates." They are underlain by granites which are exposed in the bottoms of the deep canyons. Porphyry and diorite dikes are common in the slates. A few good mines have been found in this formation.

Lying just east of the Delhi slates is a zone about 10 miles wide, consisting of alternating belts of "Calaveras" formation¹ and serpentine

¹ The "Calaveras formation" is a general term applied to the older, or Carboniferous, sedimentary rocks of the "Bed Rock" or "Auriferous Slates" series. It is composed of seven different types of sedimentary rocks, which have been given names indicating the locality in which they are most common:

1. The Clipper Gap formation; composed of clay slates, cherts and limestones.
2. The Delhi formation; black siliceous rocks, rarely schistose.
3. The Cape Horn formation; fissile clay slates.
4. Relief formation; cherts and quartzites.
5. Blue Canyon formation; fissile clay slates, quartzitic sandstones, cherts, and limestones.
6. Blue Canyon formation; contact-metamorphic rocks, chiefly mica schists.
7. Limestone lenses; in Carboniferous and Juratrias formations.

While the Delhi slates are themselves included in the Calaveras formation series, the rocks most generally called the Calaveras formation by the miners in the Forest City district, are the sedimentary schistose rocks; often referred to simply as "slate."

the range, and is the line of weakness. It is highly fractured and mineralized and forms what is known as the "Father lode" of California, in contradistinction to the "Mother lode," which occupies a similar position relative to the range in Amador, Calaveras, Tuolumne, and Mariposa counties. In this zone are found most of the quartz mines and practically all of the gravel mines.

Next to the east lies a zone of quartz porphyries, granites, slates, and occasional small dikes of serpentine. Here are found the Sierra City quartz mines, some of which have been large producers. The zone is from 3 to 5 miles in width.

Between the Sierra City quartz-porphyry zone and the summit few miles of importance have been found. Beyond the summit no mines are found until the State of Nevada is reached.

The Father Lode, or Contact Zone.—The Calaveras slates of this zone strike north and south and are tilted at a high angle to the east. The amphibolite schists and serpentine intrusions range in width from a few feet to 2,000 ft. These lenticular bodies of serpentine strike northerly and southerly, paralleling the general strike of the bedding planes of the sedimentary rocks. Some of them extend through the district for many miles; others are but short bodies.

The slates through which these intrusions are found have been thoroughly shattered, tilted, altered, and in some cases highly metamorphosed, especially near the contacts. The zone has been so thoroughly fissured and mineralized as to cause the government geologists to make special mention of that fact.

Quartz ledges are commonly found following directly along the contacts, lying in the schists or slates and paralleling the contacts or extending from the contacts out into the slates as true fissure veins. The shattered slates are full of quartz lenses, pockets, and veins of variable size and richness, but all carrying gold enough to hold the attention of the prospector.

In a general way these long, parallel, hard serpentine lenses are separated by the alternating belts of the softer and heavily mineralized slates, and here the ancient streams have cut their beds. In wending their way south and west, they seem to have experienced considerable difficulty in crossing the serpentine bodies, and often followed along the east side of the bodies until the end of the lens was reached, when the channel would swing to the west until another serpentine body was encountered. Where the channels did cross the serpentine the gold did not lodge, as a rule, while the uneven, decomposed surface of the slates seems to have formed an ideal gold-catching bed rock. Nearly all of the ancient rivers of this district followed this mineralized zone. The gold and other heavy minerals lodged in the beds of these streams.

No doubt the greater part of the placer gold came from the general erosion of the great quantities of mineralized material, but the very large pieces of gold found all along in the "Great Blue Lead," which passes through this district, evidently came from the ledges. Many of these ledges were exposed in the workings of the gravel mines, and several old mines have been reopened and developed as quartz mines; this proving that the large gold slugs do not travel far from their sources.

The lava mantle which covered this country still lies several hundreds of feet deep on the ridges, hiding the contacts and ledges and rendering prospecting slow and costly.

The early drift miners paid little attention to the quartz ledges; they knew little about handling the ore and they were not prepared to do it. The gravel was rich and they were making money. They did not care to divert their energies and, as they expressed it, "waste their time in trying to get it out of the hard rock." By simply washing the gravel in their sluices they could gather from the riffles the clean gold dust and nuggets. So the ledges were passed and many of them forgotten, and yet the memory of some of them has led to important discoveries in the present day.

Rich ledges were found in the gravel diggings of the Tightner, Red Star, Rainbow, Sixteen to One, Dead River, South Fork, North Fork, Bald Mountain, Ruby, Telegraph and many other drift mines.

In the Bald Mountain mine a ledge was discovered which has given rise to many "fairy tales" of great riches, and upon investigation I have secured enough reliable data to cause me to believe that success would follow the reopening of the tunnel for the exploration of this ledge. Success in this undertaking is still more assured from the fact that about 1 mile of unworked channel awaits development in this ground and could be reached by extending the tunnel which was run for the quartz. This ledge is from 2 to 3 ft. in width. It crops up through the bed rock and into the gravel from 1 to 2 ft., and strikes northerly and southerly, crossing the channel diagonally, since the channel was there coursing southwesterly. The ledge was followed in the breasts for from 300 to 400 ft., and the gravel just below the crossing was the richest of any worked in the 7,000 ft. of channel exhausted by the company. An unusually large number of large chunks of gold were found, many of them still being partly imbedded in the quartz matrix.

Sulphides were found in the gravel at this point and interfered with the sluicing; they were shoveled out as a nuisance. It was not until recent years, when so many rich strikes were being made in the neighboring ledges and the gold found was so often associated with arsenical sulphides, that it has dawned upon the owners that these sulphides were probably valuable.

The Bald Mountain Co. bought 9,600 ft. of Oregon creek as a tailings claim after that creek had been completely worked out by the pioneers.

the tailings. These had been carried through the sluices with the coarser gravels or had been forked out with the boulders.

The ledge itself was considered a nuisance by the miners, as it stood up in the gravel and interfered with the breasting and timbering. I have been told that free gold was seen in the quartz in the ledge in place.

The Bald Mountain and Ruby mines were noted for the great amount of coarse gold contained in their gravels. This feature of the channel which flowed through them did not cease when the Bald Mountain was passed; but the channel continued to yield large rough nuggets in the claims lying south of the Bald Mountain and even as far south as Alleghany and beyond. These nuggets did not all come from the Bald Mountain and Ruby claims, by any means, but were liberated from the ledges followed and crossed by the channel.

The Tightner, Red Star, Rainbow, and Sixteen to One have been reopened and now rank as quartz mines of value; the Tightner mine at the present time is probably paying the biggest dividends of any mine in the State. In the last year it has produced about \$1,000,000 in gold. From December, 1912, to July, 1913, it yielded in round numbers, \$700,000; in July, 1913, about \$50,000 was taken from a small stope 9 ft. square.

The South Fork, North Fork, Dead River, and Telegraph are now being developed for their quartz ledges.

A local company is making preparations to reopen the Ruby and Bald Mountain mines this summer, both for the ledges and for the gravel which they are known to contain.

To the north of the Ruby, and below the lava cap, the Downieville-Forest City contact has been traced down through Slug canyon, through Downieville, and up the next mountain side to the lava cap again, a distance of about 6 miles. The Loosner group, Carson group, Tripple Pocket group, Frost claim, York mine, Donderro claim, Bessler mine, Good Hope group, Oxford mine, Bosch claim, and Standard mine group cover this entire distance. Coarse gold has been found all along this contact from Forest City to the Standard mine, a distance of about 8 miles, and some of the above-named claims and mines have produced large sums. Slug canyon, which follows this contact for about 2 miles, derived its name from the character of its gold. This same contact passes through the Ruby mine and the Bald Mountain mine, through Forest City and on south to Alleghany. The Tightner mine lies due south from the Bald Mountain ledge, and the Tightner ledge strikes due north, but as the intervening country is lava capped it is impossible to say that the Bald Mountain ledge is the Tightner ledge.

In the Ruby mine the ancient channels followed this contact, and on the old company maps may be seen the words "Line of coarse gold," in-

dicating the approximate line along which the large slugs were found. Pieces ranging from 10 to 45 oz. were commonly found here and one nugget was found which weighed 201.56 oz. A model of this nugget is in the museum of the California State Mining Bureau. We have record of at least 12 nuggets ranging in weight between 30 and 170 oz. found in the South Fork or Live Yankee mine, just south of the Bald Mountain. Large gold slugs were found in the Tightner gravel, and after a lapse of 40 years the mine was reopened. The ledge proved to be richer than could reasonably be expected of any ledge. Adjoining the Tightner is the Sixteen to One mine, where the gravel also yielded large specimens. The ledge was rediscovered a short time ago and considerable gold has already been extracted, the last strike (a \$30,000 pocket) being made about six months ago.

Just south of these properties the Rainbow gravel yielded large slugs. One quartz boulder was found which contained \$36,000 in gold. The ledges were found in the breasts. After 30 or 40 years this mine was reopened and something over \$300,000 was taken from the quartz ledge; one slab of quartz being found which was calculated to contain \$20,468 in gold.

The Oriental mine, situated on a serpentine contact about 1 mile west of the contact just described, has produced about \$3,000,000, and it is commonly reported that one pocket yielded \$740,000. The Plumbago mine, about 3 miles to the southeast, has produced about \$2,000,000, of which \$250,000 came from one stope. The Brush Creek mine, situated on the Mountain House serpentine contact, about 4 miles to the west, yielded \$400,000 from one shoot. The Tripple Pocket mine, on Slug canyon, and on the Forest City-Downieville contact, and the Gold Bluff mine, near Downieville, were heavy producers. The Sierra Del Oro or Old Ironsides mine lies about 2 miles to the east of the Ruby mine and is the latest big strike in the district; the ledge is just being developed, but several thousand dollars have already been recovered from specimen ore.

The Downieville-Forest City and Alleghany contact line has proved to be the longest, most productive, and most constant and continuous gold producer of any contact line in the entire county.

The Ancient Channels of Sierra County.—The ancient channels of Sierra county all flowed southerly, being northern tributaries to the main ancient river which flowed eastward in Nevada county from North Bloomfield to Smartsville. The rivers flowed in parallel courses and lie so close together that it would be impossible for them to have all been in action at the same time. Moreover, their relative positions, elevations, grades and characteristics mark them as having distinct periods of action.

The lava flows, which were important enough in this section to force the channels to choose new courses, evidently did not extend as far south as the great valley occupied by the main trunk stream; for that great

for great distances.

Hence we find all the Sierra county channels of the various periods leading into the same ancient drainage basin as tributaries.

In the ridge north of the North Yuba river we find 10 prevolcanic (and at least 1 intervalcanic) channels coursing southerly and all within a section of country only 12 miles in width. Beginning on the east and naming westerly, these channels are known locally as:

Graycroft and Wide Awake.	Prevolcanic.	White and blue quartz gravels.
Excelsior and Monte Cristo.	Prevolcanic.	White and blue quartz gravels.
White Bear.	Prevolcanic.	White and blue quartz gravels.
Telegraph.	Secondary and probably intervalcanic.	White and blue quartz gravels.
Hilo.	Prevolcanic.	White and blue quartz gravels.
Eureka.	Prevolcanic.	White and blue quartz gravels.
Brown Bear and St. Charles Hill.	Prevolcanic.	White and blue quartz gravels.
McMahon's Ranch.	Prevolcanic.	White and blue quartz gravels.
Lost River and Bunker Hill.	Early secondary.	Quartz and quartzite gravels, darker colored than above.
Brandy City and Camptonville.	A very large prevolcanic.	White and blue quartz gravel channel, extending well up into Plumas county and forming a large system of its own.

These have either been mined or are now being developed, with the exception of the Hilo channel, which is a tributary to the Eureka Lead.

There is strong evidence that the Monte Cristo channel, which is unmistakably prevolcanic, is secondary, for it appears to have cut away the higher prevolcanic White Bear channel. The channel that "enters" the ridge south of Downieville at the City of Six mine, and known as the "Deep Rock Creek channel," is a secondary channel of the prevolcanic period.

In the ridge lying south of Downieville, or north of Forest City, there are 20 ancient rivers with subordinate tributaries, all lying within a district about 18 miles in width.

Since some of the channels found in the ridge lying north of Downieville undoubtedly united in the 6 miles intervening between these ridges, they would be represented in the south ridge by a lesser number, and the writer is inclined to believe that at least 11 of the 20 channels either came from other sources, are secondary, or are intervalcanic rivers, owing their origin in part to the 6 miles of intervening territory now eroded away by the North Yuba river.

Beginning on the east and naming westerly, these channels are known locally as follows:

Maple Grove.	Prevolcanic.	White and blue quartz gravel.
Bald Mountain Extension.	Prevolcanic.	White and blue quartz gravel.
City of Six and Deep Rock Creek.		White and blue quartz gravel.
Bald Mountain Channel, or Great Blue Lead.		White and blue quartz gravel.
Deep Bald Mountain.	Intervolcanic channel of first age; at headwaters gravel is volcanic, but after leaving the Bald Mountain ground it contains so much quartz gravel that it could easily be mistaken for a quartz gravel channel.	
North Fork.	Prevolcanic.	White and blue quartz gravel.
Redding and Young America.	Prevolcanic.	White and blue quartz gravel.
Corotoman, Upper Lead.	Prevolcanic.	White and blue quartz gravel.
Corotoman, Lower Lead.	Intervolcanic.	Mixed gravels.
Lucky Dog, N. & S.	Early intervulcanic.	
Lucky Dog, E. & W.	Late intervulcanic.	
Omega (Same as Lucky Dog E. & W.).	Late intervulcanic.	
Mountain House Drift Mine.	Prevolcanic.	White quartz.
Kanaka Drift Mine.	Thought to be extension of Mountain House Channel, prevolcanic.	White quartz.
Gale.	Prevolcanic secondary or early intervulcanic.	
True Grit.	Prevolcanic.	White and blue quartz gravel.
St. Clairs.	Prevolcanic, but probably secondary.	
Snowden Hill.	Intervolcanic channel of the fourth period.	
Pleasant View and Orient.	Early intervulcanic.	Appears to have flowed northerly, being the only one of its kind in this section. Average grade is about 40 ft. per mile.
Camptonville.	Prevolcanic.	White and blue quartz gravels.

All of these have been worked as hydraulic or drift mines or are now being developed.

In the ridge lying south of Forest City these channels must all exist, although many of them have not been discovered. The ridge lying south of Alleghany contains some of them. The west end of this ridge is low, however, and some of the higher channels have been completely eroded away. As the deep canyon of the Middle Yuba lies next south of the last-named ridge, the next appearance of any of these channels is in Nevada county, where in a short distance they are merged into the great deposits which fill the main ancient drainage basin, referred to above.

Of all of the northern tributary channels flowing from Sierra county, there appear to be three main tributaries or systems within themselves:

1. The Camptonville channel heads well up in Plumas county and courses southwesterly through the western side of Sierra county, joining the main trunk at North San Juan, in Nevada county. The towns, Whiskey Diggings, Howland Flat, Port Wine, Scales, Brandy City, and Camptonville are situated along this channel. As much of the deposit was but thinly capped with lava, many large hydraulic mines operated here.

2. The American Hill channel heads in eastern and central Sierra county. The upper reaches of the channel are still undiscovered, the principal developments having been made in the vicinity of American hill, where extensive drifting and hydraulicking have been carried on. While this is not an especially large channel, it is by far the largest ancient tributary in the eastern portion of the mineralized zone.

3. The Great Blue Lead of Sierra County. This large ancient channel lies midway between the other two, heads in the high mountains of Plumas county and courses southerly through the central portion, which is also the richest portion, of the Father lode or serpentine contact zone, previously described. This main channel with its numerous large tributaries has been wonderfully remunerative. Wherever found in Sierra county it has been worked with large profits, yielding as a rule from \$400 to \$500 per linear foot where breasted, and is generally considered worth from \$200 to \$225 per linear foot in profits.

This channel derived its name from the fact that its lower strata of gravel were of a bluish color, and at the time of its discovery it was supposed to be a portion of one main ancient river extending practically the full length of the State, paralleling the Sierras in a manner similar to that of the San Joaquin and Sacramento rivers. But upon the discovery of numerous other channels of similar character and off the line of this supposed single river, this "Great Blue Lead" theory gave way to the theory of an ancient drainage system complete within itself, on the order of the present-day drainage. This general fluviatic theory has proved to be the true explanation of the conditions.

The term "Blue Lead" is misleading: It refers to the color of the material usually found in the lowest portions of the channel trough. The gravel itself is composed of white and colored quartz pebbles and boulders with a filling of sand, clay, mud, sediments, etc. The iron salts in this material have not been altered beyond the ferrous forms, as the material, due to its low position, is usually under water. The gold seeks the lowest places, and hence is usually found associated with the bluish material. The same material on the high rims or benches is white, brown, or reddish, as the iron salts have there been oxidized to the ferric compounds. When the "blue gravel" is removed from the mine and exposed to the air and light it soon changes its color, or if the water has had a chance to escape from the low portions of the channel the gravel will not be blue.

The term is usually applied to prevolcanic channels, often containing little else but white quartz gravel; it could be more fittingly applied to describe intervulcanic channels, which are usually filled with dark gravels.

Why These Deposits are Suitable for Drift Mines.—To the north, near the head waters of the ancient streams, the channels are smaller, are full of big boulders, lie on very steep grades, are more deeply and completely buried with lava, and are harder to find and more expensive to mine, owing to the high altitude, severe climate, lack of roads, etc. To the south lie the broad ancient valleys, where the channels are from 1,000 ft. to a half mile in width; the gravel is generally small, light and sandy, and the deposits are from 100 to 500 ft. in depth. The gold is fine and scattered. No definite, absolute concentration has taken place. With the widening of the channels and a decrease in the grade, came a decrease in the velocity, so that the coarse gold was dropped further back in the hills, where the rims were more pronounced and the waters more confined.

Therefore these great gravel deposits form ideal hydraulic, steam-shovel and dredging fields.

Sierra county occupies an intermediate position between the two extremities. The channels are of medium size, varying in width from 150 to 500 ft. The gold is fairly coarse, yet enough fine gold is associated with the gravels to warrant the use of quicksilver in the sluices. The concentration is good, the pay having been restricted to the one main trough and to the gravels confined to the immediate beds of the channels. The grades run from 2 to 5 per cent., so that cars may be trammed directly up the beds of the channels. The size of the gravel is medium, and the boulders are usually within the "one-man" size. The gravel is from 2 to 10 ft. thick, as a rule. The breasts are usually carried about 6 ft. high, from 4 to 5 ft. being gravel, the remainder being bed rock. All of this gravel may carry some pay, and although most of the gold lies close to the surface of the bed rock, it usually pays to wash the entire body of breasted material. These conditions, when it is considered that the entire district through which these channels have flowed is highly productive of gold, mark Sierra county as the best drift-mining field known to gold miners.

Furthermore, the facilities for mining make the field inviting. The country is well watered, and timbered, the climate is most healthful, towns and roads are built and mail and telephone service is at hand.

A great many of the channels can be traced. The segments are of good length, averaging from 2,000 ft. to 3 miles, so that the discovery of only one segment means a fortune. As the development involves con-

siderable expense, companies are usually organized for working these drift mines. The deposits are usually developed by means of tunnels, so that the mines are self-draining and the material can be handled by gravity methods.

Effect of Modern Erosion on the Ancient Channels.—The ancient rivers flowed southerly. The modern rivers flow westerly and have cut away many miles of the older and higher channels and collected the gold. The deep canyons in which they flow are separated by high parallel ridges. The segments of the old channels appear near the tops of these intervening ridges still buried with the lavas, but having two ends exposed where the channels have been cut away by the erosion of the modern streams, as shown in Fig. 1. This peculiar condition has given rise to the expressions that a channel “enters,” “runs through,” “breaks out of” or “leaves” a ridge.

To an observer standing at the bottom of one of the deep canyons the country appears remarkably broken and rugged; but from the tops of the

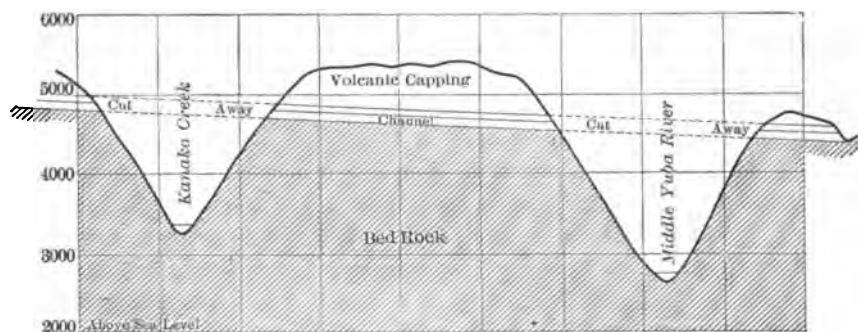


FIG. 1.—EFFECT OF MODERN EROSION ON ANCIENT DRAINAGE SYSTEM, SIERRA COUNTY, CAL.

ridges the general view is that of a vast plain, sloping gently to the westward, and into which deep east-and-west grooves have been cut.

Tracing Ancient Channels by Scientific Methods.—By applying engineering methods, many of the old channels can be traced through the country, across the canyons from one ridge to the next, by the identification of the segments found in the respective ridges.

The pioneers discovered as early as 1851 that there were gold-bearing deposits high on the mountain sides. The gold in the shallow placers was traced to these sources, but it was many years before the true cause of these deposits was understood.

To find the exact points where the channels “enter” or “leave” the ridges is often difficult, owing to the local landslides, deep loam, and dense vegetation. Wherever developed, these deposits have proved very rich. It appears that comparatively few of the ancient river segments

have been discovered and worked out; even in the mines opened up, many benches and tributary channels were overlooked, and in some instances, as in the Ruby mine, Forest City, the channel was lost after it had been followed and mined for half a mile.

As the district has been held back through lack of transportation facilities, it is in many respects a frontier, and the field offers wonderful inducements to energetic engineers and development companies who are strong enough to carry the dead expense necessarily accompanying the extensive preliminary work.

The Value of Ancient Channels

It is impossible to get complete data on the production of the drift mines of California, but those which I have been able to secure are considered reliable.

While the old channels in California have yielded from \$100 to \$250 profit per linear foot, where the yield for fairly long distances is averaged, like modern streams they have short stretches of poor pay, empty pot-holes, and wide bars of low-grade gravel; so that before judgment is passed upon the value of a deposit the work should be carried to a stage far beyond that of mere discovery.

The character of the bed rock often has a great deal to do with the richness of the gravel. Even in the Great Blue Lead, which averaged between \$400 and \$500 per linear foot for many miles, a place was found on serpentine bed rock where the channel was practically barren for more than 400 ft. In the Ruby mine, some unusually rich pay was found on the serpentine bed rock, so that the inference cannot be safely made that serpentine bed rock is always a sure sign of poor pay, although in the majority of cases it seems to be so. Any hard, homogeneous or slick bed rock is unfavorable; porphyries and granites are fair, while schists, slates, and especially the so-called "Calaveras formation," are exceptionally good gold catchers.

While it is the general consensus of opinion that Alaska and the Klondike are very rich gold fields, it is interesting to note that according to the data collected by A. H. Brooks, the average yield of the Home streams was about \$100 per linear foot, and that of the White Channel of the Klondike was only \$380 per linear foot; showing that the average yield of the California channels was from \$50 to \$150 per linear foot in excess of that of the northern streams.

In a general way it may be said that the ancient channels of Sierra county yield from \$150 to \$500 per linear foot. The Camptonville-Brandy City-Scales-Port Wine channel is said to have yielded about \$1,000,000 per mile. The White Bear channel, near Downieville, was breasted from 40 to 100 ft. wide and yielded approximately \$200,000

from 1,100 ft. The next 1,100 ft. of the same channel, worked by the New Monte Cristo Co., is said to have yielded about the same. For short stretches, this channel yielded as high as \$400 per foot, and for about two years the average value of the gravel extracted by the White Bear Co. was \$7 per car. The costs ran from 20 to 33 per cent. of the gross yield, exclusive of cost of plant and property. The mine had no road and all supplies were packed to the mine, on mules, and owing to the deep snow in winter nearly five months' supplies had to be laid in each fall.

The Bald Mountain channel yielded a total of \$3,100,000 from 7,500 ft. of channel. The gravel in the central portion of the pay streak averaged from \$3 to \$7 per car load, but as the gravel could be mined and treated for from \$1.25 to \$1.44 per car, much low-grade ground was worked, and all material extracted from the mine, whether waste or pay, was charged against the total number of loads, bringing the general average down to about \$2.17 per load.

The Ruby and the Bald Mountain Extension gravel averaged from \$2 to \$5 per car load, the costs of mining and washing being about \$1.25 per load. The Young America gravel mine, near Forest City, yielded \$7 gravel for a long working period. The North Fork mine gravel paid \$5 per load for hundreds of feet. This mine was not worked out and is now being reopened.

It is usually safe to figure that each breaster will extract, on the average, about five car loads per day. The more breasters, therefore, in proportion to the outside men, the greater will be the profits, since the wage item is about 78 per cent. of the total operating expense. In the larger channels, where the development work is the most systematic, gravel averaging \$1 per cubic yard, or about 70c. per ear, if free, will about pay for extraction. Cemented gravels require blasting and crushing or disintegration by use of mills or revolving trommels, so that the costs of mining and treating are from 50c. to \$1.50 more per ton than is true of the free gravels.

Throughout the Downieville-Forest City-Alleghany section gravel averaging from \$1.50 to \$2.50 is considered good pay; but where such values are found richer gravel is usually found in the center of the pay leads. The average yield per linear foot of the larger deposits runs from \$500 to \$1,000 and the average value per cubic yard ranges from 2c. to 40c. The gold is scattered through vast quantities of material and can only be recovered through some process where large quantities of gravel can be rapidly and cheaply treated.

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The Book Cliffs Coal Field, Utah

BY ROBERT S. LEWIS, SALT LAKE CITY, UTAH

(Salt Lake Meeting, August, 1914)

THE Utah coal field to which the name Book Cliffs is applied runs in a northeast direction from Mt. Hilgarde, in Sevier county, along the escarpment of the Wasatch Plateau to the vicinity of Castle Gate, thence southeast along the Book cliffs, which lie to the north of and generally within sight of the Denver & Rio Grande Railroad, to the Colorado line. The Book cliffs form the southern escarpment of the West and East Travaputs Plateaus, and vary in height from 2,000 to 6,000 ft. At the base of the Wasatch and Book Cliffs escarpments is a broad plain known as Castle valley. The coal beds outcropping in the escarpments along this valley are worked on the largest scale of any in the State, and this particular area is known locally as the Castle Valley coal field.

*Geology*¹

The Book cliffs are capped by the Tertiary—Eocene—formation, consisting of varicolored shales, buff sandstone, conglomerate, and subordinate thin beds of limestone. Different sections show diverse stratigraphy. This is mainly the Wasatch formation, and lies unconformably upon the rocks of the Upper Cretaceous. Fig. 1 is a generalized section of the rocks of the Book Cliffs field. The Mesaverde formation contains the cliff-making rocks of the Book cliffs, and is composed of massive sandstone interbedded with sandy shales and beds of coal and carbonaceous shale. The sandstone has greater resistance to erosion and the formation is characterized by escarpment ledges and cliffs. The shale is commonly sandy and is drab in color, but where it is associated with the coal it is usually carbonaceous. The sandstone is generally buff, though occasionally it is white, and in places red. The bedding ranges from thin to massive. In the lower part of the formation the sandstone is red in

¹ Abstracted from *Bulletin No. 371, U. S. Geological Survey*, by G. B. Richardson: Reconnaissance of the Book Cliffs Coal Field, between Grand River, Colorado, and Sunnyside, Utah.

Mancos shale is a dense black, blue, gray, and drab clay shale, which contains local lenses of limestone and, at the top, thin beds of buff sandstone, by which this formation grades into the Mesaverde. The shale is much broken by cracks and joints, which frequently contain saline films. It forms the base of the Book cliffs and underlies Castle valley.

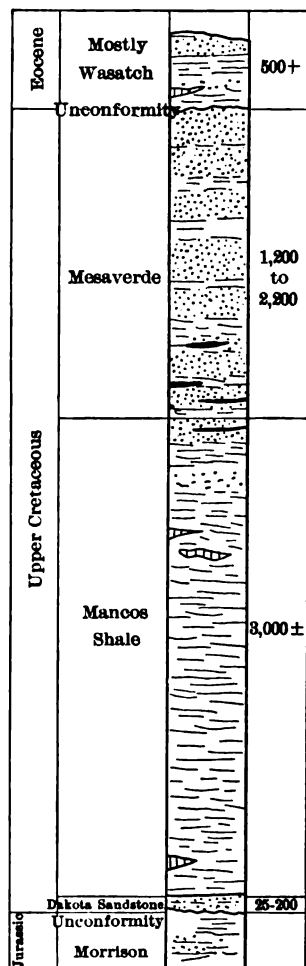


FIG. 1.—GENERALIZED SECTION OF ROCKS IN BOOK CLIFFS FIELD.

Coal

The coal beds are found in the lower 700 ft. of the Mesaverde formation, and at the top of the Mancos shale. The coal is a high-grade bituminous coal, that can be stored for a long time without deterioration.

year appeared to be in excellent condition. Cleating is often strongly pronounced and is taken advantage of in mining. Patches of resin occur in the seams, being much more plentiful in some mines than in others. The coal burns freely, leaving a moderate quantity of ash of a character closely resembling wood ashes. British thermal units range from 13,480 to 14,750. The seams worked vary from 5 to over 20 ft. in thickness, and are remarkably free from impurities. They dip at an angle of 6° or less, and the mines are usually dry—so much so that the companies have found it necessary to install elaborate sprinkling systems to wet the dust, which is of a highly inflammable nature. All the mines in the district are worked on the room-and-pillar system. Entries are driven on the double-entry plan. Mining machines have been in use for only a few years, but have come into such favor that they will soon be in universal use throughout the field. At nearly all the mines the seams outcrop high up on the steep hillsides, making it necessary to use inclines for transporting the coal to the tipples. A striking feature of the field is the great extent of the area of burned coal, which, in some cases, reaches to a distance of 2,000 ft. from the outcrop.

During 1913, Utah produced 3,289,265 tons of coal. Imported coal was consumed to the amount of 326,095 tons, most of this coal coming from Wyoming. The coal exported from the State amounted to 680,091 tons, which was distributed among the States of Washington, Oregon, California, Nevada, Montana, and Idaho. Most of the 314,694 tons of coke made at the Sunnyside plant was shipped to the smelter at Anaconda.

Operators

The largest operations in this field are those controlled by the Denver & Rio Grande Railroad and the United States Smelting Co. The former controls the mines at Sunnyside and Castle Gate, as well as the Utah, Winter Quarters, Clear Creek, and Schofield mines in the Pleasant Valley coal field. The latter controls the mines at Mohrland, Black Hawk, Hiawatha, and the Panther mine near Castle Gate. The mines at Storrs, Standardville, and Kenilworth belong to other companies. All of these mines have railroad shipping facilities. The road from Price to Black Hawk, the Southern Utah & Castle Valley Railroad, has a 5 per cent. grade in places, which is too steep for good railroading and the economic handling of the large tonnage expected from this district. For this reason the United States Smelting Co. is building a new road, the Utah Railway, from Black Hawk, to tap the main line of the D. & R. G. R. R. at a point about a mile south of Castle Gate. This line will be 24 miles long, and will have a maximum grade of 2 per cent. Construction is well along, and the road will probably be in operation some time this fall. The

accompanying sketch map, Fig. 2, shows the important mines in both the Book Cliffs and Pleasant Valley fields.

Important Coal-Producing Properties of Utah. For Year 1913*

Name of Mine	Tons	Men Em- ployed	Operator	County	Post Office
Winter Quarters...	400,214	368	Utah Fuel Co.....	Carbon...	Winter Quarters.
Clear Creek.....	272,363	225	Utah Fuel Co.....	Carbon...	Clear Creek.
Utah Mine.....	86,528	117	Utah Fuel Co.....	Carbon...	Utah Mine.
Castle Gate.....	319,692	502	Utah Fuel Co.....	Carbon...	Castle Gate.
Sunnyside.....	669,907 ^b	955	Utah Fuel Co.....	Carbon...	Sunnyside.
Kenilworth.....	402,952	405	Independent C. & C. Co...	Carbon...	Kenilworth.
Aberdeen.....	470	2	C. A. Ballinger.....	Carbon...	Price.
Carbon.....	9,019	290	Panther Coal Co.....	Carbon...	Castle Gate.
Storrs.....	116,977	268	Spring Canyon Coal Co....	Carbon...	Storrs.
Hiawatha.....	395,185	461	Consolidated Fuel Co.....	Carbon...	Hiawatha.
Black Hawk.....	176,349	308	Black Hawk Coal Co.....	Carbon...	Black Hawk.
Ketchum.....	500	15	Ketchum Coal Co.....	Carbon...	Castle Gate.
Standard.....	10,000	56	Standard Coal Co.....	Carbon...	Helper.
Nealen.....	50,266	91	American Fuel Co.....	Grand...	Nealen.
Mohrland.....	262,359	357	Castle Valley Coal Co....	Emery...	Mohrland.
Johnson.....	620	2	A. M. Johnson.....	Emery...	Orangeville.
Black Diamond...	262	2	Black Diamond Coal Co...	Emery...	Orangeville.
Rock Canyon.....	417	2	Akel Anderson.....	Emery...	Clawson.
Queatchappel.....	794	2	San Rafael Fuel Co.....	Emery...	Castle Dale.
New York.....	960	3	Utah Fuel Co.....	San Pete..	Fairview.
Larson.....	1,200	4	Mt. Pleasant Coal Co.....	San Pete..	Mt. Pleasant.
Morrison.....	2,100	6		San Pete..	Sterling.
Wasatch.....	16,701	35	Weber Coal Co.....	Summit...	Coalville.
Grass Creek.....	56,163	80	Union Fuel Co.....	Summit...	Grass Creek.
Rees.....	22,879	29	Rees-Grass Creek C. C....	Summit...	Coalville.
Briquette.....	2,800	7	Superior Fuel & B. Co....	Summit...	Coalville.
North Star.....	750	3	Geo. Gray.....	Uintah...	Vernal.
Hatch.....	1,100	4	P. E. Ritch.....	Uintah...	Vernal.
Mill.....	3,100	10	Pack & Allen.....	Uintah...	Vernal.
Kidd.....	397	3	A. C. Kidd.....	Uintah...	Vernal.
	3,283,024				

* Compiled from the 1913 report of J. E. Pettit, State Coal Mine Inspector.

^b Nearly all of this coal was utilized in making 314,694 tons of coke.

Castle Gate Coal Co.

Two mines of the Utah Fuel Co. are at Castle Gate. The No. 1 mine entrance is about 600 ft. southwest from the tippie, while the No. 2, or Willow Creek, mine is about a mile to the northeast, being reached by a motor haulage road 5,300 ft. long. This road has a gauge of 3 ft. 4 in., and is laid with 60-lb. rails. Trolley supports, made of 1½-in. pipe, set in concrete, are spaced 25 ft. apart. The road, average grade 2 per cent., passes through two tunnels, one of which is 1,000 ft. long and the other 400 ft.

Tippie.—This is of steel construction, and was built by the Ottumwa Box Car Loader Co. It has a capacity of 2,000 tons per day, shaking

in.; nut coal, from $1\frac{1}{2}$ to either 4 or 6 in.; lump coal, all over 4 to 6 in. For some years the slack coal, that passing a $1\frac{1}{2}$ -in. screen, was sold for heating and power purposes until some one suggested taking out the dust and making two products. This movement, once started, had to be followed by all the companies, and the result is that the dust has practically no market and is usually sent to the dump.

As the two mines are on opposite sides of the canyon through which the railroad runs, the tippie is provided with two Phillips cross-over dumps, one for each mine. Ottumwa box-car loaders are employed for loading the nut and lump coal in box cars; the open cars are loaded directly from chutes.

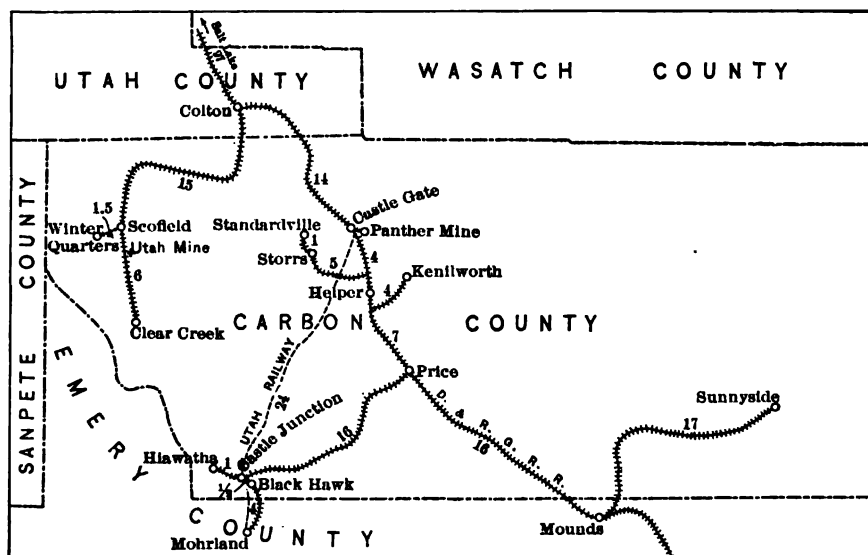


FIG. 2.—MAP SHOWING LOCATION OF IMPORTANT MINES IN THE BOOK CLIFFS AND PLEASANT VALLEY COAL FIELDS, UTAH.

Power House.—Dust is used for fuel under the eleven 125-h.p. fire-tube boilers. A drag conveyor brings the dust from the tippie to a 100-ton steel bin just outside the power house. From the bin, the coal is conveyed to the bunkers above the boilers, which are fed by American automatic stokers. The generators installed furnished direct current at 500 volts. These are now being replaced by units consisting of a 500-kw. General Electric alternating-current generator supplying current at 2,300 to 4,000 volts and running at 3,600 rev. per minute, directly connected to a Curtis turbine taking steam at 100 lb. and running condensing. One unit is now in operation, and the foundation for a second

at 500 volts for the locomotives and mining machines.

The A or lowest seam, 4 to 10 ft. thick, was the first one opened, and this is now worked out, save for the stumps of a few pillars. The B seam, separated from A by a 30-ft. parting, is not worked in this mine; the seam is too thin (3 ft.) and the coal is not very clean. Most of the present output comes from the C seam, 4 to 7 ft. thick, 100 ft. above. A 4-ton electric locomotive is used in this seam to take the coal to the head of the incline, from where it is dropped to the main haulage slope running along the A seam. The D seam, 7 to 11 ft. thick and underlain by 15 to 20 ft. of sandstone and shale, was discovered only a short time ago and the work done there is in the nature of development work. At present this seam is reached by an incline driven up from the C seam. This necessitates a rather roundabout method of getting the coal to the tippie, so the company is considering the driving of a slope from the D seam to the surface and dropping the coal to the tippie by a new incline.

The methods of mining differ in minor details in the two mines, but the following description of the working of the C seam of the No. 1 mine will give a good idea of the method employed. Raise entries are run directly up the dip, on a grade of 10 per cent. The first rooms are run parallel to the raise entries, leaving a barrier pillar about 70 ft. thick between. The side entries are turned off at right angles, giving a practically level grade. The rooms are turned off at an average angle of about 55°, which gives a grade of about 5 per cent. The rooms for hand mining are 20 ft. wide, while those for machines are driven 30 ft. wide. The pillars between the rooms are 40 ft. wide. Barrier pillars 75 ft. thick are left between the ends of the rooms and the air course of the next entry. Though all workings are surveyed, the miners are often instructed to start the last break-throughs between the rooms, by driving at right angles to the rooms at a point 40 ft. back from the end. The State law requires connections between the rooms every 125 and 159 ft. and every 100 to 125 ft. between entries. A large part of the mining in this mine is done by hand, but recently five Sullivan short-wall machines have been introduced. Little or no timber is required in the rooms, but the roof must be carefully supported when the pillars are being drawn. A thorough system of electric shot firing has been installed. Double switches, to guard against leakage, are used wherever the firing wires cross the trolley line. Inspectors examine all places where shots have been fired in order to locate any missed shots. Mules are used for haulage in the rooms.

The No. 2 mine has two workable seams; the upper one corresponds to the D seam in No. 1 mine, and is from 18 to 25 ft. in thickness; the other corresponds to the B seam and is from 4 to 7 ft. in thickness. All mining

short-wall type being used.

A General Electric locomotive, originally of 20 tons weight but reduced to 15 tons, takes the trips to the tipple. Mules are used underground for gathering and haulage in the side entries.

The output of No. 1 mine is about 850 tons per day, and of No. 2 mine from 600 to 700 tons per day. Wooden cars with end dump are used. These have a 28-in. wheel base, 18-in. wheels, and are made of rough 2-in. plank. Two sizes are employed; one holding 2,500 lb. and the other holding 5,800 lb.; the average monthly capacity of all cars is about 2,300 lb. The smaller cars are used in the narrow seams where head room is limited.

The Spring Canyon Coal Co.

The mine office of this company is at Storrs, Carbon county, Utah, which is reached by a branch railroad $4\frac{1}{2}$ miles long, connecting with the main line of the Rio Grande Western at a junction point about a mile north of Helper.

To one familiar with the hastily constructed buildings found in so many Western mining camps the town of Storrs is a pleasant surprise. Laid out along the bottom of the canyon, the office, hotel, and miners' cottages are supplied with bath, toilet, and electric lights, and are built of light-colored sandstone taken from the nearby quarry of the company.

The tipple and power house, shown in Fig. 3, are at the lower end of the town. The tipple is of steel construction and is equipped with shaking screens. The coal passes from the receiving hopper to the screens over a steel apron picking conveyor 5 ft. wide. Men on each side of the conveyor pick out the waste or "bone" as the coal passes by. When the mine is working at full capacity there is a depth of from 15 to 20 in. of coal on the conveyor, making it hard to thoroughly clean the coal. To overcome the difficulty, the management has planned to move the conveyor to the lower end of the screens, where only the nut and lump coal will pass over it. The width will be increased to 7 ft. and a central longitudinal division will be introduced to keep the two sizes of coal separate. Although the coal is no dirtier than other coals in this field, this company is the first to install a picking conveyor. The screens are arranged to give the following products: Slack, all the coal passing a $1\frac{1}{2}$ -in. opening; nut coal, between $1\frac{1}{2}$ and either $4\frac{1}{2}$ or 6 in.; lump coal, all over $4\frac{1}{2}$ or 6 in. Some contracts specify $4\frac{1}{2}$ in. as a maximum size for nut coal, while others call for coal as large as 6 in.; so two sets of screens are required. The slack is elevated to the top of the tipple, where it passes through a revolving screen with $\frac{3}{8}$ -in. openings. The oversize, or "screened slack," is sold, but there is so little demand for the undersize, or "dust," that it usually goes to the dump. The run-of-mine coal gives

about 45 per cent. lump, 30 per cent. nut, and 25 per cent. slack; the dust in the slack being from 5 to 10 per cent. of the total coal mined. The tracks under the tippie are laid on a 2 per cent. grade so that the cars will move under the action of gravity when the brakes are released. Two tracks are for lump coal and one each for the nut coal, screened slack, and dust. Gondola cars are loaded directly from chutes, but the box cars are loaded with lump or nut coal by means of Ottumwa box-car loaders.

A narrow slot is cut in the steel chute under the 1½-in. screen. This permits of sufficient slack passing through to supply the power house with fuel. A spiral conveyor carries the slack to the boiler room, where it is fed by hand to five 150-h.p. Springfield boilers. These boilers are of the marine type and give excellent satisfaction. Two 200-kw. General



FIG. 3.—POWER HOUSE AND TIPPLE OF THE SPRING CANYON COAL CO. SHOWS CHARACTERISTIC TOPOGRAPHY OF THE BOOK CLIFFS COAL FIELD.

Electric alternating-current generators, running at 200 rev. per minute, furnish current at 2,300 volts. They are directly connected to Buckeye engines. For supplying lights and extra power, when needed, there is a 40-kw. General Electric generator running at 300 rev. per minute and delivering current at 2,300 volts. This is directly connected to a B. F. Sturtevant vertical engine. The power is transmitted to the mine, where a motor-generator set transforms it into direct current at 250 volts.

The workable coal on the property is found in three seams. The lowest, or No. 1, seam is 8 ft. thick, and the more extensive workings are in this seam. Above this seam, and separated from it by a 20-ft. shale parting, is the No. 2 seam, 4½ ft. thick. This seam is not being worked at present. The No. 3 seam, which varies in thickness from 6 to 12 ft., is 180 ft. above the No. 2 seam. The outcrop of the coal is high up on the

steep mountain side, the seams dipping back into the mountain at an angle of from 7° to 10° from the horizontal; direction of dip is about N. 18° E. The seams are exceptionally free from slate. Near the mouth of the main entry the No. 1 seam pinches down to an unworkable thinness, but, fortunately, this irregularity is of small extent. The main haulage way runs NW.-SE. to reduce the grade to a maximum of 3 per cent. The locomotive pulls the loaded trip up to the head of the slope. From there it pushes the cars out to the loading station of the aerial tramway, which connects the mine with the tippie. The loading station is close to the entrance to the mine. The necessity of pulling the loads up grade introduces an element of danger, for a loaded car or the entire trip might break away and, running down the slope, cause a serious accident. The company has installed a derailing switch, which is operated by a rope running along the side of the haulage way. Should the cars break away, the trip rider can pull the rope opening the switch and thus derail the cars. Two trips have been stopped in this way, and what promised to be serious accidents were prevented. The No. 3 seam is reached by a haulage road 2,500 ft. long, which runs along the side of the mountain. This seam has only recently been opened. At present it is producing 400 tons a day, while the No. 1 seam is sending out 700 tons per day, making a total of 1,100 tons as the daily output of the property.

The side entries are run nearly east-and-west to give a grade of from $2\frac{1}{2}$ to 3 per cent. Mining is done on the room-and-pillar system. The rooms are 300 ft. long by 30 ft. wide, with 34-ft. pillars between. A barrier pillar 50 ft. wide is left between the ends of the rooms and the air course of the next entry. The first rooms were laid off directly on the pitch of the coal, giving a grade of from 7 to 10 per cent. This was done to take advantage of breaking the coal on the face cleats, but the grade was found to be too steep for safe and easy handling of the cars. The rooms are now laid off at an angle with the entry, so that the grade of the track along the rooms is about 3 per cent. Horses take the cars to the side entries. Gathering locomotives of the reel type then pull the cars to the main haulage way. The entries are 15 ft. wide and 8 ft. high. The coal is undercut by machines, and is broken down with No. 3 Guardian powder, made by the Independent Powder Co.

Two General Electric locomotives, of the outside-frame type and weighing 15 tons, are used on the main haulage way, which has 50-lb. rails laid on heavy sawed ties. Three gathering locomotives are used. They weigh 6 tons each. All the locomotives have a 5-ft. wheel base, and operate on 250 volts. Large horses are used in the rooms, because of the ample head room afforded and the great weight of the cars.

The following mining machines are used, all of the short-wall type: One Jeffrey; four Sullivan, and seven Goodman. Great care is exercised to keep the faces of the rooms as square as possible. In making the sump-

ing cut the machines are pointed outward to cut out into the rib, so that the full width of the room is easily broken. The rib is left slightly irregular, but is squared up with little trouble. The machines make a 6-ft. cut. The blasting holes are put in to a depth of $5\frac{1}{2}$ ft., the coal breaking readily to the full depth of the cut.

Sturtevant fans, 5 ft. by 2 ft. 6 in., are used for ventilation, one for each seam. Owing to the small extent of the working, no gauge pressure can be noticed. The overcasts are made with walls 10 in. thick, the bottom being reinforced with old rails. A mixture of 1 part cement with 5 parts of "bug dust," or machine cuttings, makes an excellent material for the walls. It is also used for the stoppings.

The tramway, connecting the mine and tippie, was built by the American Steel & Wire Co. It has a drop of approximately 500 ft. in a distance of 3,000 ft. The loading station is on a precipitous slope of the mountain. This location is necessary in order to avoid an angle station on the tramway. A steel truck holding two of the tramway buckets is used in place of a regular mine car. The weight of the assembled empty car is 3,300 lb. and each bucket holds on an average 2,300 lb. of coal, making the weight of a loaded car 7,900 lb., of which 41.8 per cent. is dead weight. This unduly large per cent. of dead weight makes the cost of haulage so high that the company is now designing a lighter car. The loaded cars, as they come from the mine, pass over a Strate coal scale and then on to a tipping platform, where the tramway carriers, or hangers, are attached to the lugs on the buckets. The steel truck is then lowered to free the buckets and to permit of their being started on their way. Two empty buckets are now run over the truck, which is raised, and the hangers are then disengaged. The assembled empty car is kicked on to the return track by the oncoming load. A 6-h.p. motor operates the tipping platform. The tramway is self operating, but a 30-h.p. motor is installed at the head end to bring up supplies in case no coal is being sent down. At the tippie, the buckets are dumped by hand. The carriers are automatically detached from the cable and run into the tippie on a track. On making the half-circle turn to the return track they pass over a small bin, or hopper. The dumping has to be done quickly, because of the limited dumping space, and the buckets must not come too close together. The estimated capacity of the tramway is 250 tons per hour, but this is not attained at present. It is planned to extend the dumping space so that the full capacity can be had.

The dust in this mine is highly inflammable. To minimize the danger of explosion an extensive sprinkling system has been installed, by means of which all haulage ways and working places are kept damp. The workings are free from gas and the miners use acetylene lamps. The mine is equipped with the Stromberg-Carlson telephone, and the haulage ways are lighted by electricity. Near the entrance to the mine is a repair

shop, which is provided with emery wheel, forges, drill press, lathe, and shaper.

Analyses of Coals

Seam	Volatile Matter Per Cent.	Ash Per Cent.	Moisture Per Cent.	Fixed Carbon Per Cent.	Sulphur Per Cent.	B.t.u.
No. 1	46.7	4.5	1.9	46.9	1.07	14,547
No. 2	47.4	4.6	0.7	47.2	0.98	14,730
No. 3	43.4	3.6	1.8	51.2	0.89	14,725

The mine has been producing but slightly more than a year. The company controls 2,000 acres, of which 1,200 acres are coal bearing. The elevation of the mine is 6,780 ft.

Standard Coal Co.

This mine is at Standardville, about a mile northwest of Storrs. The Standard is one of the newest companies in the field, and has not yet



FIG. 4.—REINFORCED CONCRETE RETAINING WALL AND SHAKING-SCREEN FOUNDATION AT THE MINE OF THE STANDARD COAL CO.

reached the producing stage. The work being carried on is development work, though daily shipments of from 350 to 400 tons are being made.

The mine and tipple are connected by an incline 4,200 ft. long, the first half of which is on a 7 per cent. grade, while the second half has a 12.5 per cent. grade. At the entrance to the mine the coal seam is 14 ft. thick, but this represents a rather unusual condition, as the average thickness is from 10 to 12 ft. A thin parting occurs about 5 ft. from the floor.

The reinforced concrete tipple, Fig. 4, presents some interesting features of design. In order to dispense with the approach trestle, and to give

a substantial bracing to the tippie, a reinforced concrete retaining wall, Fig. 5, was built. The wall is 18 in. thick at the bottom, 10 in. thick at the top, and is 46 ft. high at the highest point. Buttresses are placed every 13 ft. The mine cars pass continuously in one direction over a track laid close to the edge of the wall. The filling required was 12,000 cu. yd.,

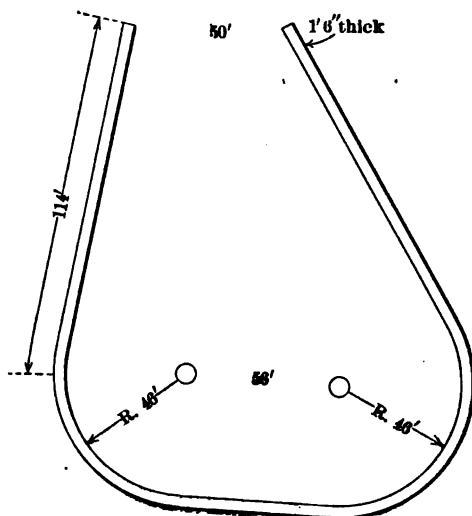


FIG. 5.—PLAN OF REINFORCED CONCRETE RETAINING WALL SHOWN IN FIG. 4.

much of it coming from the cuts made during the construction of the incline.

The shaking screens rest on roller bearings, which are secured to heavy I-beams imbedded in the inclined concrete stringers. Pillars, placed be-

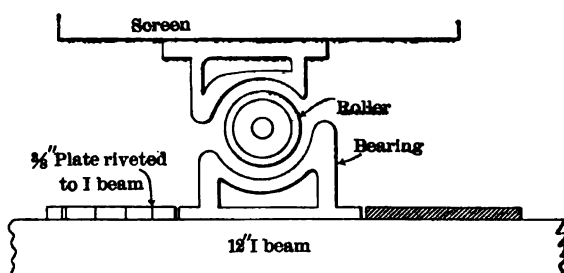


FIG. 6.—BEARING FOR SHAKING SCREENS.

tween the railroad tracks, support the stringers and screens. To secure the undulating motion of the suspended type of shaking screen, special bearings were designed. Their arrangement is shown in Fig. 6. The brace plates, placed against the sides of the bearings, are firmly riveted to the I-beams, which have their ends strongly anchored in the retaining

wall in order to make the structure as rigid as possible. It is said that the vibration, when the screens are running, can scarcely be noticed.

The slack is elevated to a revolving screen, which discharges the dust and screened slack into steel bins placed beneath the screen. A rotary dump, made by the Link Belt Co., is used for the mine cars. The railroad box cars are filled by means of an Ottumwa rocking cradle loader, the first one of this type to be used in the State.

Independent Coal & Coke Co.

This company's mine is at Kenilworth, the terminus of the Kenilworth & Helper Railroad, a branch line 4 miles long, which leaves the main line of the D. & R. G. R. R. at Kenilworth Junction, a point about a mile south of Helper. The road has a maximum grade of 6.4 per cent., requiring the use of Shay locomotives, three of which are now in operation. Two of these are of 90 tons weight and the other weighs 70 tons.

Kenilworth is a well-built town, laid out in a systematic manner on a gently sloping flat at the foot of the mountains. There are 159 buildings in all, 135 of which are for the miners. Most of these latter are four-room houses. About 400 men are employed at present.

Power House.—This is a substantial stone building containing six boilers, feed-water heaters, pumps, and generator. The boilers are 125-h.p. Kewanee horizontal tubular boilers, 72 in. by 16 ft., and are equipped with American automatic stokers. Dust is used for fuel. The 500-kw. alternating-current generator is of General Electric make, operating at 2,300 volts, and is directly connected to a Curtis turbine running at 3,600 rev. per minute and taking steam at 125 lb. The exhaust steam goes to a No. 2 Worthington counter-current jet condenser and then to the cooling apparatus, consisting of 24 Schute Koerting spray nozzles and a concrete reservoir of 100,000 gal. capacity.

Tipple.—The old wooden tipple is to be superseded by an all-steel tipple of 3,000 tons daily capacity, designed by the Link Belt Co. This tipple is of the shaking-screen type and will make five sizes of coal, in addition to the straight mine run. Two 60-ton storage bins are provided for the screened slack and dust. A drag conveyor will take the dust to the power-house storage bins for use under the boilers. Ottumwa box-car loaders will be used. One, of the standard pusher type, is equipped with horizontal steam engines; the other is an Ottumwa-Ecks loader of the belt conveyor type, illustrated in Fig. 7. This is equipped with an electric motor and is so arranged on a turntable that it can be used on either of two tracks. The dump is of the Link Belt revolving type and handles one mine car at a time. One feature of the tipple is the use of an incline car lift, shown in Fig. 8. The empty cars, upon passing the cross-over switch and starting back, run on to a short stretch of steeply inclined track, up

which they are pushed by an endless chain, running in the middle of the track, which has attached lugs that catch against the car axles. By the



FIG. 7.—OTTUMWA-ECKS ELECTRIC BELT-CONVEYOR BOX-CAR LOADER AT THE MINE OF THE INDEPENDENT COAL & COKE CO.

use of this lift the head car of the empty trip is brought to the same level as the rear car of the loaded trip, and the trip rider can change the haulage

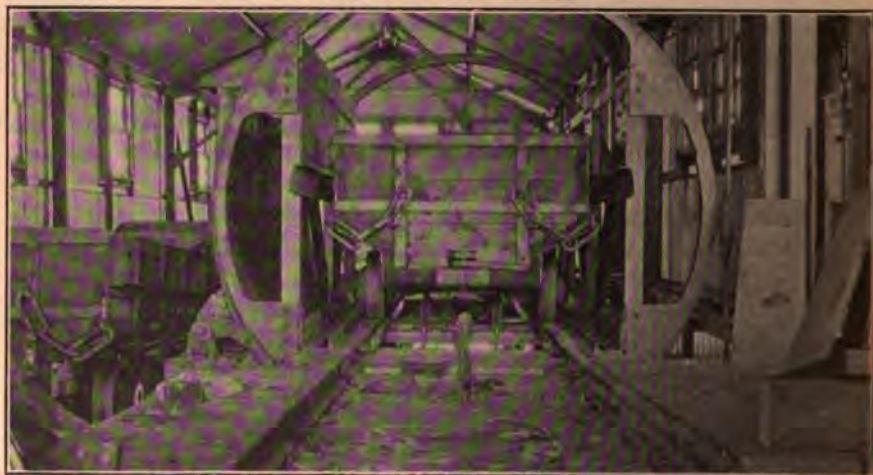


FIG. 8.—LINK BELT ROTARY-DUMP TIPPLE.
Car lift shown in lower left-hand corner.

cable from the loaded to the empty trip without being compelled to carry it from one level to another.

Mine.—Two inclines are used to drop the coal down from the mine to the tippie. The first, or Kenilworth, incline, beginning at the tippie, has a length of 200 ft. on a 2 per cent. grade, then a stretch of 620 ft. on a 27.7 per cent. grade, the last part being 1,290 ft. long with a grade of 39 per cent. This incline delivers the empty cars to the second incline, called the "shelf road." This incline is 1,200 ft. long, with an average grade of $12\frac{1}{2}$ per cent. It runs around the edge of the mountain to the mouth of the mine, following the dip of the sandstone strata. A 25-h.p. motor is used on the main incline for starting the trips, while the shelf road is provided with a 200-h.p. motor. Eight cars make up a trip. The cars are of wood and weigh 3,500 lb. each. Their average load is 7,000 lb. of coal, the cars being piled high with lumps. There are three workable seams on the property. The lowest seam averages 18 ft. in thickness, and is called the Aberdeen seam. The Royal Blue seam, 7 ft. thick, is 45 ft. above this. The Kenilworth seam is 66 ft. above this and has an average thickness of 18 ft. The present daily output of 1,500 tons comes almost entirely from the Aberdeen seam. This seam dips N. 10° W. on a 10 per cent. slope. The main haulage way runs N. 32° 58' W. along the seam on a 9 per cent. slope for 2,200 ft. and then turns due north to keep within the property lines. This slope will soon be double tracked throughout. It is provided with a double-drum 400-h.p. electric hoist, operating at a speed of 1,000 ft. per minute. Rooms are 350 ft. long and 24 ft. wide, with 50-ft. pillars between. Entries are driven 7 ft. by 12 ft., each pair being driven 65 ft. center to center. Horses are used for gathering, no locomotives being employed at present, though their use in the future is being considered. The coal is mined both by hand and by machines, six of which are in operation, one Jeffrey and five Sullivan, all of the short-wall type. Monobel powder is employed for breaking the coal, and all shots are fired by electricity. An elaborate sprinkling system has been installed. No timber is required, the mine roof being exceptionally solid. No work has been done toward the opening of the Royal Blue seam, and development on the Kenilworth seam has been in progress for only a few months. To reach these seams a connecting slope has been driven from a point at the bend in the main haulage slope of the Aberdeen seam. This connection is 540 ft. long and has a grade of 15 per cent. All the coal from these upper seams will be dropped to the bottom seam for hauling to the surface. The double tracking of the main haulage way will provide for a large increase in tonnage. A motor-generator set transforms the alternating current at 2,200 volts into direct current at 250 volts for mine use. The steepness of the main incline has been the cause of occasional accidents. The mine cars have broken away from the cable, but they hardly ever reach the tippie below, being stopped on a fill of soft earth about 200 ft. from the tippie. A large mass of sandstone broke away from the cliff above the incline and struck squarely on the rails of one track. It slid

down the track and smashed over the end of the old wooden tippie, doing considerable damage. To obviate any such damage to the new tippie, it is placed to one side of the old tippie and the tracks are curved for a short distance at the bottom of the incline. It is thought that the velocity of any runaway object will be so great that it will not make the turn, but will keep straight on past the tippie.

Analyses of Coals

	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Moisture, Per Cent.	B.t.u.
Aberdeen.....	43.97	51.62	2.96	1.45	13,480
Royal Blue.....	39.65	56.89	1.21	2.25	14,026
Kenilworth.....	42.83	52.52	2.28	2.37	13,536

Castle Valley Coal Co.

This company has its mine at Mohrland, in the northern part of Emery county, the southernmost point reached by the Southern Utah & Castle Valley Railroad.

Tippie.—This is a substantial framed structure, Fig. 9, covered with corrugated iron, and is built in the shape of the letter L; the long arm



FIG. 9.—TIPPLE AT THE MINE OF THE CASTLE VALLEY COAL CO.

contains the Phillips cross-over dump at the lower end; the upper part is long enough to hold a trip of 17 cars and serves to afford protection to the men in severe weather. The short arm contains the shaking screens. Wooden bins are used for the dust and screened slack. By their use, the tippie is not forced to shut down, should there be a shortage of cars, or difficulty in getting the loaded ones out of the way. The mine cars are

of steel and were made by the Watt Car Co. They weigh 3,300 lb. and hold from 3 to 4 tons of coal, depending upon the fullness of loading. They are equipped with link and pin coupling, but additional side chains are used for the sake of safety. Fairbanks quick-reading scales take the weights of the cars just before they run on to the dump. Christie box-car loaders, fitted with 50-h.p. motors, are used for loading the nut and lump coal. These work at a high rate of speed, throwing the coal against the ends of the cars with considerable force, which would lead to the making of a large amount of fines if the coal were very soft or brittle. The tippie has been in operation but a little over a year.

Mine.—The mine and tippie are connected by an incline 6,600 ft. long, having a maximum grade of 10 per cent. The tracks pass through two tunnels, one of which is 400 ft. long and is on a 4° curve. Sixty-pound rails are used, permitting of high speed of operation. The hoist, at the head of the incline, was built by the Denver Engineering Works. It was designed for lowering trips with a maximum rope pull of 15,000 lb. at a rope speed of 1,500 ft. per minute. This rope pull is taken care of by four band brakes, each 15 in. wide and 6 ft. in diameter. The brake rings are equipped with a large number of cooling vanes in order to radiate the heat generated. The rope sheaves are 6 ft. in diameter and are grooved for three laps of 1½-in. steel rope. Owing to unfavorable grades, the hoist is not self-starting, but is geared to a 170-h.p. motor. This is thrown out of gear as soon as the trip is well under way. The length of the hoist is 20 ft., width 10 ft., and height 7 ft. The shipping weight, exclusive of motor, was 55,000 lb. The hoists used at the Black Hawk, Hiawatha, and Standard mines are of similar design. The coal seam on this property varies from 12 to 19 ft. in thickness, and dips S. 10° 30' E. at a slope of 2.85 per cent. The main haulage way runs N. 72° 02' W. and has a slope of 0.3 per cent. in favor of the loads. Side entries are run N. 18° 58' E. and 18° 58' W. on grades of +1.95 per cent. and -2.45 per cent. respectively. Rooms are 400 ft. long by 20 ft. wide, with 40-ft. pillars between. Entries are driven 8 by 10 ft., with 50-ft. pillars between. Where the seam is thickest, the first cut is mined 6 ft. in height; a second cut of from 4 to 6 ft. is then taken out, and the remainder of the seam, left to insure a good roof, is taken down when the pillars are pulled. It is the intention of the management not to pull any pillars until all the rooms have been worked out. The pulling of the pillars will then begin at the boundary of the property.

Mining is done wholly by machines. At the present time five Goodman chain breast machines and four Jeffrey short-wall machines are used. Hauling is done by Goodman locomotives, of weights varying from 5 to 10 tons. These are equipped with reels for gathering purposes. Because of the highly inflammable nature of the dust, the mine is equipped with a well laid out sprinkling system. The main entries have 3-in. pipes, with

ring system has been installed. The powder used is 10 per cent. Hercules dynamite. Ventilation is good, but the company has recently ordered a Jeffrey fan of 300,000 cu. ft. per minute capacity against 3 in. of water. Power is generated at 250 volts, direct current, and is transmitted as such to the mine.

The coal is quite free from impurities and breaks readily and in large lumps, the output running nearly 70 per cent. lump. The daily production is from 1,600 to 1,700 tons. The elevation of the mine is 7,800 ft.

Black Hawk Coal Co.

The camp of Black Hawk is about 4 miles north of Mohrland. The main yards of the new Utah Railroad will be located here and an elaborate system of tracks is being laid out.

Tipple.—This is a frame structure built a few years ago. An incline laid with 45-lb. rails and 7,400 ft. long, with a drop of 1,000 ft., connects the mine and tipple, yet the coal is discharged near the bottom of the tipple on to an apron conveyor which elevates it to the screens. This conveyor is a source of much trouble; so much so that the present company, which inherited the tipple, is drawing plans for a new steel tipple. An interesting feature of the present tipple is the use of lip screens with openings 3 in. wide at one end, 5 in. wide at the other, and 11 in. long. This type of screen is much used in the Wyoming coal fields. Its particular advantage lies in the fact that the pieces of coal are rolled over in passing the screen, thus giving a more thorough screening and a better opportunity for the nut coal to pass through the screen. The sizes made at the tipple are the same as those made by the companies mentioned above. The company supplies much coal to the railroads. This coal is run-of-mine grade, but with a maximum size of 8 in. for lumps, as the firemen's union specifies that its members should not be compelled to shovel coal of a larger size. Though designed for handling 700 tons per day, the present daily output is from 1,100 to 1,200 tons. Box cars are loaded by steam-driven Christie box-car loaders. A Phillips cross-over dump and Fairbanks quick-weighing scales are used.

The coal seam varies in thickness from 15 to 22 ft. Mine cars are the Watt all-steel car. Haulage way and entries are driven 10 by 10 ft., with 40-ft. pillars between. The main haulage way runs S. 42° 37' W. and has a grade of -1.98 per cent. Side entries run as follows: First west entry, N. 79° 37' W. on a -0.74 per cent. grade, then N. 47° 93' W. on a +0.16 per cent. grade; first east entry runs S. 79° 37' E. on a +1.29 per cent. grade, then N. 61° 57' E. on a +2.27 per cent. grade. Rooms are 400 ft. long by 25 ft. wide, with 35-ft. pillars between. The

easy curves and gathering locomotives will soon be used. Mules are used in the rooms, and 10-ton Goodman locomotives on the main haulage way. Mining is done by machines, eight Sullivan short-wall and eight Goodman breast machines being used. An electric firing system is installed. The switch at the mouth of each room is operated by the miners. Each entry has a switch at its head. This switch is in a locked box, which can be opened only by a shot firer. The main, or firing, switch is located outside the mine. All who enter the mine must hang up checks on a check board at the mouth of the mine, taking them up on their return. The shot firer is not allowed to close the main switch until every one has checked out. Should a check remain on the board, the man must be accounted for before the shots can be fired. First-aid sets are placed in the mine, usually near the heads of the side entries. Power is transmitted from the power house to the mine at 2,300 volts, where a motor-generator set transforms it into direct current at 250 volts. The elevation of the mine is 8,100 ft.

Consolidated Fuel Co.

The office of this company is at Hiawatha, about a mile northwest of Black Hawk. Two mines are operated, Hiawatha No. 1 and Hiawatha No. 2. They are on the same seam, but are on opposite sides of the canyon. The mines are reached by an incline 10,400 ft. long, 42-in. gauge, and with a maximum grade of 10 per cent. The seam is from 12 to 16 ft. thick. Rooms are 400 by 25 ft., with 45-ft. pillars between. Entries are driven 10 by 10 ft. The cabin pillars left at the ends of the rooms are 60 ft. wide, but are now being changed to 40 ft. The seam dips at about 3 per cent. and the entries are run on grades of from 1/2 to 2 per cent. The workings are carefully laid out from surveys, and this mine is operated in the most systematic manner of any in the district. Like the other mines of the district, it is provided with an electric firing system; 40 per cent. Hercules dynamite is used. Mining is done by machines; seven Sullivan and one Jeffrey short-wall machines and one Goodman breast machine are used. Hauling is done by four 10-ton and one 5-ton Goodman locomotives. The tippie has the usual arrangement of shaking screens. The dump is of the rotary type, being the first of this type used in the State. The daily output is from 1,500 to 1,600 tons.

The Castle Valley and the Black Hawk coal companies and the Consolidated Fuel Co. are controlled by the U. S. Smelting Co., which has plans for extensive improvements of the properties. The individual power plants are to be replaced by a central station located at Black

Hawk. Two 750-kw. generators, direct connected to steam turbines, are to be installed and sufficient space is to be left for two more. The tippie at Black Hawk is to be replaced by a steel one of 3,000 tons daily capacity. Its location is to be such that the tram line will be shortened to 6,000 ft. and will be in a straight line, instead of having a curve near the middle point as at present. The Hiawatha tippie is to be replaced by a steel one of 5,000 tons daily capacity. During the month of November, 1913, the payroll of these companies amounted to \$107,000, about 1,800 men being employed.

Analyses of Coals

	Moisture, Per Cent.	Fixed Carbon, Per Cent.	Volatile Matter, Per Cent.	Ash, Per Cent.
Castle Valley.....	2.16	47.56	44.60	5.64
Black Hawk.....	2.14	47.29	43.53	7.04
Hiawatha.....	3.29	51.22	40.98	4.51

Utah Fuel Co.

The Utah Fuel Co. has two mines at Sunnyside. The seams are cut by a canyon, so two mines have been developed. Two seams are worked; the upper seam is 6 ft. thick and the lower seam is from 10 to 12 ft. in thickness. The parting is from 18 to 40 ft. thick in the No. 1 mine and from 12 to 30 ft. thick in the No. 2 mine. The coal dips a little north of east at an average of 9 per cent.

The main slopes are driven directly down the dip, and 200-h.p. hoists are used at the head of the slope. These are capable of handling trips of 16 cars, each car weighing 5,800 lb., at a speed of 1,500 ft. per minute. Side entries are turned off at right angles and at intervals of 400 ft., making the rooms 350 ft. long except where a fault of 30 ft. displacement cuts through the property; the rooms driven to this fault are 600 ft. long. The rooms are turned off at 45° with the entries to reduce the grade. Horses and mules are used in the rooms and side entries, while electric locomotives haul the coal from the main slopes to the tippie. The two locomotives used at No. 1 mine weigh 8 to 13 tons, respectively, while the two at the No. 2 mine, where the length of haul is 2.65 miles, are of 13 and 15 tons weight. The Jeffrey fan at No. 2 mine is capable of supplying 300,000 cu. ft. of air per minute against 5 in. of water. Its motor takes alternating current at 4,000 volts directly from the transmission line, but a motor-generator set is used to supply direct current at 500 volts for the locomotives and mining machines. All mining is done by hand at present, but four Sullivan short-wall machines have been ordered.

This is a coking coal and practically the entire daily output of the mines, about 2,500 tons, is made into coke. The breaker, which also

contains shaking screens in case coal is to be shipped, is in two units, each of which consists of one set of toothed rolls, 36 in. diameter by 40 in. face, which break the coal to nut size (largest pieces 3 in.). From the rolls the coal goes to a No. 8 Williams Jumbo disintegrator. A Jeffrey continuous bucket elevator 36 by 12 by 12 in. elevates the coal to a revolving trommel, 5 ft. in diameter by 12 ft. long, with $\frac{3}{4}$ -in. openings. The undersize goes to the oven bins, while the oversize is returned to the disintegrator. Six-ton electric larry cars are used for charging the ovens. There are 726 ovens in operation, but the company can hardly meet the demand for coke, most of which goes to the smelter at Anaconda. The ovens are of the beehive type, 12 ft. in diameter, and take a charge of 6 tons. The time of coking is 48 hr. and the coke made is 55 per cent. of the charge. All the coke is hand pulled.

The power house has 14 fire-tube boilers, of 125 h.p., 12 of which are kept in continual operation. Both alternating and direct current generators are used, but the company is planning to put all power on an alternating-current basis, and a 500-kw. turbo generator, similar to the one at Castle Gate, has just been installed.

Analyses of Coals

	Moisture, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Sulphur, Per Cent.	B.t.u.
Castle Gate.....	2.88	39.57	54.41	3.14	0.493	13,698
Sunnyside.....	1.87	39.22	54.78	4.13	1.022	13,944

Acknowledgments

The photograph of the Standard Coal Co.'s tipple was taken by D. H. Blossom, chief engineer. I also wish to acknowledge my indebtedness to J. E. Pettit, State Coal Mine Inspector, Prof. R. B. Ketchum, of the University of Utah, and to the various officials of the coal companies through whose kind assistance I have been enabled to secure the data here presented.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Lead-Matte Converting at Tooele

BY OSCAR M. KUCHS, TOOEELE, UTAH

(Salt Lake Meeting, August, 1914)

In January and February of the current year, at the Tooele plant of the International Smelting Co., Tooele, Utah, a modification of the usual method of converting, for the treatment of copper-lead matte from lead blast furnaces, was developed and put into successful operation. In brief, the method consists in blowing molten matte direct from the blast furnaces in basic-lined converters, without the addition of siliceous fluxing ores, to the almost complete elimination of the lead contained, and to the production of copper bullion, the resulting lead fume being recovered by the filtration of the converter gases through woolen fume bags.

The process is carried out in conjunction with the converting of reverberatory copper matte, the heavy iron slag, together with the copper precipitated, being transferred to converters blowing copper matte. The purpose of this transfer is to reduce the copper oxide formed, due to the necessary overblowing of the lead-matte charge, and to utilize, as far as possible, the value of iron from the lead matte, for the fluxing of siliceous dry ores. The final slag from the copper converters is then skimmed off and transferred molten to a reverberatory smelting furnace treating copper ores.

This variation from general converting practice was the outgrowth of results obtained during 1913 when it was endeavored to convert copper-lead matte in the usual manner, with the direct addition of siliceous fluxing ores. To make clear the reasons lying back of the process it is necessary to recount at some length the results of operations during that year.

In the design of the lead plant it was planned to treat the copper-lead matte from the furnaces by direct converting without reconcentration and thereby to recover the copper, silver, and gold as quickly as possible. A converter plant of ample capacity, treating copper matte, already being in operation, it was only necessary to install a bag house, a fan, and the necessary flues to complete the equipment.

The bag house constructed has a bag cloth filtering area of 136,000 sq. ft., provided by 960 fume bags, 18 in. in diameter by 30 ft. in length. A No. 20 Sirocco fan of 180,000 cu. ft. capacity delivers the gases from

the converter plant to the bag house. The converter plant consists of five stands of horizontal, cylindrical type shells 96 by 150 in. in size. A common flue received the gases from all stands. The admission of outside air was mainly relied upon for cooling the gases to the desired temperature. Means were provided for by-passing gases to the stack in case of excessive temperatures. Temperature control of gases was made possible by the installation of recording thermometers.

Converting operations were begun in February, 1913, with the bag house equipped with woolen fume bags of cross-woven fabric, 24 threads to the inch, warp and filling. As regards the converting operation itself, test runs had shown that no difficulties attended the handling of this type of matte and operations were conducted in a manner parallel to the treatment of copper matte.

For a period of two months no difficulties were encountered. The production of by-products, however, appeared as a disadvantageous feature. The passage of 45 per cent. of the lead contained in the matte, to the slag, necessitated its re-treatment in the blast furnaces.

Bag-house difficulties began after several months of operation. Leakage developed first through perforations at the tops of the bags. While leakage of water through a prepared roofing material, that had become affected by the converter gases, was partly responsible for this condition, it soon became apparent that the destruction of the fabric was due to condensation of acid inside the bags. This condensation manifested itself in the form of spots, first at the top and later at the bottom of the bags, about 5 to 6 ft. at each end being affected. At the points where it appeared the fabric became weakened by acid corrosion and would eventually be blown out by fan pressure, leaving enlarged perforations. The nap of the cloth was also rapidly corroded away, giving a screen-like appearance to the bags, and finally a general weakening of the whole fabric resulted in their being torn from the thimble floor. Frequent renewals of bags became necessary to prevent leakages, and inability to correct the underlying causes made it necessary to discontinue operations pending a thorough investigation of the problem.

A solution of all the difficulties required either the destruction of acid by the addition of a neutralizing agent, such as ZnO or CaO , or the production of a fume in itself sufficiently alkaline to preclude the formation of acid, and the production of a converter slag sufficiently low in lead to permit of its disposal by pouring back into reverberatories.

As a result of considerable experimental work, the process as outlined at the beginning of this paper was worked out. Tests carried out intermittently during the day shift in a converter set aside for the purpose showed a rapid formation of a uniform magnetite coating on the walls of the converter. It was expected that difficulty would be encountered due to the gradual closing in and reduction in capacity of the converter.

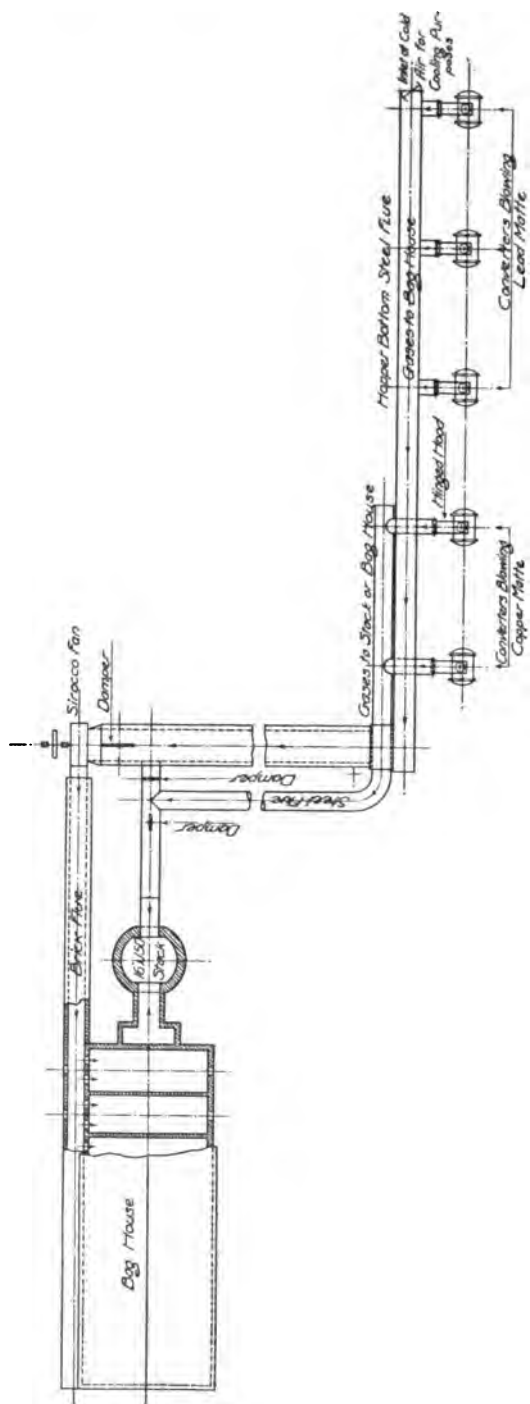


FIG. 1.—ARRANGEMENT OF CONVERTERS, FLUES, AND BAG HOUSE FOR BLOWING LEAD MATTE.

being heaviest at the mouth, due to spattering of the charge, and running out toward the bottom. The fume produced was more actively alkaline, the percentage of zinc eliminated from the converter as ZnO approximating 50 per cent., as against less than 10 per cent. under the previous conditions. It was not deemed necessary, therefore, to consider the addition of a neutralizing agent to the gases.

Preparatory to putting the process into continuous operation, changes were made in the flue system in order that gases from lead converters only would be filtered through the bag house. Three converter stands were consequently connected with a separate flue for lead-matte work, the remaining two stands being connected directly to the stack for copper-matte blowing. Fig. 1 shows the present arrangement. The necessity for better temperature control was realized and flues were equipped with dampers to control the admission of outside cooling air. An automatic alarm system was installed, to record by gongs and colored lamps the passing of the allowable maximum and minimum temperatures. The operating range of temperature of the gases entering the bag house is maintained at from 210° F. to 230° F. The bag house was equipped with bags from the La Porte Woolen Mills, Pendleton Woolen Mills, and Knight Woolen Mills. The former are cross-woven and have 24 threads to the inch, warp and filling; the latter is a twilled weave bag.

Converting operations were begun in February, 1914. The formation of a monolithic magnetite lining, as stated, did not develop. Rather rapid building up of the converter mouth at first gave considerable trouble; however, enlargement of the mouth relieved this condition materially. Corrosion at the tuyère line proved to be somewhat more severe than results from the blowing of copper matte, and more frequent patching is necessary.

The time required for the blowing of a charge of 10 tons is about 2 hr., the time depending on the care given to the tuyères. Frequent punching is necessary. A tendency on the part of the tuyères to blind when silica is absent is very marked. Fume is evolved in dense clouds as soon as blowing begins. The volume of fume gradually diminishes after the blow is half over, and finally, if the blowing is continued long enough, fades out. The end of the blow is thus indicated by the character of the fume and flame discharged from the converter, the flame becoming more and more visible as the fume diminishes. The charge in the converter at this stage contains about 1 to 1.5 per cent. lead and has been reduced in weight to approximately 65 to 70 per cent. of the matte charged. The fume produced under the new conditions does not seem to affect the bags materially, conditions at the bag house being greatly improved. After three months of operation only four bags had failed.

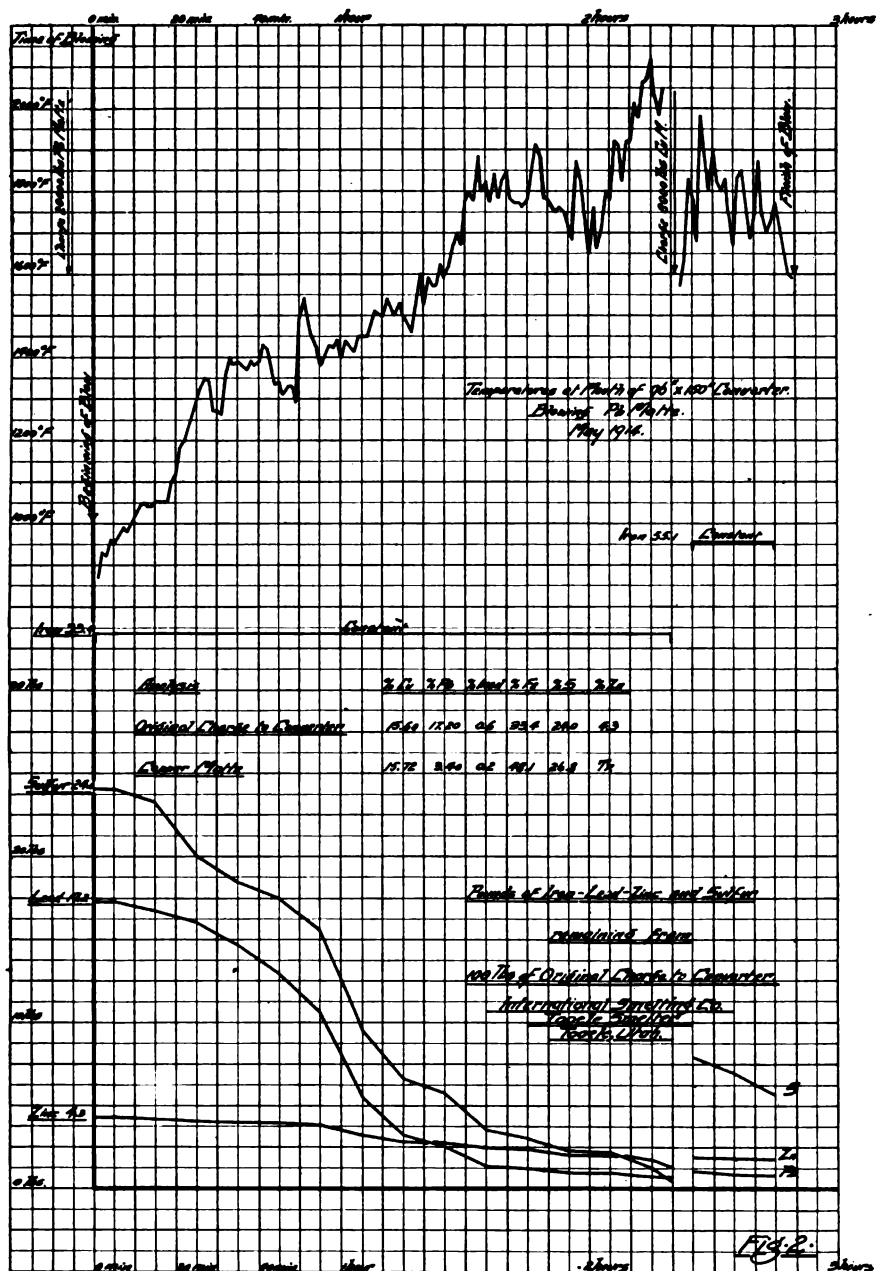


FIG 2.—TEMPERATURE AND ELIMINATION CURVES

These failures were due to apparently defective material, in that one failed through splitting at the seam and the remainder were torn, shaking of the bags appearing to have gradually opened up and extended flaws in the fabric. No acid spotting of the bags is noticeable.

To show the rate of elimination of lead, zinc, and sulphur from matte and the temperatures generated when the blow is continuous and the tuyères are kept fairly free, elimination and temperature curves applying to a single test are given in Fig. 2. These curves are derived from data obtained during a test when copper matte was added at the end of the blow to reduce copper oxide. Iron is taken to be constant in the converter. These curves show about 75 per cent. of the lead and 65 per cent. of the sulphur are eliminated in 70 min. of a 140-min. blow. Zinc is eliminated at a fairly uniform rate throughout the blow.

The following are comparative figures of matte blown and products formed under the former practice as compared with the present practice:

1913—*Converting Lead Matte with the Addition of Siliceous Fluxing Ores*

	Pb, Per Cent.	Cu, Per Cent.	Ag, Oz. Ton.	SiO ₂ , Per Cent.	Fe, Per Cent.	CaO, Per Cent.	S, Per Cent.	Zn, Per Cent.
Matte	16.5	7.66	32.8		37.4		22.5	5.0
Siliceous ore	3.4	0.3	21.3	62.2	6.0	5.1	-0.7	
Converter slag	6.8	5.8	9.2	22.0	38.7		3.2	
Bag-house fume	63.3	1.2	5.6	1.1			67.5	2.5

1914—*Converting Lead Matte without the Addition of Siliceous Fluxing Ores to Lead-Matte Charge*

Lead Elimination Blow:

	Pb, Per Cent.	Cu, Per Cent.	Ag, Oz. Ton.	SiO ₂ , Per Cent.	Fe, Per Cent.	CaO, Per Cent.	S, Per Cent.	Zn, Per Cent.
Lead matte	15.0	9.05	20.3		37.9		23.0	5.4
Transfer slag to copper converter	1.4			1.7	59.1		1.8	4.0
Bag-house fumes	64.2	0.37	5.7	0.2	0.4		6.00	10.4

Copper Converter Blow:

Copper matte	22.2	21.8		41.7		26.6	
Siliceous ore	3.4	0.3	17.1	75.1	7.4	3.0	0.8
Converter slag	2.3	1.79	1.6	23.4	48.8	1.5	1.2
							2.9

As indicated by these analyses, the converter slag from present practice can be disposed of through either reverberatories or blast furnaces, depending on economic conditions. The high active alkalinity of the fume, as indicated by the percentage of ZnO present, should afford ample protection against acid corrosion for the fume filter bags, the condition of the bag house after nearly four months of operation seeming to vindicate this view.

In conclusion, I desire to express my appreciation of assistance rendered by A. Austin, J. C. Welch, and A. C. Wandel in the preparation of this paper.

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The Descriptive Technology of Gold and Silver Metallurgy

BY A. W. ALLEN, CAMBRIDGE, ENGLAND

(Salt Lake Meeting, August, 1914)

THE technological study of the treatment of gold and silver ores has been largely responsible for the phenomenal strides which have marked the progress in this branch of metallurgy during recent years. In no other application of science to industry is system more imperative at every stage; correct formulation of result is only second in importance to efficiency of operation.

Progress in method has been largely the result of individual initiative, aided by the work of technical journals and metallurgical societies.

There still remains a field for profitable discussion, and it is my intention to deal with some inconsistencies of expression common to a number of writers on the subject; and also to draw attention to the false impressions created by a statement which may, possibly, be only unintentionally misleading. I also take the opportunity of tracing the source of both metallurgical and economic result; and to question the desirability of the methods usually employed in arriving at a final statement.

In the first place, I wish to deal with the question of extraction. This term may mean anything or nothing, and it is often used to mislead. In a report of a mining engineer as to the result of operations subsequent to the remodeling of a treatment plant prominence was given to the statement that the extraction had amounted to 91 per cent. On examination it was found that only a small percentage of the so-called extraction was calculated on actual bullion return, a proportion being figured from the gold left in the zinc boxes, and not recovered. In addition to this, a proportion of the extraction was said to be due to gold in concentrate. As there was no concentrating apparatus on the property and as concentration did not form a part of the scheme of treatment the matter was investigated, with the result that the concentrate, of lower gold content than the original ore, was found to be the result of an inefficient pumping system which failed to provide for the whole of the battery product delivered to it. The balance was stacked and, although itself of no marketable value, the gold content was allowed to form a basis of extraction figures.

An ore may be treated experimentally with cyanide solution and 90 per cent. of the gold or silver dissolved. This may be quoted as an extraction of 90 per cent., but it is preferable not to do so, since it is, at best, only an experimental extraction; and by no parallel method of operation in practice could such an "extraction" be actually realized as a recovery. The percentage obviously refers to solution, and the result of the test might as well be stated as such. In practice, an assay or theoretical extraction may be a valuable indication as to the progress of operations, but it is of no value for the purpose of economic comparison unless it happens to check exactly with actual recovery. Again, an ore may be treated by a system of concentration and 70 per cent. of the gold or silver "extracted." Such an extraction is still further removed from actual recovery of metal or its equivalent money value. When the actual weight of the concentrate and the assay of the same are available for computation then the result may be termed an extraction; but when the amount of metal associated with the mineral is determined by a difference in assay content of the ore before and after concentration then the result is obviously only an assay extraction.

In the case where the final product from the reduction works is solely in the form of bullion the statement of result is, comparatively, an easy matter. The assay extraction shows the amount of metal which has been removed during treatment, as indicated by the difference in assay results. The actual recovery must necessarily be based on the actual amount of metal recovered in the form of bullion. In other instances where a proportion of income is derived from the direct sale of bullion, and a proportion from the sale of a concentrated product, then the assay or theoretical extraction is obtained, as before, by estimation from the difference in assay result before and after concentration and cyanidation. As a check upon this result the combined valuation of the bullion produced, together with the gold and silver indicated in concentrate returns, may be calculated as an actual extraction. Under such conditions actual recovery figures are out of the question; and any statement which adds bullion valuation to smelter returns of concentrate treatment should contain a definite designation of the total as an extraction and not a recovery. Extraction and recovery are terms which are not necessarily interchangeable and the time is ripe for due recognition of the fact.

The final result of metallurgical operation should be stated in a form bearing a definite significance and capable of general application for comparative purposes. By common consent it is invariably reported as a percentage of the original metal content in the ore. At this stage of the discussion it will be necessary to introduce other considerations

which will be dealt with in detail. In the statement of assay or theoretical extraction two figures alone are necessary: namely, the result of the assay of the feed and of the residue. The latter figure is generally easily obtained and, for reasons which will be dealt with later, can usually be relied upon. With regard to original metal content in the ore it may be noted that, with most installations, a reliable sample can be obtained by automatic methods after milling. Where the ore is being milled in a cyanide solution containing gold or silver the situation is complicated. It is imperative that the personal element should be eliminated in the sampling of any unground feed and, to this end, automatic sampling and splitting of a considerable proportion of the tonnage handled are imperative. With a reliable knowledge of the metal content of both feed and residue the control of operations is considerably simplified; and the figures obtained can be subsequently checked against the actual extraction or actual recovery figures obtained by direct valuation of bullion, concentrate, or other final product; or from smelter returns giving actual metal content in such product.

Theoretical extraction may be calculated daily, but in order to average the figures obtained a knowledge of the tonnage being handled is necessary. With these data a forecast can be made as to probable yield.

In dealing with actual extraction or actual recovery it has already been pointed out that the total amount of metal ultimately produced can be found by direct valuation. Two other figures are needed before the percentage of actual extraction or actual recovery can be estimated. One of these refers to tonnage, of which a daily estimation is imperative. The difficulty of calculating the actual weight of dry ore going to the mill has already been discussed elsewhere¹ and it is generally conceded that tonnage calculations are best made after the ore has been ground and by means of direct calculation or specific gravity methods. A disadvantage in the latter method of tonnage estimation has been suggested by T. T. Read,² who refers to the possible formation of colloid hydrates during fine grinding. I should like to see further discussion on this point and in the meantime I would tentatively suggest that the colloidal state may not be induced by simple comminution. Recent research would seem to allow us to assume that all substances occur in the colloidal, if not in the crystalloidal state; and from Cornu we learn that "the gels of the mineral kingdom are the typical products of every normal weathering process." I do not think that any amount of fine grinding could transform kaolin into colloidal clay.

In connection with the specific gravity method of arriving at tonnage figures it may be added that, in the case where slime treatment is continuous, there are obvious advantages to be gained by the inclusion in

¹ A. W. Allen: *The Estimation of Tonnage*, *Mining and Scientific Press*, vol. civ, No. 6, p. 308 (Feb. 24, 1912).

² *The Estimation of Tonnage*, *Mining and Scientific Press*, vol. civ, No. 12, p. 443 (Mar. 23, 1912).

these the actual dry tonnage being handled can be obtained by specific gravity methods.

In addition to the tonnage figure, an additional estimate, dealing with gold and silver content either before or after treatment, will be needed before actual recovery percentage can be calculated. Such a figure must be reasonably accurate and for a number of reasons the assay of residue should be used for this purpose. This particular sample can be relied upon as being more representative than the feed sample for the following reasons:

1. The ore is finely ground and thoroughly mixed during treatment before being sampled.
2. Coarse metal of value has been removed, thus reducing the possibility of error in assaying.
3. The final treatment process allows of a correct average sample being taken from a large bulk of material by simple means.
4. Where classification is efficient there is no chance of the personal element entering into the question; neither is it possible to vary the metal content at will except in the rare instance where residues are discharged with different percentages of moisture, the latter being in the form of valuable solution.

An additional reason why reliance may be placed on the residue sample lies in the fact that the figures obtained from the assay of such material may, in the great majority of instances, be regularly and periodically checked by fresh sampling and assaying. This is an important point.

A further advantage is seen in instances where, if the assay of feed had been used, an actual extraction at times of over 100 per cent. would be recorded. There is always a possibility of the actual extraction reaching or exceeding the theoretical or assay extraction, but there is no possibility of the actual extraction reaching or exceeding 100 per cent. A logical method of arriving at final result should preclude the possibility of absurdity of statement.

The most important phase of descriptive technology dealing with metallurgical operations undoubtedly refers to final result; and in this connection the value of clarity of statement is obvious. I recently attempted to systematize the information, published in book form, and dealing with metallurgical operations on a number of important properties. I found that no comparison of result was possible. In one instance only was a serious attempt made to give information to the reader. In this case the percentage of gold recovered as bullion was calculated from the average assay content of the original ore multiplied by the tonnage treated. In three instances the "extraction" referred to certain assay results, and was obviously only as assay extraction. In two instances

there was no indication as to how the published extraction figures had been obtained. In two instances the results were either contradictory or were based on figures bearing no connection whatever with the actual metallurgical extraction. In the final instance the phrase "metallurgical recovery" had been used to include the metal in the rich ore sorted at the breaker plant, the metal shipped in the concentrate, and the metal really recovered as bullion. In this instance the word "recovery" seems to have been more sadly misused than the previous term "extraction." The percentage figures tell us nothing as to the metallurgical efficiency of the plant or the suitability of the scheme of treatment. As the amount of ore shipped to the smelter rises so the "recovery" increases. By shipping all the ore to the smelter the "recovery" would reach 100 per cent.

A definite standardization of such results is badly needed, and accounts of metallurgical operations would have an added interest and value if accompanied by a clear and concise *résumé* of the actual metallurgical result of such operations. Actual recovery might well be stated in all instances where the final product is in the form of bullion. Actual extraction might be stated in all cases where the final product is in a less concentrated form, and where the yield consists of bullion as well as concentrate, etc. If extraction and recovery figures are unobtainable then a plain statement of the fact is alone necessary. Assay extraction by another name is of little or no value except when it coincides with actual recovery. In the latter event it might just as well be stated as an actual recovery. Assay extraction can always be checked and it is the actual extraction or actual recovery which alone is of economic significance and metallurgical importance.

The point is often overlooked that facts as well as opinions are looked for by the majority of readers. On the subject of filtration of slime pulp fully 90 per cent. of descriptive accounts of filters and filtration omit the fundamental point as to whether or no the filter can or does work efficiently—they avoid a plain statement as to what percentage of displacement of dissolved metal the filter is able to effect.

On the question of the statement of metallurgical result I venture on some definitions:

1. *The assay or theoretical extraction* is the amount of metal which is indicated by assay results as having been removed from the ore during treatment.

2. *The actual extraction* indicates the amount of metal removed from the ore and isolated either as bullion or in other concentrated form. Such actual extraction is estimated by the direct valuation of the bullion and the statement of return from smelter or buyer showing actual metal content in concentrated product, upon which the sale was effected.

3. *The actual recovery* can only be based on the direct valuation of bullion, and the term is only available for use when the bullion produced is approximately the same market value, *per se*, as the contained gold and silver.

In the calculation of percentage extraction or recovery the following summary will explain the method of statement:

Assay Extraction

Where a = the assay of feed,

z = the assay of residue, and

t = the tonnage represented,

then $\frac{100(a-z)}{a}$ = *percentage assay extraction*.

and $\frac{a_1t_1 + a_2t_2 + a_3t_3 + \dots}{t_1 + t_2 + t_3 + \dots}$ = *average feed assay = A*,

and $\frac{z_1t_1 + z_2t_2 + z_3t_3 + \dots}{t_1 + t_2 + t_3 + \dots}$ = *average residue assay = Z*.

then $\frac{100(A-Z)}{A}$ = *average assay extraction percentage*.

In the case where an appreciable tonnage is associated with the metal extracted the average assay content in residue *per ton milled* must be calculated and used in the formula, otherwise actual extraction will exceed assay extraction.

Actual Extraction

Where M = the amount of metal in bullion, concentrate, or precipitate,

T = the tonnage milled,

t = the tonnage of concentrated product removed, and

Z = the average actual residue assay,

then $\frac{100 M}{M + Z(T-t)}$ = *average actual extraction percentage*.

Actual Recovery

Where M = the amount of metal in bullion,

T = the total tonnage, and

Z = the average residue assay,

then $\frac{100 M}{M + TZ}$ = *average actual recovery percentage*.

It is, of course, assumed that the same unit of weight would be used throughout in the calculation of metal content in ore, bullion, concentrate, precipitate, etc.

Any metal in a concentrated product removed from the ore after or during reduction, and the value realized elsewhere by direct sale or otherwise, may be included in the statement of actual extraction. The total amount of metal so extracted would form a basis of calculation for percentage actual extraction. On the other hand, the metal in rich ore

abstracted before treatment of the main bulk can form no part of, nor may it be used to influence, a synopsis of such treatment. Its cost of extraction cannot be included as an expenditure *per ton milled* and its content cannot be added to any statement of recovery made in the reduction plant without falsifying figures which might be obtained dealing with the result of actual metallurgical treatment.

The next consideration refers to the case of calculation of extraction from ore being treated for both gold and silver content, and in the case where both metals occur in quantities of economic significance. A numerical average is of no significance, neither is extraction based on combined metal content before and after treatment of other than metallurgical import. The extraction in current metal value is sometimes of use for comparison, but such a method of statement is not of value for general purposes. The reason for this lies in the fact that the result is influenced by market fluctuations which are beyond the control of the metallurgist. The only exception to this occurs in the rare instance when the percentage extraction of both metals is identical. The extraction and recovery percentages should be stated in such a manner that an alteration in the price of one of the metals would not necessitate an explanation as to a rise or fall in the extraction or recovery figures caused solely by such fluctuation.

With regard to the system to be used in the statement of gold and silver content in ore and bullion, it must be admitted that the metric system is preferable to any other. In troy measurement the units bear no definite relation to one another, and there is no regular factor of difference. The use of the troy grain is now practically discontinued, and the juxtaposition of the facts that 24 grains equal 1 pennyweight and 20 pennyweights equal 1 ounce has often led to serious errors in calculation. It is perhaps too much to hope that the centimeter-gram system may be extended to all calculation used in ore treatment within the lifetime of the present generation.

In most English-speaking countries there seems no objection to the use of the troy ounce in the calculation of silver content in ores, and it is in general use in connection with bullion in most places. The logical equivalent for the use of gold ores is the pennyweight (dwt.), which is perhaps preferable to the use of the decimal subdivision of the ounce. The use of the American dollar as a unit of gold content is illogical, although in the technical literature of the United States its use is practically compulsory. It is common to see a statement that a gold ore has a value of \$1. This may mean that the ore assays 1 dwt. of gold, or thereabouts, and this metal if completely recovered in its original state of purity would have a value of \$1. The original statement is really a jumble of contradictions, because the ore may not be worth 10c. when placed on the market. The statement of metal content in terms of

The remarks previously made with regard to extraction are equally applicable in connection with experimental work. Extractions of metal based on differences of assay results should be definitely stated as assay extractions. When they refer to differences in gold and silver content as a result of experimental cyanide treatment such extractions are preferably referred to as solution extractions. They should, in all cases, be distinctly differentiated from probable or possible recovery. These latter figures are only obtainable in the case where the experimental plant is capable of duplicating actual practice in every detail, and where the ore being assayed is not freed from the normal amount of dissolved gold or silver generally found associated with it after ordinary treatment. When preliminary work is carried out on a small scale and where there is no working size filtration or leaching plant available, the assays of the washed ore must be used for the purpose of computation of possible recovery figures. The following ratios are available as a rough guide for this purpose:

Experimental Assay Extraction, Per cent.	Scheme of Treatment	Probable Recovery, Per cent.
100	Amalgamation.....	100
100	Leaching of sand with precipitated solution.....	99 to 99.5
100	Sliming, followed by continuous replacement with barren solution during agitation. Filtration of pulp.....	95 to 98
100	Sliming, followed by filtration and washing.....	90 to 98

These figures introduce the consideration that the more complicated the treatment and the finer the ore is ground the lower becomes the actual recovery of metal. This point is invariably overlooked in reports of experimental work where solution percentage is confused with possible recovery.

The Statement of Working Costs

In conclusion, I would refer briefly to the question of the statement of working costs, and in this connection I would urge the importance of the local segregation of working cost from general expense, in order that operators in positions of responsibility should be given every opportunity of

keeping those expenditures over which they have control under constant scrutiny, so that continued economies in total working costs may be effected and clearly indicated in the returns.

On the question of amortization, or the liquidation of the original cost of the plant by means of payments out of profits of treatment, there is little to be said, because in general mining work the provision of such a sinking fund has been deemed unfeasible. There exists, however, no logical reason why theoretical amortization figures should not be taken into consideration in the first place when some definite scheme of treatment has to be decided upon. In the latter event two factors alone are generally supposed to influence the decision: the probable recovery, and the cost of treatment. If amortization costs were added to the cost of treatment in each case and the matter considered from a strictly business standpoint, it is obvious that these charges, coupled with the necessarily higher operating costs, would afford a sound reason against the selection of any process which involved an unnecessarily extensive equipment, involving the necessity for comminution of the ore beyond an economic limit. Even if amortization charges are not feasible in actual practice, there is no reason why they should not be considered in preliminary estimation work.

On the question of repairs and renewals, I would suggest that in a statement of working costs these two items be distinctly differentiated. I would even go further and suggest that items coming under repairs should be added after the total working costs have been arrived at. My reason for this lies in the fact that actual operating costs are for the purpose of locally checking extravagance and unnecessary expenditure and for the encouragement of the introduction of economies. I would point out that it seldom occurs that mill superintendents who are held responsible for working costs are also responsible for either the scheme of treatment or the initial plant arrangement. Repair costs are largely an index of the class of plant erected in the first instance; and if such an item were isolated and added to the total working costs it would form a powerful argument against misplaced economy in the first instance.

In the statement of construction costs I would urge the necessity for the exclusion of all sums paid for wearing parts from the final statement of cost. These should be segregated out, placed to a suspense account, and charged out as operating expense. The absurdity of charging everything required to complete the plant to construction has been exemplified in every instance, but the practice is still in general vogue. In one instance which came under my notice a ball-mill plant ran with an entire avoidance of liner expense, the principal running expenditure, for over five months. In another case an extensive conveyor plant ran for over two years with no belting expenditure charged out to working costs. As soon as the original equipment deteriorates there is a rise in operating cost, due to

to be blamed is the system of book-keeping responsible for such anomalies.

In placing before the members of the Institute the foregoing remarks on the descriptive technology of gold and silver ore treatment I realize that the subject has been dealt with from a single viewpoint only—that of the practical metallurgist. Many of my contentions are admittedly argumentative and I trust that a critical discussion will tend to throw more light on the subject.

Dip Chart

BY HOWLAND BANCROFT, DENVER, COLO.

(Salt Lake Meeting, August, 1914)

THE writer has observed that some confusion is experienced by many mining engineers in making vertical sections of ore deposits in mining properties which they have examined. Having much examination work calling for the correct interpretation of vein structure and the subsequent

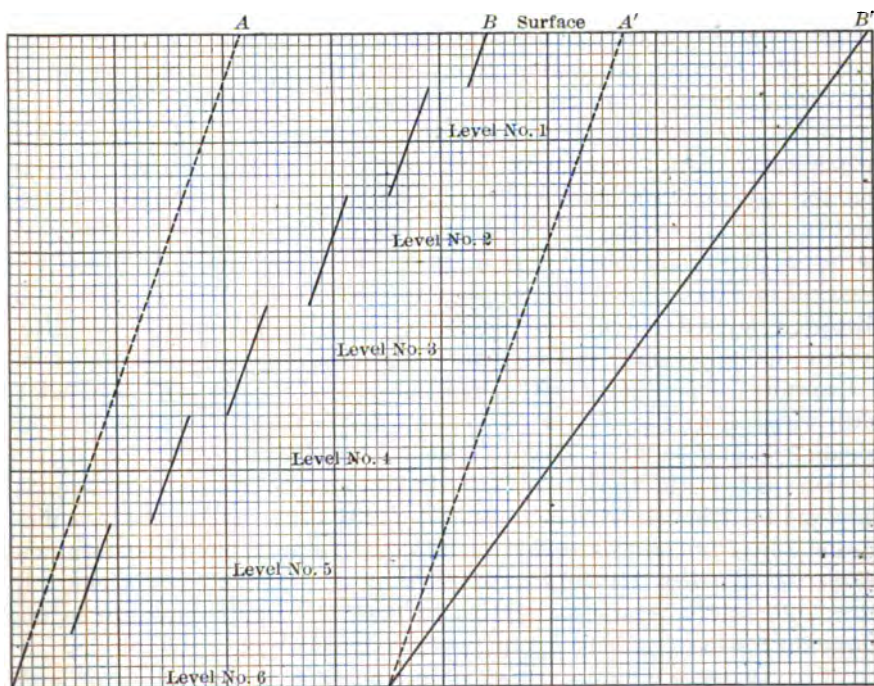


FIG. 1.—CORRECT AND INCORRECT WAY OF PLATTING A VERTICAL SECTION OF A VEIN.

plattling of the same on sections, it has occurred to the writer that the accompanying diagram may be of use to others who at times may be called upon to prepare numerous sections. For example, a great mass of data on strike and dip of ore bodies is collected underground in separate

of the deposit are frequently platted just as recorded underground. The result is that a deposit, which may be continuous from the surface to the lowest level, appears on the vertical section as a series of disconnected fissures having dips ranging from 0° to 90° . This is due to the fact that observations are taken in various parts of the mine and the plane of the vertical section is rarely, if ever, at right angles to the strike of the vein throughout the course of its dip plane.

Fig. 1 represents the correct and incorrect way of platting a vertical section of the same vein where observations have been taken at different levels on a vein having a uniform dip. In this ideal figure it is presumed that each level intersects the vein at the same angle. *A* and *A'* represent the platting of a vein having a uniform dip of 70° on a section the plane of which is at right angles to the strike of the vein. *B* represents the same vein incorrectly platted on a section the plane of which intersects the strike of the vein at an angle of 30° . Observations of the dip and strike of the vein were made on each level and the vein platted with its actual dip. *B'* represents the same vein correctly platted on a section the plane of which intersects the strike of the vein at an angle of 30° .

While the formula for determining the apparent dip of a vein to be platted on a vertical section may be known to many, this method of determining apparent dips involves a very tedious series of computations if the sections are numerous. The accompanying dip chart, Fig. 2, is submitted with the hope that it may save much laborious calculation in preparing vertical sections. This diagram is not original with the writer. D. W. Brunton devised a somewhat similar method for his own use at Butte, about ten years ago, and he now has in the process of manufacture "Brunton's Slope Chart," a 9 by 10-in. cardboard diagram with a swinging arm which can be read to a quarter of a degree and is complete for dips ranging from 0° to 90° . This will be entirely satisfactory for office use.

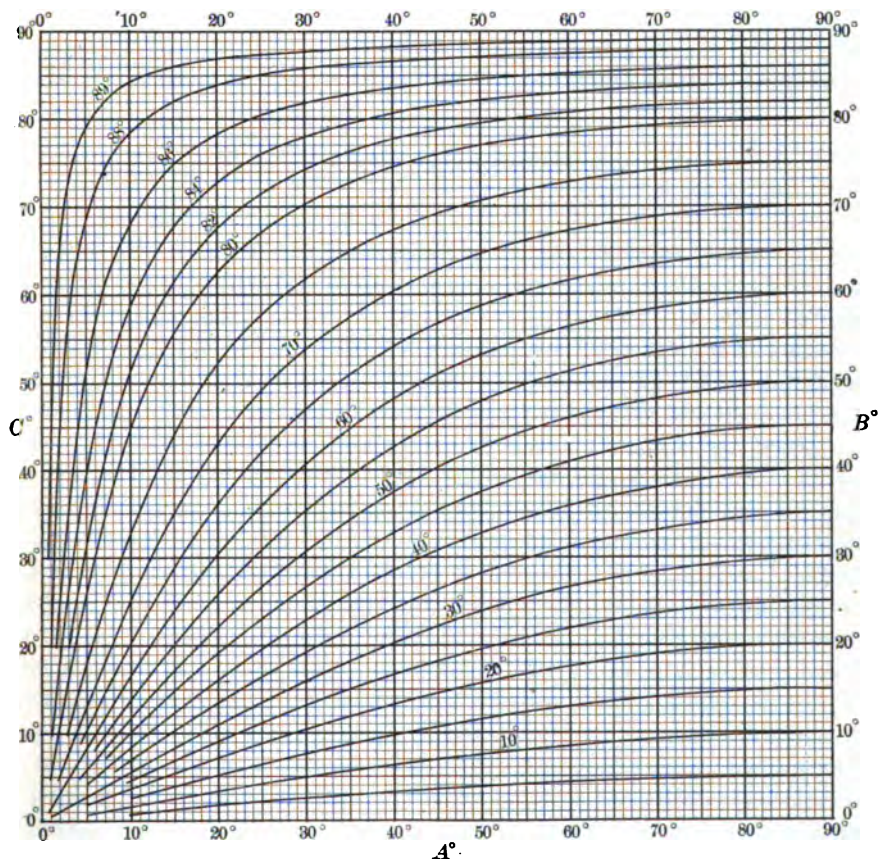
In 1912, D. F. Hewett published¹ a brief description of A Graphic Method for Dips on Geologic Sections. This was accompanied by a diagram, 7 by 7 in., on thin paper, which showed graphically the determinations for platting dip angles differing by 5° . Mr. Hewett's chart is available only to readers of *Economic Geology*. Furthermore, it is too large for field use, and the dip angles have 5° intervals.

The accompanying dip chart, prepared by A. B. Crosley, is of such size that it can conveniently be placed in the back of the ordinary field notebook, and the intervals between angles of dip of veins from 80° to 90° are only 2° . Hence, intersections with the vertical section for veins dipping over 80° can be more accurately platted with this diagram than by Mr. Hewett's.

¹ *Economic Geology*, vol. vii, No. 2, pp. 190, 191 (Mar., 1912).

of the diagram here presented. However, it is not practicable to carry Mr. Branton's chart in the field, its use being confined primarily to the office.

In using this diagram it is to be remembered that the angles indicated at the bottom (*A*) represent the difference in strike of the vein it is



A VEIN WITH DIP B° , INTERSECTING A LINE OF SECTION AT AN ACUTE ANGLE A° , IS PLATTED ON THE SECTION WITH AN APPARENT DIP C° . FORMULA $\tan C^\circ = \sin A^\circ \tan B^\circ$.

FIG. 2.—DIP CHART.

desired to plat and the plane of the vertical section. The curved lines represent dips of veins (B). The points of intersection of the vertical lines (above A) with the curved lines (true dip lines B) at the horizontal lines (C) are the apparent dips at which the veins should be platted on the vertical section. For example, the strike of a vein dipping 86° intersects the plane of a vertical section at an angle of 25° . It should be platted on the vertical section with an apparent dip of $80^\circ 30'$.

The Treatment of Complex Ores by the Ammonia-Carbon Dioxide Process

BY S. E. BRETHERTON, SAN FRANCISCO, CAL.

(Salt Lake Meeting, August, 1914)

MOST metallurgists appreciate the great need of a process for the extraction and recovery of valuable metals from complex ore, where the presence of one metal increases the cost of extracting the other metals to such an extent that not any of the metals can be extracted at a profit. We need only refer to some of the mines in Colorado, Utah, and New Mexico, where oxidation has taken place and the sulphur, zinc, copper, and some of the iron and other metals have been leached out, large bodies of oxidized ore remaining, rich in lead, silver, and sometimes gold. These metals have been concentrated by the removal of the other metals, leaving a very desirable ore for treatment in the blast-furnace for the production of lead bullion, and recovery of the silver and gold. If lead is not present, the gold and silver are usually recovered by some method of milling, or the ore is shipped to a smelter, where it is mixed with lead ore or copper-iron sulphide ore from some other mine. In these same mines, when developed to greater depth below the leached or oxidized zone, the ore is found to contain the original mixture of sulphur, zinc, copper, etc., added to, or I should say left with, the lead, silver, and gold values; but instead of adding to the value of the ore, the presence of these elements, especially zinc, makes the copper, silver, and gold of less value, and often worthless. Such was the experience of the oldest mines in Leadville, Colo., until methods of mechanical separation of the lead from the zinc were developed sufficiently to allow of some profit by the making of a concentrate rich in zinc to be shipped to the zinc smelter, and sometimes a lead concentrate which could be shipped to the lead smelter; but even in these cases the waste of the different metals is a very serious matter. In the very best concentrating plants some zinc will be left in the lead concentrate, to be lost and to give considerable trouble in the lead-smelting furnace. On the other hand, some of the lead, with a large percentage of the copper and silver, will follow the zinc into the zinc concentrate, to be, perhaps, recovered by smelting the cinder or clinker left in the re-

this residue is often greater than the combined values of the gold, silver, and copper in it. The loss of silver by volatilization is also much greater than the loss of either gold or copper, so that zinc ore rich in silver is not suitable for this process. On the other hand, a sulphide ore containing the same amount of zinc as the above complex ore, with none of the other metals, can be easily concentrated. Often it is a simple matter to make a concentrate containing from 40 to 60 per cent. zinc (as is now being done in the middle Western States) which can be sold to the zinc smelters, who will pay a much higher price per unit for the zinc contents at 60 per cent. than at 40 per cent. Any one familiar with metallurgy will appreciate that one reason for this is, that the higher the percentage of a metal in an ore, the higher will be the percentage of recovery obtained.

A sulphide copper ore containing, say, 3 per cent. and upward of copper and little or no zinc is very cheaply reduced to a copper matte, if in sufficient quantity and within a reasonable distance of some railroad. The presence of silver and gold is of great benefit in assisting the copper to make profits, for the reason that the freight on precious metals, when shipped with either copper or lead products, is almost negligible. But supposing such ore to contain 14 to 40 per cent. zinc with barium and other impurities, and the structure of the ore not suitable for any mechanical means of concentration; in such cases the zinc content decreases, and sometimes destroys, the value of the copper, while the ore is not rich enough in zinc for the recovery of zinc by any ordinary methods. Such ore as this, containing copper, lead, silver, gold, and zinc, occurs in a number of large mines in California, Colorado, Utah, Nevada, Arizona, and New Mexico, some of which mines are being worked successfully by concentrating the ore into zinc concentrates and lead concentrates, with sometimes a copper concentrate; but the per cent. of loss of the different metals is excessive. There are a great many mines the ore of which cannot be concentrated by any known method. I have had such ore as this in mind for several years, and after investigations I decided to have the ammonia-carbon dioxide process tested and developed. This process is the only one I know of for the treatment of the complex zinc sulphide ores of the Rocky Mountains by which the gold, silver, copper, and lead can be recovered and saved, after first extracting and saving the zinc, thus turning the zinc into an asset instead of a liability, and enabling a great number of mines now idle to become profitable.

There is another view to be taken of this situation: The United States government should and perhaps will insist on all railroad companies and large mining companies treating their timber to prevent decay, except, perhaps, that used for temporary structures; first, to lessen the number of accidents, and second, to stop the unnecessary waste and destruction

of timber in the country. This would not be any hardship on either the railroad companies or the mining companies, for, looking at the subject from an economic standpoint, it would be a great benefit to them to have their timbers last from 12 to 20 years, instead of only 2 to 5 years, as is often the case at present. The high price of creosote almost prohibits its use, except perhaps as an outside coating for timber already treated with a cheaper wood preservative, such as zinc chloride. (The question is, could not some cheaper oily substance be used for a surface coating to timbers in wet ground to prevent the leaching out of the zinc chloride?) I have mentioned the use of zinc chloride for wood preservative purposes to show that there is a demand for zinc in this direction, which will help to solve the problem of marketing the zinc when recovered from the immense quantities of complex zinc ore in the Rocky mountains. At the same time that the zinc is being recovered from such ore, considerable quantities of lead, copper, silver and gold will be added to the world's supply. In addition to the demand for zinc to be converted into zinc chloride for wood preservative purposes, the use of the metal for roofing, for zinc foil for wrapping tea and other packages, and for the manufacture of brass is constantly growing; as is also the use of zinc oxide for the manufacture of pigments for painters, and in the manufacture of rubber goods.

W. R. Ingalls, in his *Metallurgy of Zinc and Cadmium* (p. 668, writes):

"The merits of zinc white as a pigment were summarized by Maximilian Toch in a paper recently read before the New York Section of the Society of Chemical Industry. He stated that the consumption of white lead is decreasing and that it should not be used alone. The specifications of the United States Lighthouse Department demand a mixture of 75 per cent. of zinc oxide and 25 per cent. of white lead. Zinc white mixes well with other pigments, particularly with white lead. Zinc oxide made directly from ore is more durable than that made by burning spelter."

The process¹ I have in mind for the treatment of the complex zinc ores is briefly as follows: Grind the ore, roast, regrind, leach out zinc with ammonia and carbon dioxide, precipitate from the solution any copper which has been dissolved with the zinc, and put back in the residue. Smelt the residue for the recovery of copper or lead, with the gold and silver. The ammonia and excess carbon dioxide are distilled from the solution containing the zinc and recovered for further use. As the ammonia is being distilled, the zinc is precipitated as a pure white basic zinc carbonate, filtered out, and then calcined to zinc oxide, or the basic zinc carbonate can be reduced direct to either zinc chloride or spelter. Due to lack of funds, we have not yet installed a large commercial plant at the Afterthought mines, in California, so that at present we only have a 150-lb. capacity plant.

Since writing my last article for the Institute, our metallurgical chem-

¹ *Bulletin* No. 80, August, 1913, pp. 1481 to 1487.

ist, F. L. Wilson, has made a number of continuous runs, each of several days' duration. The following is a copy of his last report of a 14 days' continuous run: (It should be noted that this report is based on a test of our lower-grade zinc ore containing about 20 per cent. zinc. The extraction from our ore containing 30 to 36 per cent. zinc is from 85 to 95 per cent.).

Ingot, Cal.,
March 6, 1914.

S. E. Bretherton,
Mills Building,
San Francisco, Cal.

Dear Sir:

I herewith report results on test No. 65 secured from 12 complete cycles and during a period of 14 days. Two days were required for unavoidable repairs.

Preparation of Ore

The raw sulphide ore was crushed to 30 mesh (maximum). Its sulphur (soluble) content varied from 18.7 to 20.5 per cent. and averaged 19.7 per cent.

A sulphatizing roast was given the ore. The weight of the roasted ore showed a slight increase over the raw ore in several cases, the average weight remaining nearly the same.

The roasted ore was wet slimed for 10 hr. in each case. No attempt to classify the product was made. The coarse product remaining on 150 mesh was but little in each case.

Extraction

Pressure throughout agitation, filtering, leaching, etc., was 30 lb.

Time throughout the cycles was less than 8 hr. average. This time was inclusive of time taken to operate valves, etc.

Time of Cycle

	Agitation	Filtering	Leach Wash	Water Wash	Total Time
	Hr. Min.	Min.	Hr. Min.	Hr. Min.	Hr. Min.
A.....	4 20	50	1	... 45	6 55
B.....	3 15	20	1	1	5 35
C.....	3	13	2	2	7 13
D.....	3	20	3 30	1 30	8 20
E.....	3	18	3 40	... 50	7 48
F.....	3	20	4	... 20	7 40
G.....	3 05	20	3 30	... 15	7 10
H.....	3 15	20	3 25	... 20	7 20
I.....	3	20	4	... 20	7 40
J.....	4 22	25	3 30	... 30	8 47
K.....	3	25	3 40	... 30	7 35
L.....	3 35	35	2 40	... 30	7 20

30-lb. charge; whereas, with a 25 per cent. zinc ore a 40-lb. charge, and 30 per cent. zinc ore a 50-lb. charge. This is due to the relative shrinkage in volume of the residue when the zinc is extracted.

Zinc extraction varies from 79.7 per cent. on an 18.6 per cent. roasted zinc ore to 84.9 per cent. on a 20.72 per cent. roasted zinc ore.

Copper extraction was not recorded, as the cement copper was filtered with the residue.

Silver extraction is negligible, which remains in the residue. The slight extraction is no doubt due to discrepancies in the weights of the net ore and residue.

Analysis and Extraction

	Roasted Ore				Residue				Extraction	
	Ag Oz.	Cu Per Cent.	Zn Per Cent.	S Per Cent.	Ag Oz.	Zn Per Cent.	S Per Cent.	Loss in wt. Per Cent.	Ag Per Cent.	Zn Per Cent.
A.....	7.56	2.76	19.00	5.92	12.34	8.20	0.35	39.1	0.53	73.7
B.....	8.28	2.96	19.40	6.33	13.20	7.60	0.41	33.8	Nil	74.4
C.....	8.16	2.84	20.30	6.25	12.76	7.40	0.39	42.4	0.98	80.8
D.....	8.16	2.84	20.30	6.25	12.70	7.20	0.41	37.3	0.24	78.1
E.....	8.20	3.12	19.80	5.04	12.96	6.90	0.55	42.9	0.98	80.2
F.....	7.48	2.96	21.40	6.84	11.20	6.52	0.58	41.4	1.21	82.1
G.....		3.00	20.60	6.74		6.75	0.66	42.9		81.0
H.....			20.80	6.78		6.82	0.62	43.8		81.6
I.....			19.7	7.56		6.55	0.67	41.2		80.6
J.....			18.60	7.02		6.86	0.62	45.2		79.9
K.....			19.30	7.30		6.45	1.10	44.0		81.3
L.....			20.72	7.74		6.45	0.77	51.1		84.9

Recovery

Boiling out with the use of the extra still kept the amount of solution down to a minimum. Live steam was admitted into the third still at intervals to facilitate the boiling out of the zinc. The excess solution was used in washing the residue and saved to grind the ore in.

Zinc precipitate was off color for the most part. The steam from the first still carried over rust and oil enough to discolor. The rust was caused by the filter press and pipes having been idle for some months previous to this test. Also fine rust and ore were carried into the receiving still from the ore filter. Decantation from the copper precipitator was tried, but the rust occurred as before. By refiltering the solution through a finer filter (filter paper placed on the leaves of the press) the resultant zinc carbonate was as white as could be desired. The volatile moisture, etc., contained in the precipitate varied from 65.2 per cent. to 78.6 per cent. The recovery of the zinc basic carbonate and hydrate was 99.7 per cent. This 0.3 per cent. was left in the solution.

Zinc Recovery

	Zinc Precipitate			
	Weight Pounds	Zinc Per Cent.	Zinc Recovered Pound	Zinc Extracted Pounds
A.....	9.05	53.2	4.82	5.94
B.....	7.86	53.0	4.17	3.68
C.....	12.23	54.4	6.66	3.86
D.....	5.24	53.6	2.83	4.00
E.....	9.20	53.8	4.95	4.58
F.....	12.65	53.0	6.71	5.34
G.....	4.94	56.0	2.76	4.44
H.....	9.71	51.0	4.96	5.78
I.....	12.70	52.2	6.64	5.59
J.....	6.94	51.6	3.58	4.48
K.....	4.56	53.2	2.43	4.49
L.....	13.90	55.4	7.70	6.32
		Total	58.21	58.4

Losses

Ammonia losses were 4.44 per cent. during the run (not counting A, B, C and D). Of this amount 0.28 per cent. was unaccounted for numerically, but due to intermittent leaks in the system, 17 per cent. of the total ammonia was tied up as fixed ammonia. No lime was used to free this in the run. Compressed air was used to drive the charge through the filters and the air remaining in the agitator and copper precipitator was released through a scrubber. In like manner the absorbers and separator were connected with scrubbers. Washing the residue first by hot solution from the first still and then with steam brought the residual ammonia down to from 1.3 lb. to 3 lb. per ton of ore. The zinc precipitates held some back as a salt, there being no smell of volatile ammonia.

	Per Cent.
Caught in scrubbers (lutes).....	2.25
*Remaining in residue.....	0.67
*Remaining in zinc precipitate.....	1.24
*Unaccounted for.....	0.28

Carbon dioxide losses were mostly in the residues as tabulated below. The total loss was 24.72.

* This loss of ammonia, with the exception of the trace lost in the residue and precipitate, was higher than usual with us, and was due to defective old valves and pipes, and the use of compressed air. Practically all of the ammonia can be saved by drying the residue and zinc precipitates. The carbon dioxide can be recovered by calcining products.

	Ammonia		Carbon Dioxide	
	Residue Per Cent.	Pounds Per Ton of Roasted Ore	Residue Per Cent.	Zinc Precipitate Per Cent.
A.....			1.43	6.68
B.....			1.99	7.00
C.....			2.54	7.70
D.....			1.5	6.00
E.....	0.525	10.5	0.62	11.00
F.....	0.430	8.6	0.80	8.60
G.....	0.355	7.1	0.50	8.20
H.....	0.150	3.0	1.22	7.40
I.....	0.140	2.8	0.80	11.40
J.....	0.125	1.5	1.00	8.60
K.....	0.100	2.0	1.20	9.20
L.....	0.065	1.3	0.70	7.60

Remarks.—In recovery of the ammonia by boiling out, the ammonia content of the liquor in the absorbers could be controlled at will; 15.3 per cent. ammonia was obtained in one instance in solution to be used again.

Respectfully submitted,
F. L. Wilson,
Metallurgical Chemist.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Leaching Copper Products at the Steptoe Works

BY W. L. AUSTIN, RIVERSIDE, CAL.

(Salt Lake Meeting, August, 1914)

At the Steptoe metallurgical plant, where ore of the Nevada Consolidated Copper Co. is beneficiated, a small copper-leaching annex has been in operation treating flue dust from roasting-furnace dust chambers, together with a siliceous oxidized ore. This leaching plant was designed to obviate smelting low-grade siliceous flue dust, which interfered with the operation of the reverberatories by forming a blanket on the charge.

The major portion of the copper content of the flue dust accumulating in the upper chambers of the roaster flue, and at the stack, is water soluble, and also contains much free sulphuric acid and soluble sulphates. It was found that sufficient acid could be obtained by mixing this material with water to provide a lixiviant for leaching the siliceous copper ore mentioned. This was the sole source of the lixiviant used in the operation to be described.

An analysis of the flue dust showed it to be composed as follows:

	Per Cent.
Copper.....	3.00
SiO ₂	23.41
Fe.....	4.08
CaO + MgO.....	2.72
Al ₂ O ₃	6.07
Free H ₂ SO ₄	19.94
SO ₃ combined as sulphate.....	25.10
H ₂ O combined with the sulphates.....	10.67
O.....	2.50
Total.....	97.49
Copper soluble in dilute H ₂ SO ₄	2.56

The oxidized ore, locally known as carbonate, has the following composition:

	Per Cent.
Copper.....	3.47
SiO ₂	71.1
Fe.....	2.3
CaO.....	0.6
Al ₂ O ₃	9.5
S.....	0.9
Total.....	87.87
Copper soluble in dilute H ₂ SO ₄	3.23

A screen analysis of the "carbonate" ore discloses the distribution of the copper throughout the several sizings:

Mesh	Weight, Pounds	Copper, Per Cent.	Per Cent. of Total Copper
+ ¼	19.5	3.86	21.8
+ 10	20.5	3.85	22.9
+ 20	11.8	4.14	14.1
+ 30	9.8	3.82	10.8
+ 40	5.0	3.50	5.1
+ 60	7.5	3.65	7.8
+ 80	4.7	3.47	4.6
+100	4.4	3.00	3.7
+150	3.0	2.45	2.1
+200	4.5	2.00	2.5
-200	9.3	1.76	4.6
Average.....		3.475	100.0

It is evident from these analyses that the copper in this ore is largely mineralized as silicate, for cryptocrystalline chrysocolla does not break into small particles as do the more brittle carbonates. The screen analysis shows that when crushed to ¼ in. 70 per cent. of the copper mineral remains on a 30-mesh screen. It is also interesting to note that the copper in the mineral crushed to this degree of fineness is readily soluble in dilute sulphuric acid. It has been repeatedly observed in other cases when treating ore in which the copper is present largely as silicate that a high percentage of the metal may be extracted with dilute sulphuric acid.

The method of operation pursued at the Steptoe plant consists in hydraulicking the piles of flue dust which have been withdrawn from the dust chambers, as an economical means of dissolving the soluble contents and of transportation to the tanks. The stream of water is conducted to vats, or impounded in dams, where the mud is allowed to settle. The clear solution resulting, which contains the soluble salts and is now the lixiviant, is drawn from these reservoirs to a wooden vat, fitted with a filter bottom. This is the ore-leaching vat proper. It is 40 ft. in di-

ameter and 10 ft. deep. It is filled with crushed "carbonate" ore to a depth of 7 ft.; the ore column through which percolation of $\frac{1}{4}$ -in. material is effected is therefore about 7 ft. high.

The excess space in this vat is used to store the lixiviant, for the hydraulicking operation is intermittent, while ore leaching is continuous. Enough lixiviant is accumulated above the charge in this vat to continue leaching during the night.

If the filtrate from the ore leaching vat (the lixivium) still contains considerable free acid, it is pumped back and passed through the ore bed a second time. Otherwise it is allowed to flow through a filter of "carbonate" ore, and thence over scrap iron placed in precipitating boxes.

About 85 per cent. of the free acid carried by the lixiviant is utilized in the ore-leaching vat, the remaining 15 per cent. being left in the lixivium flowing over the scrap iron, to prevent formation of basic iron salts and insuring clean precipitate. About 2 lb. of scrap iron are consumed per pound of precipitate produced.

The cost of the copper recovered from the flue dust and siliceous "carbonate" ore capping is approximately 8c. per pound.

The above data were supplied by C. B. Lakenan, General Manager, and R. E. H. Pomeroy, Acting Superintendent, of the Steptoe plant.

Coal Mining in Alberta in 1913

CONTRIBUTED BY THE COMMITTEE ON COAL AND COKE

The following tables showing the output of coal during the year 1913 have been secured by R. H. Morris, of Pocatontas, Alberta, Canada, for the use of the Committee through the kindness of John T. Stirling, Chief Mine Inspector of Alberta.

The principal mining districts are, Crow's Nest Pass, Calgary, Lethbridge, and Edmonton, the outputs from which were as follows:

	Tons
Crow's Nest Pass.....	1,849,435
Calgary.....	627,461
Lethbridge.....	966,020
Edmonton.....	863,430
Total.....	4,306,346

Of this quantity there was exported to the United States, principally from the Crow's Nest Pass district, 138,201 tons of 2,000 lb., that being the ton used throughout the Province.

This production consisted of lignite, bituminous coal, anthracite, coking coal and coal used in making briquettes, as follows:

	Tons
"Lignite coal" (lignitic?).....	1,763,225
Bituminous coal.....	2,374,401
Anthracite coal.....	168,720
Coking coal.....	104,012
Coke produced.....	65,167
Briquettes produced.....	130,861

The only district in which coke was made is the Crow's Nest Pass district, and the entire product was consumed in Canada.

The total number of persons employed is as follows:

Above ground.....	2,231
Underground.....	5,837
Total.....	8,068

The per capita production was as follows:

Tons of coal mined per employee.....	533
Tons of coal mined per underground employee.....	737

activity in the installation of new plants in the districts tributary to both the Canadian Northern and Grand Trunk Pacific Railways.

A recent development of the district is the mining of coal by stripping with steam shovels in the neighborhood of Edmonton.

The following table shows the yearly increase in output from 1901 to 1913, inclusive, illustrating the remarkable growth which has attended the opening of these coal fields and the settlement of the country.

Year.	Output in tons for N. W. T. (Alberta and Saskatchewan).	Output in Tons. Alberta.
1901.....	346,649	
1902.....	510,674	
1903.....	622,939	
1904.....	782,931	
1905.....		811,228
1906.....		1,385,000
1907.....		1,834,745
1908.....		1,845,000
1909.....		2,174,329
1910.....		3,036,757
1911.....		1,694,564
1912.....		3,446,349
1913.....		4,306,346

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Basic-Lined Converter Practice at the Old Dominion Plant

BY L. O. HOWARD, GLOBE, ARIZ.

(Salt Lake Meeting, August, 1914)

THE practice of using acid-lined converters at the plant of the Old Dominion Copper Mining & Smelting Co. was discontinued early in January, 1913, and they were replaced by basic-lined converters. The new equipment consists of one electrically operated 12-ft. upright converter stand and two basic-lined shells for same. These shells are provided with twenty-four 1 1/4-in. tuyères. The mouths are 5 ft. in diameter.

Owing to the fact that there is no bedding system or other large storage capacity between the mine and the smelter, the furnaces must handle as fast as produced ore and concentrates, which may vary considerably from day to day, both in tonnage and copper content. Consequently the converter is often crowded to its utmost capacity by the frequent fluctuations in the daily matte supply. From the foregoing, it will be seen that in no way can the converter be favored by running light charges or by transferring to it from other stands in operation. Neither can it regularly be allowed to cool between blows, and only occasionally can it be run at as low a temperature as might be wished. No copper precipitate or other foreign material high in copper is available to help out the tonnage. Repairs and alterations or the removal of defective tuyères must wait until such time as a slight falling off in the matte supply allows the converting operations to ease up. These variable conditions all combine to interfere with the systematic operating of the converter.

The first 12-ft. shell was blown in on Jan. 6, 1913, and ran until June 27, 1913, when it was removed to patch the brick. A record of operations covering this period follows:

Commenced blowing.....	Jan. 6, 1913.
Taken off stand for patching.....	June 27, 1913.
Total blowing time.....	2,689 hr. 40 min.
Total number of blows made.....	664
Average time of blow.....	4 hr. 3 min.
Tons matte charged.....	18,157
Average copper content of matte.....	43.9 per cent.
Average time blowing one ton copper.....	22.5 min.
Total copper made.....	7,250 tons

Average copper per converter hour.....	2.66 tons
Average air used per minute.....	9,000 cu. ft.
Average blast pressure.....	13.3 lb.
Ore fed.....	3,304 tons

Average Slag Analysis

	Per Cent.
Cu.....	2.5
SiO ₂	22.7
Fe.....	52.1
CaO.....	0.8
Al ₂ O ₃	1.6

Average Matte Analysis

	Per Cent.
Cu.....	43.9
Fe.....	28.8
S.....	22.6
Ins.....	0.4

The average air—9,000 cu. ft. delivered by the blowing engine—is calculated from cylinder displacement multiplied by revolutions for the given time, assuming 100 per cent. efficiency for the engine, and is in excess of that used in converting, for, as only one converter is in operation, some air must necessarily be wasted whenever the shell is turned down while skimming, charging, etc.

Between Jan. 6 and Feb. 15, the casting machine was not in commission and copper was poured from the converter directly into molds. This operation took about 1 hr. per blow, and the molten charge cooling in the front of the converter mouth caused it to close up rapidly. To remove the incrustations thus formed, hooks of various sizes and shapes were made, but none proved satisfactory and the opening gradually grew smaller until it measured only 18 by 24 in. Operations were then suspended for 24 hr. and 14 concentric holes parallel to the sides of the shell were burned through the accumulation to a depth of 4 ft. with an electric arc. Using round iron wedges and driving them into these holes by pounding with a heavy weight suspended from the crane, it was possible to break up the mass and enlarge the opening to its original size. The cause had not been overcome, however, and this condition was a constant source of annoyance until the collar puller shown in Fig. 1 was designed. By its use the opening in the mouth of the converter can be kept at any desired size. It works rapidly and very effectively, easily cutting through hot incrustations 3 ft. in thickness. Unlike a hook, it does not tend to suddenly pull loose, and therefore if the shell has accidentally been lifted off the rolls in the efforts to remove the collar there is less chance of its dropping and breaking the pillow blocks. In use, this machine, suspended from the crane, is lowered into the converter and the cutting tool brought up against that part of the collar to be removed. On lifting, the "tail" comes up against the opposite side of the shell, and not only prevents the tool from pulling loose but pushes the cutter into the collar. If the incrustation is very heavy, deep parallel channels may be cut that will ultimately weaken it sufficiently to allow

large pieces to be pulled loose. After the opening has been enlarged to the desired size the mouth is coated with mud, which forms a line of weakness that renders subsequent collar pulling less difficult. The use of a casting machine has, of course, made pouring copper into the molds from the converter unnecessary and since this practice has been discontinued the mouth does not build up as in the past.

As no one in the plant had any previous knowledge of basic converting, the mistake was at first made of attempting to blow very large charges. Trouble immediately resulted. Tuyères frequently became blocked and had to be replaced and in so doing the lining was injured,

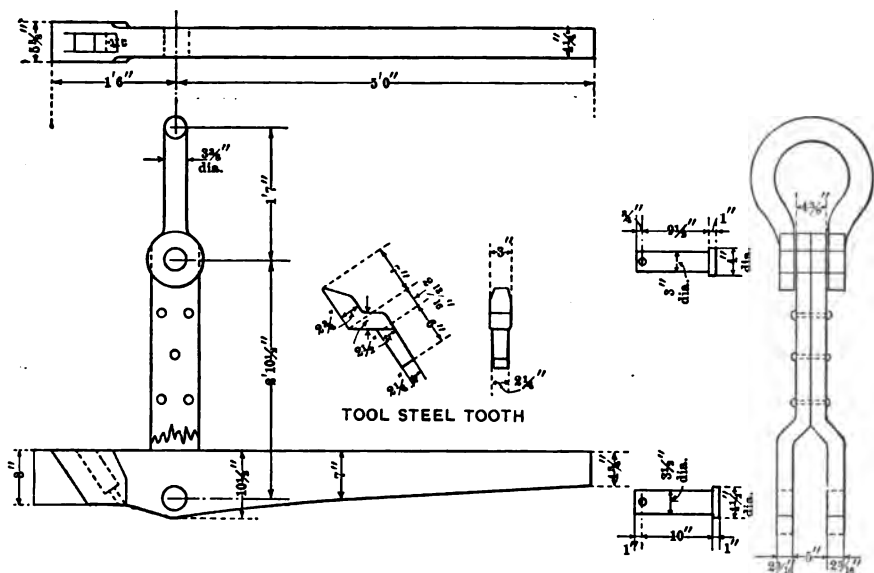
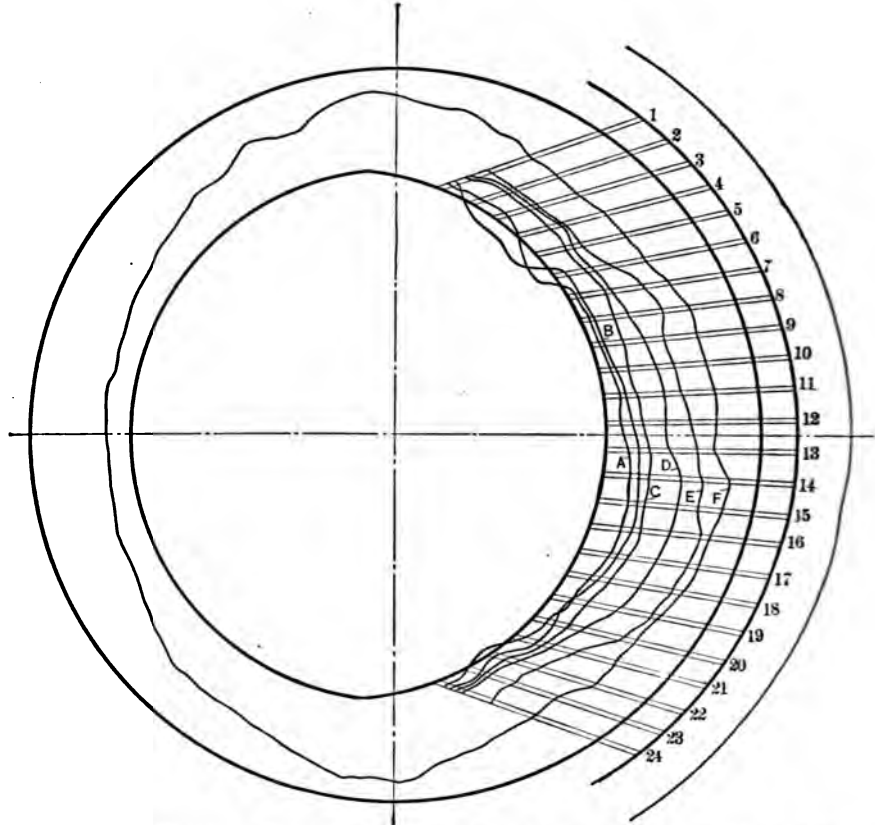


FIG. 1.—COLLAR PULLER.

notably between tuyères 13 and 17. Fig. 2 shows that the wear at this point kept in advance of the remainder of the shell throughout the campaign.

To guard against any possible slagging of the brick by silica, a slag more basic than had been the custom in acid practice was made. On several occasions when the silica had been exceptionally low the tuyères became hard to punch, and on turning the converter down to skim, it was seen that the adjacent brick were coated with some foreign material. Little significance was at first attached to this until in the latter part of January, 1913, it was noticed that this coating had spread and nearly all the brick were covered. Examination showed it to be very high in iron and magnetic, and it at once became evident that under certain conditions magnetic iron oxide was being formed in the converter and deposited



Measurements taken at intervals during life of lining to first patching. Matte average = 43.9 per cent. of copper.

- A. After making 4,000,000 lb. of copper.
- B. After making 6,000,000 lb. of copper.
- C. After making 6,000,000 lb. of copper.
- D. After making 10,000,000 lb. of copper.
- E. After making 12,000,000 lb. of copper.
- F. After making 14,501,962 lb. of copper.

Brick was kept coated with Fe_3O_4 . Measurements A and B were taken with this coating in place. For all subsequent measurements it was, as far as possible, removed. Brick finally gave out at back about 20 in. above tuyères. Shell in operation from January 6 to June 27, 1913.

FIG. 2.—DIAGRAM SHOWING WEAR ON BRICK AT TUYÈRE LINE IN A 12-FT. GREAT FALLS TYPE BASIC-LINED CONVERTER.

on the brick. The thought immediately suggested itself that if this coating could be controlled at will it would, owing to its melting point being considerably higher than matte (about $1,525^{\circ}\text{C.}$), remain on the brick and greatly prolong the life of the lining by protecting it from all chemical and abrasive action. To attain this end the logical method was to blow matte without any silica present, thus oxidizing the iron to Fe_3O_4 , when it was hoped that by later lowering the temperature of the charge by the addition of cold material, the magnetite would be precipitated and cover the interior of the converter. Experiments along these lines proved successful, and by the middle of February the shell could be coated at will. It was quite another matter, however, to keep the coating in place. Hard driving of the converter when the matte supply was large made proper running impossible, and patches of brick periodically became bare and the lining suffered in consequence. Finally after about four months' operating under these conditions, thorough control of the process was attained and bare brick in the interior of the converter became a thing of the past. The lining, however, in the meantime, had received such hard usage that shell No. 1 was taken off the stand to patch the brick and was replaced by shell No. 2.

This second shell, blown in on June 27, 1913, has benefited greatly by knowledge gained in operating the first and it has been possible to keep the magnetite coating intact from the time the shell was first started. After having made over 20,000,000 lb. of copper in less than eight months, much of which was produced under conditions calculated to give the greatest wear to the lining, the brick shows but little sign of the strenuous campaign through which it has passed and the converter appears to be good for many million pounds more before a patching is necessary.

An increased tonnage to the furnaces has thrown more matte to the converter and it has been necessary to drive this shell much harder than the first. Over 50 per cent. of the time it is averaging better than 21 hr. actual blowing time out of the 24, and one charge follows another in rapid succession. Seventy tons of copper have been turned out in 24 hr., and for the period between Jan. 22, and Feb. 20, 1914, the converter averaged 57 tons of copper per day. Such work as this has been made possible only by keeping the brick covered with the magnetite coating and thereby protecting the lining from wear that would otherwise speedily ruin it.

To apply the magnetite covering, no hard and fast rule can be laid down, as the process must necessarily be varied in minor details to suit existing conditions. The following method is one frequently used at this plant: Into the new converter, previously heated by coal and coke to expand the brick lining, a charge of about 12 tons of molten matte is poured and blown for about $\frac{1}{2}$ hr. without any silica. By this time, most of the iron should be oxidized to Fe_3O_4 , but, owing to the high

temperature attained, the magnetite will be quite fluid and not apt to adhere to the brick. The addition of about 4 tons of cold hood cleanings or other material low in sulphur will lower the temperature sufficiently to cause the magnetic oxide of iron to become pasty and deposit on the lining. After this addition, blowing may be continued for a short time to spread the Fe_3O_4 evenly over the interior of the shell. More matte and just enough silica to flux the iron therein contained are added and the charge finished in the usual manner, care being taken to keep the temperature not higher than $1,200^\circ \text{C}$. This may be done by the addition of cold material when the charge begins to show excessive heat. Care must be taken, however, that the coating is Fe_3O_4 , and not cold matte and slag, as a proper foundation on which to build is essential. Should the converter be allowed to get too cool these materials will coat the brick in places, and any Fe_3O_4 that may be applied subsequently will disappear as soon as the temperature rises above the melting point of these more fusible substances. This is a frequent source of failure and the cause of much uneven coating. By repeating the above treatment with each successive blow, a protective covering of any desired thickness can be applied. One 3 or 4 in. in depth should answer all purposes. When too thick, it makes tuyère punching very difficult. Once in place, the permanency of the coating is simply a matter of temperature control. In fact, it is impossible to prevent Fe_3O_4 from depositing on the interior of the converter when the practice is to make a very basic slag and the temperature is kept down, for a charge will occasionally be blown with insufficient silica and Fe_3O_4 formed and deposited on the brick. To cool the charge, an iron ore carrying 53 per cent. Fe, 10 per cent. SiO_2 , and 2 per cent. Cu is frequently used, though cold slag, matte, hood or pit cleanings or scrap copper is equally satisfactory. Cold matte in quantity is not to be recommended unless one has the process well in hand, as the heat of oxidation is later apt to defeat the purpose for which it was added.

An average analysis of the converter slag made during the last eight months is: SiO_2 , 22.7; Fe, 67.6; CaO, 1.0; Al_2O_3 , 2.9 per cent. The matte averaged 43.5 per cent. Cu for the same period. As a matter of interest it may be mentioned that fluid slags have been made in coating the converter that gave the following analyses:

SiO_2 Per cent.	FeO Per cent.	CaO Per cent.	Al_2O_3 Per cent.	Cu Per cent.
4.2	83.2	0.2	0.8	6.0
10.0	81.1	0.3	0.9	4.2
11.0	79.2	0.3	1.1	4.1



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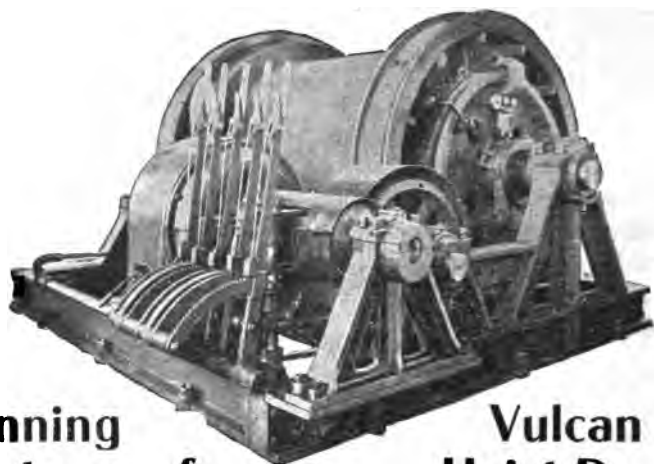
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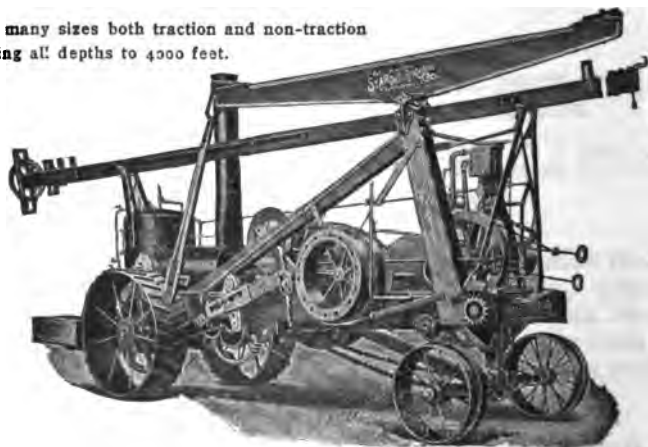
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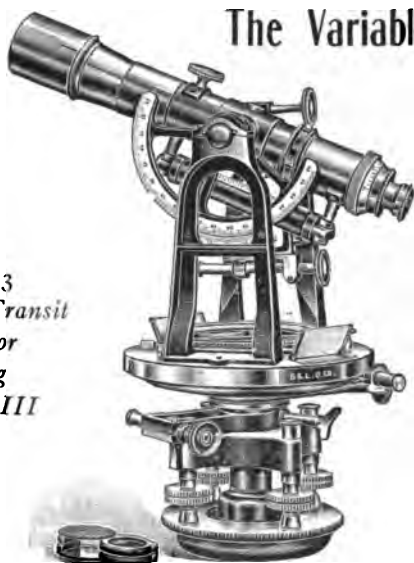
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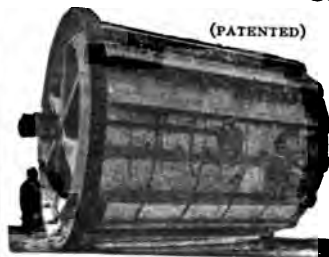
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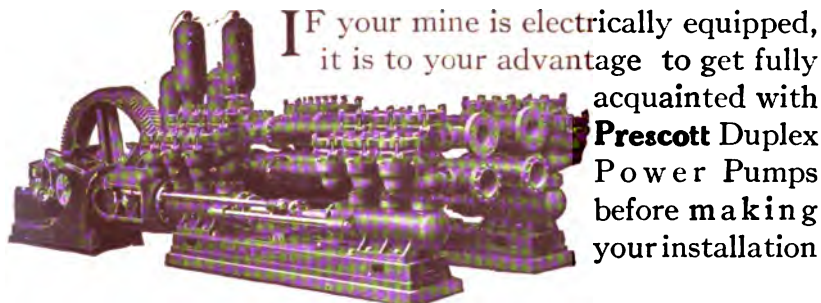
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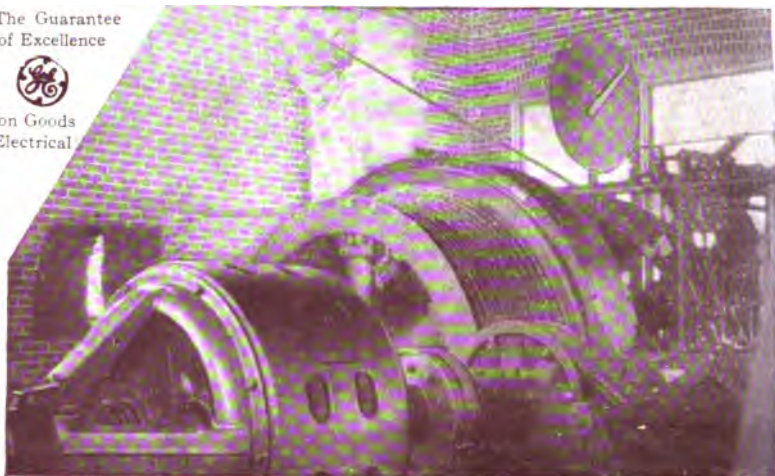
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